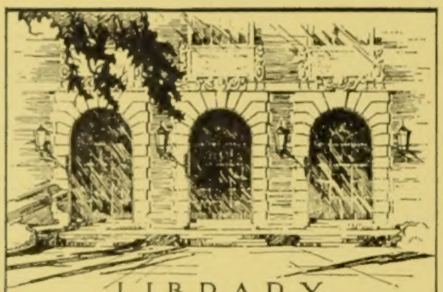

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Ourselves.

Glückauf—the old good-luck salutation of the German miner—we proffer with the compliments of the season to our readers and friends; and it also indicates our hopes for the continuous advance of this journal, under the new name it assumes with the present issue. Started in September, 1902, the purpose has consistently been to make the journal an authoritative exponent of electrochemical science and its various applications; and the steady increasing number of subscribers has proven not only that its pages have appealed strongly to those connected with the electrochemical and electrometallurgical industry and science, but also to the rapidly increasing number of progressive chemical and metallurgical engineers. Thus we found it necessary, almost from the beginning, to pass beyond the bounds of the strictly electrochemical field, and to take account of new chemical and metallurgical developments with which electrochemical processes may have to compete, or which may react in some essential way on that industry. An electrochemist is almost necessarily interested in all things new in the chemical field, and an electrometallurgist in all things new in the metallurgical field. Thus, a glance over the indexes of our first two volumes will show that we could not have served the highest interests of electrochemistry without touching on a host of developments purely chemical or purely metallurgical. What we therefore have done in enlarging the title of this journal has simply been to indicate formally what has already been the working policy of its editor and management in the past.

Any new process can be industrially successful only if products are obtained by its means more cheaply than by any other. The estimate of the value of a new process must, therefore, always be based on a comparison with existing methods, with an allowance for possible and probable improvements in the old methods. To judge and understand the industrial posibilities of an electrochemical or electrometallurgical proposition, it is absolutely necessary to canvass intimately the old chemical and metallurgical arts. The reverse is equally true, for the engineer who has applied purely chemical or metallurgical processes only, must always look out for possible improvements, and the greatest menace to his established methods of working is at present from the application of the electric current to chemical and metallurgical reactions—at least in certain final steps of operation. Our modern metallurgical and chemical plants have already networks of electric lines for lighting and power purposes, so that the application of electrical energy to chemical and metallurgical processes does not assume a revolutionary aspect. This, perhaps, explains why our journal has made friends far beyond the strictly electrochemical and electrometallurgical field and why it has found, apparently, its way to the desk of practically all pro-

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gressive American metallurgists. It is in metallurgy—in special problems where the old metallurgical art has lacked in success—that electrochemical methods will probably find the largest application in the near future. Electrochemistry won its first great industrial laurels in the production of new compounds; a new electrochemical plant meant a new industry, like those of aluminium, calcium carbide, carborundum, artificial graphite, artificial emery, etc. While these industries show a steady and healthy development, important new developments are to be expected in future in certain stages in the production and treatment of ordinary metals—as, in the steel industry—and it becomes more clear from day to day that the progressive worker in these fields will have to be intimately acquainted with the old metallurgical art as well as with all the possibilities of electrical methods. To record carefully and analyze the highest position of the arts of to-day, and to do pioneer work for the art of to-morrow, will in the future, as in the past, be the programme of this journal, so far as its ability extends.

The development due to the introduction of electrochemical methods into metallurgy has, of course, proceeded for some time. The refining of copper is an old and firmly established industry; now we also have the refining of zinc, nickel, lead on a commercial scale. In all these cases it has taken time and experience to learn at what stage it was profitable to introduce electrolysis. Attempts have been repeatedly made with an anode of ore without success, until it was finally learned that the treatment of the copper from the mine up to the production of pure copper is best divided into a series of distinct steps, of which the first ones are all mechanical and metallurgical, and only the last purely electrolytic. One may specialize in one of these several steps, but it is impossible to cope with the entire subject without being familiar with metallurgical as well as with electrochemical methods.

Very instructive in this connection is the development of gold metallurgy in South Africa, characterized, as it has been, by a battle royal between zinc precipitation and electrolytic precipitation. It might be considered enough to discuss in an electrochemical journal simply electrolytic precipitation, but we have held to a broader view of the situation. What is wanted is the best method in any case, and according to the special conditions of the case, one or another method may be preferable. To understand the conditions which favor each method, is the question every gold metallurgist is after; and from this view-point we have covered the subject in the past year in various articles from most authoritative sources. It is quite interesting to refer in this connection to a note of Mr. Charles Butters, the pioneer of electrolytic gold precipitation in South Africa, who now employs in several plants a combined zinc and electrolytic precipitation process. But this example is also instructive in another aspect. We have repeatedly pointed out that zinc precipitation is essentially an electrochemical reaction, due to the great many very small short-circuited galvanic cells in the precipitating vat itself, resulting in the solution of zinc and the precipitation of gold. This also explains the success of the use of "couples" in other processes of zinc precipitation, especially in the case of the zinc-lead couple; the success of Betty's method of adding free cyanide at the head of the precipitation box in the zinc-lead couple process appears quite analogous to the addition of free acid to electrolytic refining or electroplating baths. This shows how knowledge of electrochemical methods and processes may be helpful in the explanation of purely metallurgical processes, and it suggests how this knowledge may assist the metallurgist in his work. In fact, the science of electrochemistry and of physical chemistry in general can be made a powerful tool for metallurgists. To present such knowledge in practical form will be, as heretofore, a special aim of this journal.

A technical journal, to fulfill its functions to the highest degree, must grow with the industry which it endeavors to serve and adapt itself to changing conditions. If the profession lends its aid, if the foremost men of industry and science contribute information to its columns, then, and then only, can the paper be of the highest service to the science and industry represented. For ourselves and our readers, we wish to express our appreciation to those who have contributed in the past to our columns, and with a sincere desire that in the future this journal may continue to merit the good wishes and the collaboration of all those who work in the field of electrochemistry or metallurgy, we renew our greeting, and trust that all of our readers will share in the benefits of the expanding arts in which they and we are all so deeply interested.

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Metallurgy of Zinc.

In this issue we publish three articles on the metallurgy of zinc, two dealing with the production of zinc, the third with the use of zinc for coatings on iron. An able resumé of the present situation of zinc metallurgy is given by Dr. Franz Meyer, who is eminently qualified to speak on this subject, both through his connection with progressive metallurgical developments in this country and his former position as general manager of a German zinc company, which enjoys the reputation of being one of the most progressive and best-managed concerns in the field. Dr. Meyer's lucid exposition of the present situation of the industry and his critical discussion of the tendencies towards improvements are, therefore, based on an intimate acquaintance with the conditions and needs of the present art and should be accepted as the conclusions of a specialist who is both conservative and progressive. While at present it is probably more important to look forward towards improvements of details in the present methods, yet it is most interesting to note the activity along entirely new lines, especially in the attempts of introducing the blast furnace and the electric furnace into zinc metallurgy. Concerning the DeLaval electric zinc process, mentioned by Dr. Meyer, we understand that zinc made by this process has been sold for at least a year in Europe in competition with, and at the same price as, purest Vieille Montagne (Altenberg) zinc; the chief feature of the electric furnace zinc is its high purity, the content of lead being below 0.1 per cent. This is certainly a symptom favorable to the future of the electric furnace in the zinc industry, though it may be doubted whether DeLaval's arc furnace represents really that design of electric furnace which is best adapted for zinc distillation. Mr. Woolsey McA. Johnson contributes to this issue an interesting article on a special problem of zinc distillation, that is the effect which sulphur compounds have on the reaction in the zinc retort. One special remark of Mr. Johnson is worthy of emphasis; he urges that in roasting the ore the object is to obtain an ore which will give the best results in the distillation furnace, not an ore which is especially low in sulphur. The correctness of this maxim is self-evident, yet the fact that a metallurgical process is generally a series of operations succeeding each other and carried out in different departments of the works, has too often caused the superintendent and engineer of one department to specialize in his own field and to lose sight of the whole problem. The best results in metallurgical art require co-operation of all departments, and planning the working field of each department so as to make the total efficiency of the whole works a maximum.

The third paper published on zinc in this issue, is Prof. C. F. Burgess' able article on the properties of zinc coatings. Zinc is very extensively used as a coating on iron for protection from corrosion; the coating is mostly applied by the hot dip and in comparatively rare cases only by electrolysis. An accurate comparison between the properties of the zinc coatings, obtained by these two methods, is the object of Prof. Burgess' paper. The methods of testing the coating are themselves extremely interesting; Prof. Burgess has worked out careful methods of testing the durability against corrosion, the adherence, the effect of heating and cooling, the toughness and strength of coatings, and the resistance to abrasion. The results obtained by these methods are equally interesting. Only with respect to one property—that is, the resistance to the effects of heating and cooling—electrolytic zinc seems inferior to the older form; in this case, the electrolytic zinc coating blisters. Where the material is to be heated—as for pipes leading from heating furnaces—it is therefore preferable to use a zinc coating obtained by the hot dip. On the other hand, with respect to some other properties, the electrolytic zinc coating is decidedly superior to the other form. Especially with regard to durability against corrosion, adherence and toughness, Prof. Burgess' tests reveal a remarkably favorable showing of electrolytic zinc coatings.

The results obtained by Prof. Burgess in careful tests with accurate methods are, indeed, so favorable to electrolytic zinc coatings in many respects that it would seem economical to use such coatings for various purposes to a greater extent that at present, in spite of the higher first cost. The results given in the article will enable one to specify in such cases what dimensions an electro-coating must have to be at its best. Prof. Burgess' article should be read with great attention by zinc platers, since it is suggestive with respect to possible improvements in plating solutions. Finally, in one point his results have a bearing on the theory of electroplating. It is often assumed that the adherence of zinc coatings on iron is due to an alloying; in fact, the formation of an alloy is sometimes considered to be a fundamental condition which must be fulfilled to obtain an adherent electro-coating. Prof. Burgess' adherence tests apparently indicate that no such alloying occurs between the zinc and iron, although the evidence to this effect does not seem conclusive, since the adherence between the two metals would undoubtedly depend on the thickness of the boundary layer of alloy and this thickness might be very small in the special case of a zinc coating on iron.

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Electric Resistance Furnaces.

Besides descriptions of various types of electric furnaces and processes scattered through the volumes of periodical engineering literature, the only authoritative and compresive work on the subject of the electric furnace has been, up to the present, Moissan's classical book. The subject of the resistance furnace which is daily increasing in industrial importance, had never found adequate treatment. This vacancy is now being filled in an admirable way by the series of articles by Mr. FitzGerald, which was started in the September issue of our last volume. Mr. FitzGerald, whose long connection with the two most important electric resistance furnace industries qualifies him to speak authoritatively on the subject, has so far covered in his serial the following subjects. After a discussion of some first principles of electric resistance furnaces in the September issue, he dealt in the November issue with the use of silico-carbides, carborundum and carbon as refractory materials for furnace lining, and discussed in the December issue the materials suitable for resistors, especially granular carbon resistors; in an article in the present issue he discusses miscellaneous accessories, notably means for regulating the voltage and methods of electrode connections.

The term resistor, introduced by Mr. FitzGerald, appears to be a useful addition to our engineering vocabulary. Just as we distinguish between a conductor and its conductance, we should distinguish between the resistor—*i. e.*, the material—and its resistance. It would appear at first sight that the calculation and design of an electric resistance furnace would be an extremely simple problem, since the whole effect is the change of electrical energy into heat according to Joule's law. In view of the simplicity of this law, the problem would appear to be just as simple as, for instance, the calculation of the voltage loss along a transmission line, according to Ohm's law. However, there is one vital difference between the two cases. The resistance of the transmission line is practically constant during operation. But this is by no means the case with the resistor in a furnace, as forcibly brought out by Mr. FitzGerald in his article in our last issue. The resistance of the resistor may change enormously, and this results in the necessity of voltage regulation. In order to use the full available power throughout the furnace run, the maximum and minimum voltages attainable must have to each other the ratio of the square roots of the maximum and minimum resistances of a resistor. Thus, it is absolutely necessary to provide for means of regulating the voltage within considerable limits. Various practical means for this purpose are described by Mr. FitzGerald in his article in our present issue.

American Electrochemical Society.

The next annual meeting of the American Electrochemical Society will be held in Boston, Mass. The date preliminarily fixed is April 25 to 27, but it may be necessary to hold the meeting a week earlier. Prof. W. H. Walker, of the Massachusetts Institute of Technology, is chairman of the local committee. The hotel headquarters will be at the Hotel Lenox, Boyleston and Exeter Streets, Boston.

At the November meeting of the Board of Directors the following gentlemen were elected members of the society: Francis X. Govers, Oswego, N. Y.; Y. Shingo, Tokyo, Japan; J. Sigfrid Edstrom, Vesteras, Sweden; Herman Schlundt, Columbia, Mo.; Geo. G. Grower, Ansonia, Conn.; E. H. Seaman, Wantagh, L. I., N. Y.; Birger Carlson, Mansbo, Avesta, Sweden; Ralph E. Myers, State College, Pa.; William Koehler, Cleveland, Ohio, and Arthur W. Burwell, Lakewood, Ohio.

At the December meeting of the Board of Directors the following gentlemen were elected members: I. Baum, Uniontown, Pa; F. Winteler, Buffalo, N. Y.

At the January meeting the names of the following gentlemen will come up for election: C. J. Thatcher, Brooklyn, N. Y.; J. R. Powell, Galesburg, Ill.; John Nelson, Peru, Ill.; Samuel W. Parr, Urbana, Ill.; Sigmund Saxe, New York; Edward Wray, Madison, Wis.

Electric Steel Furnaces in Europe.

In an editorial on The Canadian Commission Report on European Electric Iron and Steel Furnaces, we remarked that "there is at least one electric furnace process—that of Girod—making special steels on a commercial scale not mentioned in the report." We are informed by Dr. E. Haanel that the French company operating Girod's process makes only ferro-alloys on a large scale and has given up the intention of making steel.

Our remark was based on the fact that the exhibit of the company at the St. Louis exhibition contained besides various ferro-alloys, also tungsten steel and vanadium steel.

Noble Prizes.

The awards of the Noble prizes have now been published. That for physics has been awarded to the Right Hon. Lord Rayleigh, F. R. S., and that for chemistry to Sir William Ramsey. The distribution of the prizes took place on Saturday, Dec. 10, in the great hall of the Academy of Music, Stockholm, in the presence of King Oscar. A banquet followed, at which nearly 200 guests were present. It was stated in the *London Times* that Lord Rayleigh intends to present the value of his prize to Cambridge University. The sum of money attaching to each Noble prize is about $39,000.

News and Notes.

Slimes.—An editorial discussion of the definition of the term slime from the metallurgical standpoint, in the *Engineering and Mining Journal*, of December 22, arrives at the result that slime is an unclassified impalpable mud which it is difficult to decant or to leach.

Worcester Chemical Club.—At a recent meeting of the Worcester (Mass.) Chemical Club, held in the chemical laboratory of the Worcester Polytechnic Institute, Prof. J. H. Perry presented a paper on the chemical composition of rocks, and the importance of this study in showing the manner of their origin. Most interesting material for such a study is to be found in the various granites abounding in Worcester County. The author stated that this study has already led to an entirely new classification of the igneous rocks, based on chemical analysis and not on structure, texture or mineral constituents. B. B. Wright then spoke on the equivalence of the six hydrogen atoms in the benzene molecule. There was also some discussion on the manufacture of carbon tetrachloride and on the purification of water by ozone.

Institute Annual Dinner.—The annual dinner of the American Institute of Electrical Engineers will be given in the ballroom of the Waldorf-Astoria, New York City, on February 8, and promises to be a most interesting occasion. In view of the recent opening of the Subway, and the adoption of electric traction on some parts of the New York Central, the Pennsylvania and the Long Island Railroad, the Institute has decided to devote this dinner to emphasizing the triumph of electric traction. A number of pioneers and leaders will be present, an original menu has been designed, and some novel features will be introduced, while the list of speakers includes men of national and international reputation. The dinner will be served for $5 per cover, without wine or cigars, and, as is usual on these occasions, ladies will be present. The participation of the ladies was a feature that elicited Mr. Carnegie's enthusiastic commendation at the famous Institute Library dinner, which he made forever memorable by his million-dollar gift for the United Engineering Building.

Polytechnic Institue of Brooklyn.—The College of Arts and Engineering of the Polytechnic Institute, of Brooklyn, has arranged evening courses in chemistry and electrical engineering, for which eminent chemists and electrical engineers have been appointed. Dr. Fred W. Atkinson is the president of the Polytechnic Institute, Dr. Samuel Sheldon the department chief and professor of physics and electrical engineering, Dr. J. W. Fay professor of chemistry. The subjects of the courses to be held by the consulting professors are as follows: G. C. Whipple on sanitary and industrial water supply; A. C. Langmuir on shellac, varnishes and glycerine; H. W. Wiley on foods and their adulteration; William McMurtrie on general technical chemistry; C. F. Scott on commercial electrical engineering; F. A. C. Perrine on electrical power transmission; W. S. Barstow on central station practice; T. D. Lockwood on telephony, telegraphy and patent practice; Louis Duncan on electric traction. Admission will be free to all undergraduate students of the Polytechnic, while others interested in the subject may attend for a fee which varies according to the length of the courses.

Phosphate Fertilizers.

An interesting paper on the sources of supply and methods of manufacture of phosphates and potash salts was presented on December 15, before the Franklin Institute of Philadelphia, by **Professor E. B. Voorhees**, director of the New Jersey Agricultural Experiment Station in New Brunswick. The paper was fully illustrated by a great many lantern slides, so that without the aid of illustrations it is somewhat difficult to do justice to the paper in a brief abstract. However, the following summary giving the gist of Professor Voorhees' address should be welcome to our readers, since electrochemical methods have repeatedly been made use of for the production of phosphate fertilizers, (for instance in Palmaer's method, Electrochemical Industry, Vol. II., page 470).

Of the mineral elements required by plants, phosphorus and potassium are of the greatest importance. The term "soil exhaustion" means the exhaustion primarily of these constituents rather than any of the other minerals that plants seed. In the crops of cereals, hay, flax and cotton, grown in this country, there are removed from the soils annually phosphoric acid equivalent to about 7,000,000 tons of superphosphate, containing 14 per cent. of phosphoric acid, and an equivalent of 2,500,000 ton of muriate of potash containing 50 per cent. of actual potash. Phosphates useful for the purpose of the manufacturing of fertilizers are obtained from two sources, organic and mineral; organic phosphates consist chiefly of animal bone; this phosphate has been in use from very early times, though it is only within the last century that it has been used in a rational way. The very good results derived from the use of bone, both raw and dissolved, have encouraged its wide use, and often in preference to the phosphates from mineral sources. Mineral phosphates, first discovered in England in 1845, were called "coprolites." Other deposits have been found since in various part of Europe and other countries, notably the "Bordeaux phosphates" in France; "phosphorites" in Spain; "apatites" in Norway; "phosphatic guanos" in the West and East Indies, and the Islands of the Sea, and deposits of greater or less value in Japan and Algeria; in fact in nearly every country of the globe are found deposits of phosphates useful in the manufacture of fertilizers.

Nevertheless, the most abundant and the richest deposits are found in America, and mainly in the United States. The deposits of apatite in Canada, of purely mineral origin and discovered in 1867, for a time served as a considerable source of supply, but owing to the difficulties in mining and the expense of separating and handling the phosphates, the output, notwithstanding its high quality, has rapidly declined since 1884. The South Carolina deposits near Beaufort, South Carolina, first exploited in 1867, and continuously worked since; the Florida deposits discovered in 1888, and the extensive Tennessee phosphates discovered in 1893, have been rapidly developed, and are now the chief sources of supply of phosphates of the finest quality. These deposits are not only extensive, but are easy to work, particularly those of Tennessee, and especially well adapted for the manufacture of superphosphates.

The various phosphates, both organic and mineral, are in their natural condition insoluble and thus are not in a condition to quickly supply the needs of plants nor to meet all the conditions of cropping. Investigations have revealed, however, that for many crops and under many conditions the organic phosphates as animal bone and the mineral phosphates when finely ground, do serve to supply the phosphoric acid rapidly enough to meet all demands. When such is the case the necessity for further treatment is not apparent, though because these conditions are limited, it is not likely that a very large demand will be made for phosphates in their insoluble form, and the only treatment of which is their fine subdivision.

Very early in the history of their use, therefore, studies were made of methods of treatment which would change the insoluble and slowly available phosphates into more quickly available forms, and it was found by treating phosphate of lime, tri-calcic phosphate, with sulfuric acid, that the reaction resulted in forming a mono-calcic or soluble phosphate and sulphate of lime. This treated product is the superphosphate of commerce. At present, therefore, a large portion of the organic phosphates, and practically all of the mineral phosphates are charged into superphosphates before being applied to the land.

The value of a phosphate when used in its natural state is based chiefly upon its content of phosphoric acid, but the value of a phosphate for the manufacture of a superphosphate depends both upon the amount of phosphoric acid contained in it and the amount and kind of other constituents associated with it. Phosphates which contain considerable amounts of iron and alumina are not well adapted for the manufacture of superphosphates, and no method has yet been perfected which will convert these products into as valuable superphosphates as those derived from phosphates containing minimum amounts of the foreign minerals. Various methods of treatment of phosphates unsuitable for superphosphates have been used which, while not making them as valuable under all conditions as superphosphates, materially increase the availability of the phosphoric acid. Still these phosphates are better adapted for use as phosphates than as constituent parts of a manufactured fertilizer, of which superphosphates or acid phosphate now constitute nearly one-half the total weight.

A prejudice existed for a long time against superphosphates because it was believed that they contained free acid and would thus injure the land. This has now largely passed away, though this feeling, together with the fact that the continual application of superphosphates without proper association with other substances does have a tendency to change the physical character, and to some extent the chemical character of soils, has given rise to the demand for neutral phosphates, those in which there can be no possibility of acid being present. These, however, have not yet been perfected, and the chances are that a broader knowledge of the composition and methods of use of superphosphates will rather encourage their use than increase the demand for products of another sort.

CORRESPONDENCE.

Materials for Resistors=Kryptol.

TO THE EDITOR OF ELECTROCHEMICAL AND METALLURGICAL INDUSTRY:

Sir—Since writing the article on "Materials for Resistors," which appeared in ELECTROCHEMICAL INDUSTRY, December, 1904, a new "incandescent electric material," designed for use as a resistor and called "Kryptol," had been examined, and a note on the subject may be of interest.

The specimen purchased for inspection was found to be in the form of black grains of uniform size. The grains would mostly pass through a screen having 18 meshes to the linear inch and remain on a screen of 24 meshes to the inch. The grains had a dull appearance that did not suggest the presence of graphite; but when a specimen was heated in an open crucible the amorphous carbon present burnt away more rapidly than the graphite and the presence of the latter thus revealed. As it is not possible to determine by mere inspection whether a specimen of carbon is in the graphitic state or not, a sample was heated with a mixture of nitric acid and potassium chlorate which showed clearly that amorphous carbon was present. A sample of "kryptol" was heated in an open crucible till all the carbon was burnt and the residual ash, reddish in color and apparently consisting principally of ferric oxide, was weighed. The ash amounted to 0.24 per cent.

It is not possible from the examination of the "kryptol" to infer with certainty the method of manufacture; but probably it is formed by mixing graphite and some form of amorphous carbon, such as lamp black, with a liquid binder, baking, grinding the resulting material and grading carefully.

The manufacture of this material is founded in a patent granted to Dr. August Voelker* and a critical examination of the specification will serve admirably to bring out clearly some of the misconceptions connected with the use of granular carbon resistors and at the same time to emphasize the principles on which resistors are constructed.

A supposed non-recognition of the obvious fact that in using a granular carbon resistor the resistivity of the mass varies with the size of the grains is the basis of the patent. It is scarcely necessary to point out to anyone familiar with the use of granular carbon as a resistor that the necessity of grading the particles of carbon used in order to meet the various conditions of the work, has been fully recognized for many years. The equally obvious fact that the resistance of the resistor can be varied by a judicious choice in the selection of components having various resistivities, has also been well known for a long time. But Dr. Voelker has apparently discovered a remarkable law connecting volts with the size of carbon granules, so that a "thorough utilization of the electrical energy" may be obtained. The assertion is made that "for currents of two hundred volts and more * * * only grains of from three to seven millimeters can be employed." Five main groups of granular carbon may be formed; for currents of 200 to 300 volts, grains of 3 millimeters; for 300 to 400 volts, grains of 4 millimeters and so on. In short, the law might be expressed, according to Dr. Voelker's figures, as follows. In using granular carbon as a resistor for voltages of

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* United States Patent 770,991; Sept. 27, 1904.

100 *n* — 100 to 100 *n* volts the proper diameter of the carbon grains is *n* millimeters.

Unfortunately for this remarkable law the heating effect of a current is a junction of the product of volts and amperes; but there is nothing said about amperes, size of resistor, heating surface, etc., and that there is any subtle law connecting the "thorough utilization of the electrical energy," the volts and the size of the carbon grains, is difficult to believe.

If it has been found by experiment that in order to heat a certain furnace to a definite temperature a current of E volts and I amperes is required then in order always to obtain the same temperature the following equation must hold:

$$EI = K$$

where K is a constant. If the resistance of the resistor is R ohms, then

$$I = \frac{E}{R}$$

and substituting in (1)

$$\frac{E^2}{R} = K$$

whence

$$R = \frac{E^2}{K}$$

This means, under the conditions described, that the resistance of the resistor must be varied as the square of the voltage. This condition must be fulfilled, and this cannot be done so simply as Dr. Voelker's patent would assert. The study of graular carbon resistors shows the number of factors which determine the resultant resistance.

The specification also errs in omitting the consideration of the effect of temperature on the resistance material. If silicates are present, then at high temperatures carbides will be formed and the resistance of the mass will be changed. At the highest temperatures there will be a marked permanent change in the resistivity of the amorphous carbon, thus causing a change in the resistance of the mass which will reach a maximum when the carbon is converted into graphite.

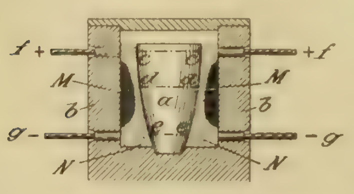

ILLUSTRATION FROM DR. VOELKER'S PATENT.

(M is a "material so compounded that it does not allow the current to pass at all, or only a very limited portion of it," while N is a granular material which conducts the current.)

Finally, in the drawing which accompanies the patent, wrong impressions are conveyed. It is correctly asserted that the maximum generation of heat should occur at *cc*; but as shown in the figure the maximum generation of heat would occur at *d d*, for at this point the cross-section of the resistor is least, consequently the resistance of this section is highest, and the heat generated per second will be greatest.

It is not intended to criticise "kryptol" as a resistor material in this note, nor to criticise adversely the importance of grading granular carbon carefully when using it as a resistor. The object merely is to call attention to the misleading impressions which are conveyed by the patent specification and may cause the inexperienced experimenter much trouble and waste of time.

F. A. J. FitzGerald.

Niagara Falls, N. Y.

Electrolytic Production of Phosphate Fertilizers.

To the Editor of Electrochemical and Metallurgical Industry:

Sir—With respect to the article on my electrolytic process for producing bicalcic phosphate in your issue of November, 1904, page 470, I would like to make the following remarks.

It is said in the article: "The raw material must be sufficiently pulverized, so that no grains of phosphate are embedded in silicates or other insoluble minerals." This is true, of course, but on the other hand, it is quite a merit of the method that in general the raw phosphate does not need to be reduced to a finely powdered state, as is always necessary in the manufacture of superphosphate. It is a fact that raw phosphate has been used in lumps of a diameter of as much as 5 cm.

"The capital needed amounts to about $65.00 per e. h. p. used in the manufacture, etc." (page 471). This sum includes not only the first cost of installation for the plant, but also the working capital; but the cost of installation for the electric power plant is not included.

Wilhelm Palmaer.

Stockholm, Sweden.

Proposed Combination of Massachusetts Institute of Technology with Harvard University.

To the Editor of Electrochemical and Metallurgical Industry:

Sir—In connection with the proposed combination of the Massachusetts Institute of Technology with Harvard University, the following authoritative statement of foreign opinion (translated from *Zeitschrift des Vereines deutscher Ingenieure* of Sept. 24, 1904) is of interest:

"At a meeting of the Union of German Engineers, held at Munich September 12, with the participation of thirty eminent representatives of technological schools and universities, as well as of other schools, and of industries, the following resolutions were adopted:

"1. It is not advisable, so far as can be foreseen, to attempt to meet the need of new technological schools by the addition of technological faculties to universities, but rather by the establishment of independent institutions; for the technological schools would be hindered in their independent development by attaching them to universities. This separation should not, however, impede the welcome development of intellectual good will between the two institutions. The attachment to universities would also in no way involve economies of consequence.

"2. The Union of German Engineers stands now, as before, by its expression of 1886, as follows:

"'We declare that the German engineers have the same needs and will be subjected to the same judgment as to their general culture as the representatives of other professions based on higher scientific education.'

"In this view we rejoice as the conviction more and more gains ground that a considerably greater significance is to be attributed than before to mathematical and natural science as a means of culture. Knowledge of these branches is becoming more and more an indispensable constituent of general education. The predominantly linguistic education now received by the majority of our gymnasium graduates does not satisfy the demands which must be made on the leading classes of our people, in particular, in respect to the increasing significance of economic questions."

Tech Graduate.

Boston, Mass.

[It is questionable if the conditions upon which the German opinion quoted is based, apply to higher educational conditions in this country. Abroad, and in Germany, particularly, a great gap has too long existed between the spirit of the old university and that of the modern technological school. Here, on the contrary, our universities have always been eager to extend their curriculum to meet the requirements of the engineering profession. It would appear that both the Massachusetts Institute of Technology and Harvard University would be benefited by a working arrangement whereby the more purely technical studies would be pursued in the former, and the more purely scientific branches, as well as general culture studies, in the latter.—Ed.]

Review of the Zinc Industry.

By Franz Meyer, Ph. D.

The zinc industry has during the last year kept up its reputation of being the most conservative of all metallurgical arts. No inventions of a revolutionizing character have been made. Improvements of minor importance have been introduced or recommended, and the present state of this industry is briefly as follows:

Ores.

Due to the introduction of magnetic separation, and especially of the Wetherill Process, a great many ores and products of wet separation which, on account of their low zinc contents and their high percentage of impurities, were heretofore not available as zinc ores, are now advantageously used for the manufacture of spelter.

Whether the introduction of the Blake-Morscher electrostatic separator[1] will add to these concentrates remains to be proven. A plant equipped with Blake-Morscher separators, and which was put to work in Wisconsin has been closed again; the reason given for it is the small output in marketable product.

There is, however, a field for an electrostatic separator which will separate rosin blende from pyrites in a raw state, as the magnetic separators do not solve this problem. If the blende is in the form of "black jack," that is to say, if it contains chemically-combined iron, it can be separated from pyrites, without roasting, by the Wetherill process, but if it is in the form of rosin blende, it is not magnetic, and in order to separate it from pyrites the ore must be partly roasted so as to render the pyrites magnetic. This does not only add to the cost of separation, but it is also objectionable from the zinc smelter's standpoint, as a partly roasted blende requires more fuel in roasting than a raw blende, and as the gases from roasting such a material are, on account of their low contents of SO_2, not desirable for acid-making. For these reasons it is to be hoped that the electrostatic separators will soon be so perfected that they can be used commercially on this class of zinc ores.

But even without this, there is no fear that there will be a scarcity of zinc ores in the near future. According to a recent publication of F. H. Bathurst[2], the Barrier mines in Australia, which are famous as lead producers, will soon come to the front as large producers of zinc ores.

The process[3] invented by G. D. Delprat, the general manager of the largest of these enterprises, the Broken Hill Proprietary Company, is said to be an entire success for the concentration of the tailings of these mines. His method of separation consists in subjecting the tailings to the action of a heated aqueous solution of an acid, the density of which has been increased by the addition of a suitable substance. By the action of the acid on the sulphides, hydrogensulphide is developed, which gas carries the ore particles to the surface, whence they are removed, while the gangue falls to the bottom of the spitzkastenlike vat, in which the process is carried out.

As the apparatus is simple, the chemicals used, nitre cake or salt cake and sulphuric acid, are cheap, and as the Barrier Companies have huge heaps of these tailings on the surface, it is to be expected that the introduction of the Delprat process on a large scale will be the source of large quantities of low grade blende for the next years.

Roasting.

While by most of the European zinc smelters the roasting gases are utilized for the manufacture of acid—both sulphuric and anhydrous liquid sulphurous—in this country, by far the largest part of these gases still goes to waste. This difference in the operations is partly due to the fact that in many European countries the law compels the smelters to remove the obnoxious SO_2 from the roasting gases before they are discharged into the air, partly to the unfavorable conditions for the sale of acid in many parts of the United States. Consequently, in this country, reverberatory kilns are mostly used, while in Europe, muffle furnaces are in almost general use.

These furnaces are, with a very few exceptions, rabbled by hand, while on account of the higher cost for labor in the United States the kilns are usually of the mechanically-stirred type. Of the reverberatory furnaces the Brown and the Ropp types are mostly employed, while the only type of mechanically-rabbled muffle furnace which is in use in this country is the Hegeler furnace.

The works which employ the Hegeler furnace, viz.: the Matthiessen & Hegeler Zinc Co., the Illinois Zinc Company, the United Zinc and Chemical Company, the Mineral Point Zinc Company, the Graselli Chemical Company and the United States Zinc Company, utilize the roasting gases for making sulphuric acid, with the only exception of the last named concern, who have no market for acid, but who are, nevertheless, roasting all of their ore in muffle furnaces, on account of the higher losses in zinc if blende is roasted in reverberatory kilns.

The newer designs of mechanically stirred muffle furnaces belong more or less to the MacDougal type. A. R. Meyer[4] and O. Hoffmann[5] propose to muffle the three lower hearths of a MacDougall furnace, 7 hearths high. There is no doubt that these kilns, if properly constructed, will roast blende down as low as the Hegeler furnace does, but the fuel consumption in them will be excessively high, as the passage which the gases of combustion make is too short to utilize their heat to the best advantage.

This loss of heat is avoided by F. Meyer's[6] construction, who intends to heat the two lower hearths of a series of kilns built according to the MacDougall type from one fireplace, common to the whole series. By this arrangement the gases of combustion are allowed to travel the same distance, which is known in the best practice of hand-worked muffle furnaces as the most economical for fuel consumption.

F. J. Falding's[7] mechanically-stirred muffle kiln, which differs from the three furnaces just mentioned regarding the transportation of the ore, was tried by the Mineral Point Zinc Company, but it is not in use any more. The constructions of A. R. Meyer, O. Hoffmann and of F. Meyer have not yet been tested in practice, as far as it is known.

The five concerns who are manufacturing sulphuric acid from blende roasting gases in the United States, use the chamber process with the exception of the Mineral Point Zinc Company, who are working assording to Schroeder's[8] contact process, which was invented and developed at the works of the Actien Gesellschaft fuer Zink Industrie, vormals Wilhelm Grillo, in Oberhausen (Rhine), and the rights of which, in this country, are owned by the New Jersey Zinc Company.

Smelting.

For reducing zinc oxide to spelter it is of great importance that the oxidized ore is intimately mixed with the reducing carbon. This mixing was formerly done by hand; nowadays it is performed by machinery, at least, in the up-to-date plants. In Germany and also in Belgium, the Vapart Mill which has also been installed at the most modern zinc smelter in this country, the United States Zinc Company, in Pueblo, Colorado, is used for this purpose with great success.

Still more important is the use of the very best refractory material obtainable for the construction of the furnace, and especially of the retorts.

An ideal zinc retort should have the following properties.

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[1] U. S. Patents Nos. 668,791-2; Engineering and Mining Journal. Jan. 24, 1903, p. 146; Electrochemical Industry, Vol. I., 1903. p. 255.

[2] Engineering and Mining Journal, of Dec. 1, 1904, p. 871.

[3] U. S. Patents Nos. 763,662 and 768,035.

[4] U. S. Patent No. 750,877.

[5] U. S. Patent No. 768,748; Electrochemical Industry, Vol. II., p. 430.

[6] U. S. Patents Nos. 731,114 and 761,691; Metallurgie, 1904, p. 456.

[7] U. S. Patent No. 756,485.

[8] U. S. Patents Nos. 636,924 and 636,925; Journal of the Society of Chemical Industry, of 1903, page 348; Electrochemical Industry, Vol. II.,, 1904, p. 348.

Its exterior should stand the combined action of an oxidizing flame and of the fluedust of the coal, while its interior should resist the influence of a reducing atmosphere, of zinc and lead vapors, of the fused gangue, especially of ferrous oxide, and not seldom of metallic iron, at a temperature of 1,300° C. At the same time it should be a good heat conductor while it should not absorb any zinc, nor be penetrated by zinc vapors, and last, but not least, it should be available at a reasonable price. Many of these qualities are found in retorts made from silicium carbide (amorphus carborundum[9]) or ciloxicon[10], but the fact that the price of the product is so high, and that the articles made therefrom are attacked by an oxidizing flame at high temperatures, are still preventing their use. It is, however, to be hoped that the latter difficulty will be overcome, and that then the many direct and indirect savings effected by their longer life will warrant their introduction even at their comparatively high cost.

The clay retorts now in general use have been steadily improved, so that in well-managed plants an average life of the retorts of from 6 to 8 weeks is secured. This good result is greatly due to the use of C. **Mehler**'s hydraulic retort press, by which the retorts are made in one operation and in one piece, thus doing away with the seam between the bottom and the cylinder.

The furnaces are either direct fired, or they are heated by natural or producer gas. The direct-fired furnaces, which usually are equipped with 5 rows of retorts of the Belgian type, are constantly replaced by gas-heated furnaces of the Rhenish-Belgian type, which, as a rule, have only 3 rows of retorts of a size between that of the Belgian and the old Silesian retorts. In these furnaces the air, or sometimes both the air and the gas are preheated by either the recuperative or the regenerative system.

As there are rumors[11] that the natural gas in the Iola district will not last much longer, this type of furnace will be, in all probability, also the general one in this country before long, if new gas fields are not discovered.

Of the most recent propositions to reduce zinc ores to metallic zinc in a shaft furnace the following may be mentioned:

John Armstrong[12] has designed a shaft furnace, of which the part above the exit for the zinc vapors consist of two or more compartments separated from each other by vertical partition walls. The center compartment takes the charge, while the lateral chambers are filled with anthracite or coke, so that the charge going down in the shaft is always surrounded by highly heated fuel, which is to reduce the Co_2 formed by the action of CO on ZnO again to CO and thus prevent the vapors of metallic zinc from reoxydizing. It is not known whether this furnace has ever been put into operation, but it is to be feared that its product will be a mixture of zinc dust (blue powder) and of zinc oxyde instead of spelter.

Paul Schmieder[13] proposes the use of a shaft furnace, the upper part of which is externally heated by producer gas, while into its lower part wind is blown. The products, made in the two parts, and which will be chiefly spelter in the upper part and impure zinc oxyde in the lower one, are caught separately. There is no doubt that the residue from this furnace will be lower in zinc than that from the ordinary zinc retort, but as the shaft can only be made of a small diameter, in order to raise the central part of the charge to the temperature of distillation, without injuring the walls between the shaft and the heating chambers by overheating, the furnace will hardly prove to be practicable.

The same must be said about F. **Kellermann**'s[14] smelting furnace. This inventor proposes to treat a charge consisting of the ore, the reducing coal and a flux, without the aid of wind, in a shaft 4 feet x 16 inches x 5 feet high. The external heating of the shaft by means of producer gas necessitates the use of very thin and consequently delicate walls between the shaft and the heating chambers in order to get the heat through them.

W. C. **Wetherill**[15] tries to overcome the difficulty of heating the charge in a shaft furnace uniformly, by placing rows of horizontal heating chambers arranged one above the other in the furnace, thus dividing the charge into several vertical narrow sections. The furnace resembles an ordinary zinc furnace, in which the spaces between the retorts are filled with the charge, while the heating gases pass through the retorts. As the walls of the heating chambers must be thin for the reasons given above, a great drawback in this furnace is the fractability of these chambers, which will probably render the use of this furnace, which otherwise has some very good features, impracticable.

As it will soon be tried at the works of the Prime Western Spelter Company, it is to be expected that the results of this trial will be learned during this year.

On a principle entirely different from those above, E. **Lungwitz**'s[16] process of treating zinc ores in a shaft furnace is based. This inventor recommends to place the shaft under a pressure of about 45 pounds per square inch, in order to recover the zinc in a liquid state in the furnace. The method offers many advantages from a theoretical standpoint; the practical difficulties which will have to be overcome in operating a blast furnace under such high pressure are, however, considerable.

The process will be tried on a large scale during this year, and it will be of interest to all zinc smelters to watch the results of this experiment.

Oskar Nagel, in his United States Patent No. 699,969, proposes a process for producing spelter in a blast furnace which consists in subjecting a suitable zinc ore, preferably mixed with coal, to the action "of water gas which is practically free from inert, indifferent and oxidizing gases, and by which the zinc will be reduced and as an evolved vapor will be condensed in an atmosphere which permits the formation of pure spelter." Nagel explains the failure which all prior attempts to produce spelter in a blast furnace have met with by the presence of these indifferent and oxidizing gases, such as, *e. g.*, nitrogen and carbon-dioxide, which, according to his experience, will in every instance cause the zinc vapors to condense to dust and zinc oxide instead of the liquid spelter.

While in this patent the inventor lays great stress upon the absence of these gases, in his later United States patent[17] No. 766,279, in which he proposes the use of hydrocarbons, and especially of natural gas, containing about 85 per cent. methane and 15 per cent. nitrogen, he states that this mixture is about equal to water gas, and that the nitrogen will do no harm to his process.

Nagel's process is now tried on a large scale by the Actien Gesellschaft fuer Zink Industrie, vormals Wilhelm Grillo in Hamborn, but according to a private communication from the inventor, these experiments have not yet advanced far enough to allow a conclusive opinion on the value of the invention.

In this connection it seems timely to recall a similar process described in Gustaf M. Westman's United States patent No. 383,202, of May 22, 1888. This inventor claims a "process of reducing zinc ores, consisting of subjecting the zinc ores in mixture with coal to the action of highly heated carbonic-oxide, and subsequently reheating and returning the gas through the charge." In the specifications he further states that he avoids the admixture of air or oxygen with the gas, that the solid carbon mixed with the zinc ore acts as the reducing agent, and that the only object of the carbonic oxide gas is to introduce from the outside the heat necessary to the reducing operation, and finally that the gas should be absolutely free from oxidizing matters.

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[9] Benjamin Talbot's U. S. Patent No. 628,288.

[10] E. G. Acheson's U. S. Patent No. 722,793; Electrochemical Industry, Vol. I., p. 287.

[11] W. R. Ingalls, in Metallurgie, 1904, p. 330.

[12] German Patent No. 132,139.

[13] German Patent Application K1. 40a, Sch. 18,544.

[14] Berg-und Huettenmannische Zeitung, 1904, p. 369.

[15] Belgian Patent No. 168,010.

[10] U. S. Patents Nos. 538,785 and 555,961.

[17] Electrochemical Industry, Vol. II., 1904, p. 366.

This process was tried years ago in some place in New Jersey[18], but it is no longer in operation.

Of the many *electric processes* for the manufacture of spelter only two may be mentioned which are in actual operation. C. G. P. de Laval[19] subjects a mixture of preferably finely ground ore and of finely pulverized carbon formed in a pile in a furnace chamber to the action of radiant electric heat upon a sloping surface of said pile, and conducting away and condensing the metallic vapor generated at said surface.

This process which is in operation in Sweden has the advantage that it is carried out in a simple apparatus, and that the electrodes probably have a long life as they come into contact with the metallic vapor only, and not with the charge directly. As, however, only the radiant heat is made use of, it can be worked advantageously only where cheap power is available.

C. Hoepfner's[20] process is in operation at the works of Brunner, Mond & Company, in England, for the manufacture of pure zinc which is guaranteed to be over 99.9 per cent. pure, and which is sold in competition to the Sterling zinc, made from the New Jersey Zinc Company's willemite in retorts. What kind of raw material, whether ores or scrap (hard zinc from the galvanizer?) Brunner, Mond & Company use is not known. A plant which was installed by the late Hoepfner at Fuerfurt-on-the-Lahn, Germany, does not produce spelter any more, but it has been changed into a lithopone factory. At these works it was intended to work the cinders of the zinciferous pyrites from Siegen, which are used by German manufacturers for the production of sulphuric acid. These cinders were subjected to a chloridizing roast, the roasted ore was leached with water, the solution was, after settling, freed from the glaubersalt formed in roasting by crystallization, and from the iron and manganese by precipitation. The remaining pure zinc chloride solution was electrolysed, using revolving annular zinc cathodes on which the pure zinc was deposited in a dense form. The chloride evolved on the anodes was used for the manufacture of bleaching powder.

Zinc Dust (Blue Powder).

This product which is mainly formed in the first period of distillation is a by-product with the European smelters, who use sheet-iron "prolongs" on the condensers to collect it. It contains from 85 to 93 per cent. metallic zinc; the rest is chiefly zinc oxide. As it is used as a strong reducing agent in indigo-dying and in the manufacture of organic products, it is valued according to it contents of metallic zinc.

In this country where the prolongs are not used, it is manufactured purposely and in special furnaces by the New Jersey Zinc Company according to Convers' and DeSaulles process.[21] Same is carried out in an "apparatus comprising a zinc-distilling muffle, a furnace for heating the same externally, and a collecting and expansion chamber immediately adjacent to the muffle and of such relative size and capacity as to be kept heated by the zinc vapors, and to precipitate said vapors therein in their substantial entirety as zinc dust." As the zinc oxide formed at the beginning and at the end of the operation is caught separately in a sheet-iron cone, the blue powder is very high in metallic zinc.

Zinc White.

The bulk of this paint is in this country made directly from ores on the Wetherill grate, franklinite, a product from the magnetic separation of the New Jersey Zinc Company's Franklin Furnace ores, and roasted blende being the chief raw materials.

In Europe this process is not in operation, due to the lack of cheap anthracite coal; all European zinc white being made from metallic zinc, mainly from hard zinc, the by-product of galvanizing iron.

A special paint consisting of a mixture of zinc oxide and lead sulphate is also made in this country according to the Bartlett process by treating refractory ores containing galena, blende and pyrites in such a manner that the zinc-lead paint results, and at the same time the precious metals present are largely saved.

There has also been proposed an electrolytic process for the manufacture of zinc-white by Jacques Oettli[22] during the last year. He subjects zinc plates to the action of electrolysis in a solution of 1 per cent. sodium sulphate at a temperature of approximately 60° C. obtained by means of a current of 10 amperes per square decimeter. Hereby, at the anode zinc sulphate and at the cathode, caustic soda are formed which latter precipitates the former as zinchydroxyde. This is then separated from the solution by filtration, dried and calcined to zincoxyde.

The inventor figures that it will take 74 kw. = 96 electrical horse-power to produce 1 ton of zinc-white by his process. At this figure he is able to compete with the makers of zinc-white who start from spelter, provided that his product has the same good mechanical qualities as that made by burning spelter. Against the United States practice which uses zinc ores as a raw material he can, however, not compete.

Conclusion.

As seen from the foregoing, the zinc industry still offers many opportunities to an inventive mind.

The smelting of refractory ores containing both lead and zinc sulphides in such a manner that without preliminary mechanical separation both metals are recovered is still an unsolved problem. The Lungwitz process, if practicable at all, might form a solution.

But even in working straight zinc ores, the results obtained are far from being satisfactory, as with the best practice known, not over 90 per cent. of the zinc in the ores is recovered. Improvements in this respect are to be expected from the use of a material more refractory than clay for the retorts.

The greatest improvements possible lie, however, in the saving of fuel, the consumption of which is still four pounds of coal per pound of spelter produced, working with ores with 50 per cent. zinc and more, and using regenerative furnaces with gas producers. A central gas plant for the production of both heating and power gas would be a decisive step in the right direction, and at the same time it would result in a saving of labor, which is another heavy item in the manufacture of spelter.

Miscellaneous Accessories of Resistance Furnaces.

By F. A. J. FitzGerald.

It may be said that in general alternating is preferable to direct current in resistance furnaces, for electrolytic effects are avoided and the voltage across the furnace terminals can be more economically regulated. The variation of resistance with the change in temperature in the resistor necessitates some method of varying the voltage if it is desired to use a constant power throughout the working of the furnace. In dealing with small furnaces it is not necessary in many cases to have means for varying the voltage, for, on account of the small size of the furnace it soon reaches its working temperature and there is not much waste of time in attaining the desired point. To take an extreme case: The resistor of an incandescent lamp rapidly reaches its working temperature, and there is consequently no need for voltage regulation. On the other hand, a 1000-hp. carborundum furnace would take a long time to attain its working temperature if no means of regulating the voltage were available.

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[18] Private communication from Dr. R. C. Schuepphaus.

[19] U. S. Patents Nos. 736,611 and 768,054; Electrochemical Industry, Vol. II., p. 423.

[20] Eng. and Mining Journal, May 16, 1903; Electrochemical Industry, Vol. I., 1903, pp. 357 and 570.

[21] U. S. Patents Nos. 727,297 and 727,298.

[22] U. S. Patent No. 771,025; Electrochemical Industry, Vol. II., 1904, p. 465.

Whenever it is possible, some means of regulating the voltage at the furnace terminals should be provided, for it is a very difficult matter to calculate before hand just what the resistance of the resistor will be under working temperatures. If the voltage at the terminals is invariable, then the furnace must be so constructed that when the temperature desired is reached the resistance becomes constant, and this is by no means easy, unless many experiments have been made.

If possible, the regulation should be effected so that the total available power may be used no matter what the voltage is. This can generally be done within limits where an alternating current is used, but the apparatus used for the purpose is somewhat expensive, so that in experimental work it is often desirable to use merely a rheostat. For this method of regulation a water rheostat may be used, but it is believed that a more satisfactory apparatus is found in the granular carbon rheostat. This consists simply in a trough of fire brick filled with granular carbon and connected in series with the furnace. An ammeter is included in the circuit and a voltmeter connected to the furnace terminals, and from the readings of these instruments the power generated in the furnace may be determined. By increasing or diminishing the amount of carbon in the rheostat its resistance may be varied, so that the amount of energy developed per second in the furnace may be regulated as desired. When delicate regulation is required it is not desirable to attempt the removal or addition of carbon; instead, the change in resistance by pressure is made use of. In Fig. 1 a photograph of a granular carbon rheostat is shown. At the left is part of the carbon terminal common to the furnace and the rheostat, while at the right, part of the terminal of the rheostat is shown. The trough is 3 bricks, that is, about 27 inches long, and is partly filled with granular graphitized coke. The three bricks standing on end in the trough are resting on the granular coke and serve in that way for more delicate regulation of the energy developed in the furnace. If the experiments made in the furnace require very delicate regulation of temperature the granular carbon rheostat should be made long, so that the placing of a brick or other weight on the carbon produces but a small change in the resistance of the whole mass. Very delicate regulation may also be obtained by laying pieces of asbestos board, over the rheostat, the asbestos board not touching the granular carbon, but simply resting on the side walls of the trough. By this means the radiation of heat from the rheostat is adjusted, and consequently the resistance regulated with great accuracy.

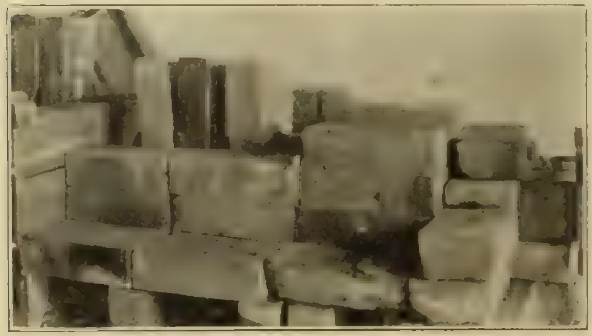

FIG. 1.—GRANULAR CARBON RHEOSTAT

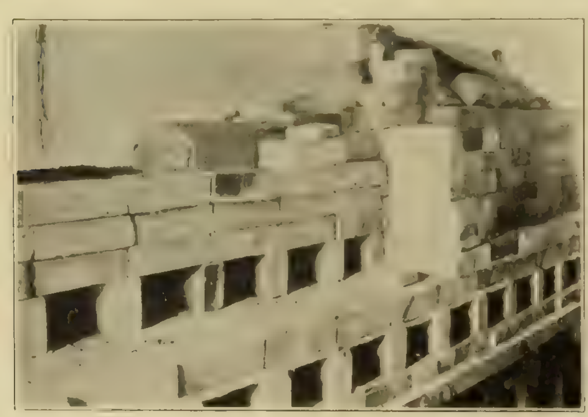

FIG. 2.—LONG RHEOSTAT, USED FOR BAKING.

In Fig. 2 is a view of a long rheostat connected with a furnace, the latter being at the right-hand side of the figure. As a considerable quantity of heat is generated in the rheostat it was found useful in the experiment to use some of this for low temperature baking, hence, the muffle resting on the granular carbon.

Fig. 3 is a view of the furnace of Fig. 2, but taken from the other side. At the right side is the terminal common to the furnace and the rheostat, while at the left side is the cable connected with the other terminal of the furnace. The voltmeter is connected to the common terminal and the furnace terminal, so that the energy generated in the furnace may be determined. The object of arranging the furnace and rheostat with a common terminal is to permit of cutting out the rheostat altogether, thus using the full available power in the furnace.

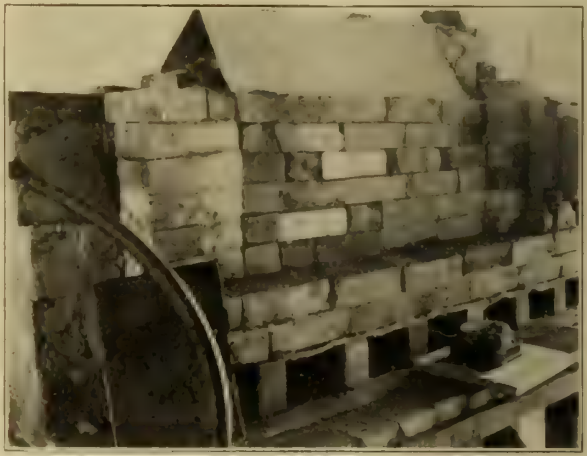

FIG. 3.—RHEOSTAT AND FURNACE

When a furnace does not require the full available power at any stage of the experiment it is better to omit the common terminal and use the arrangement shown in Fig. 4. This shows the foundation of a rheostat. At the left of the figure is the terminal, while to the right are four graphite blocks which serve to connect the granular carbon of the rheostat with the resistor of the furnace. The smaller a furnace is the more difficult it is to avoid losses by radiation and conduction of heat, hence the importance of constructing the apparatus so that these are reduced to a minimum. The heat conducted away by the terminals is a relatively high percentage of the total, hence the advisability of avoiding structures like a common terminal when possible.

It is obvious that in dealing with large furnaces, a rheostat is not a satisfactory apparatus for regulating the voltage at the furnace terminals on account of the energy absorbed by it.

There are two principal methods of regulating the voltage: (1) An ordinary transformer is used to step-down from the high-voltage line, and then beyond that is a regulator which acts as a variable flux transformer, boosting or lowering the voltage on the line going to the furnace; (2) a transformer with a specially constructed primary which permits of varying the number of turns by a switching device, thus varying the voltage of the secondary.

A good example of the first method is found at the works of the International Acheson Graphite Company, where 750-kw. resistance furnaces are used. The current is supplied at 2200 volts and is brought to a 25-cycle, 800-kw. transformer which delivers a secondary current at 140 volts. The regulator is a 6-pole, 560-kw., 25-cycle apparatus with 140 volts primary, and 60-volt, 9350-ampere secondary. By means of the regulator the voltage of the current delivered to the furnaces can be raised or lowered by 60 volts, thus giving either 3740 amperes at 200 volts, or 9350 amperes at 80 volts, or the corresponding current at any intermediate voltage. The following description of the regulator was kindly furnished to the writer by Mr. W. S. Moody of the General Electric Co., who built the apparatus.

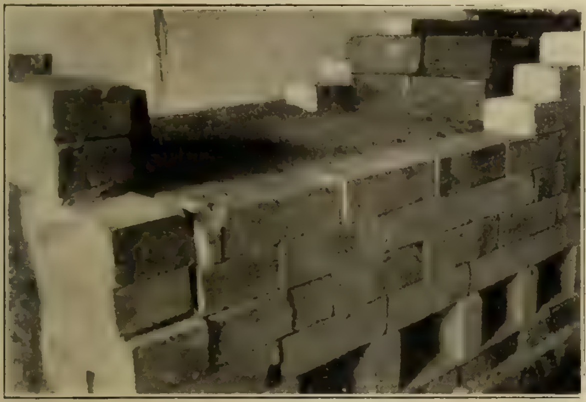

FIG. 4.—ARRANGEMENT OF RHEOSTAT.

The punchings and windings of the regulator are assembled in a substantial oil-tight, cast-iron case having a semi-spherical cover which is made absolutely tight, so as to prevent dust and dirt getting into the gearing and operating mechanism. Both the stationary and movable cores are built up solid of carefully annealed sheet-iron, provided with a large number of slots for the winding. The windings consist of formed copper bars carefully insulated and assembled in these slots. The ends of the windings are brought out by means of flexible cables to a set of vertical bus-bars brought through the center of the cover. In order to avoid an excessive impedance in the device, and obtain as good a power factor on the system as possible, the windings are sub-divided and the bus-bars of both transformer and regulator are arranged so that the alternate bars are of the same polarity. The sub-division of the regulator windings is obtained by connecting all of the 6 poles of the machine in parrallel, and the extremely low impedance so obtained, together with the exceptionally small magnetizing current for which the regulator is designed, gives a power factor quite as good as is obtained with a transformer having a variable ratio.

The moving core of the regulator contains the primary or shunt winding which is connected to the stationary one by means of flexible cables, and the regulator being 6-pole, the total range of voltage is obtained by a rotation of the core of 60°, that is, 30° each side of the neutral point. The series or secondary winding is placed on the stationary core, and in order to avoid excessive impedance in this winding when the regulator is in the neutral position, the movable core is provided with a short-circuited winding placed at right-angles to the active or shunt winding. This winding being at right-angles to the primary has no effect on the latter, except to concentrate the magnetic flux at the poles, and is only in active operation in the neutral position of the regulator, in which position the winding carries the same current as the secondary. In this position the primary or shunt current does not have to carry current, except the magnetizing current, which is quite a small percentage of the normal full load current. Thus, the loss is not appreciably increased by the addition of this extra winding.

The regulator is shown in Fig. 5, the semi-spherical cover having been removed so as to show the cables connecting the windings of the regulator to the interlaced busbars and the mechanism for rotating the primary core.

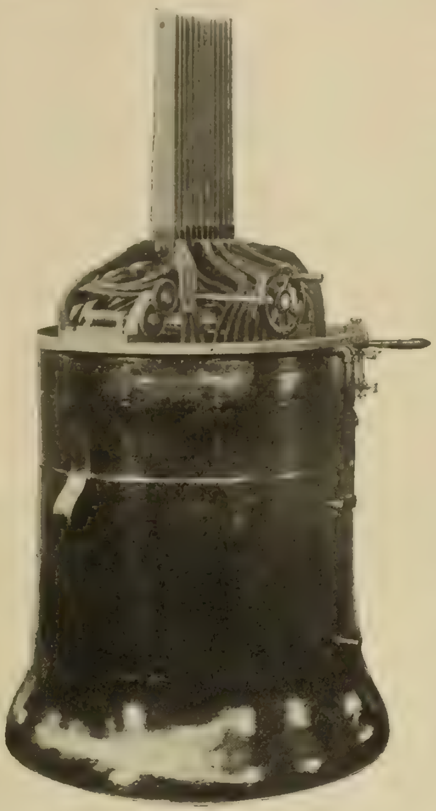

FIG. 5.—REGULATOR.

An example of the latest practice in the second method of regulation is found in a 1600-kw. apparatus built by the Westinghouse Company for the Carborundum Company's furnaces. P. M. Lincoln has kindly furnished photographs and description of the apparatus. This consists of an oil-switch regulator working in connection with the main transformer, and an oil-drum regulator working in connection with an auto transformer. In Fig. 6 there is a general view of the apparatus; at the right is shown the main transformer with the leads going to the oil-switch regulator; at the left the auto transformer connected with the oil-drum regulator. In Fig. 7 the oil-switch regulator is shown with the top removed, discovering the arrangement of cams and levers by which the oil switches are worked. One of the oil switches is shown below with its covering removed. In Fig. 8 the drum regulator is shown removed from its case.

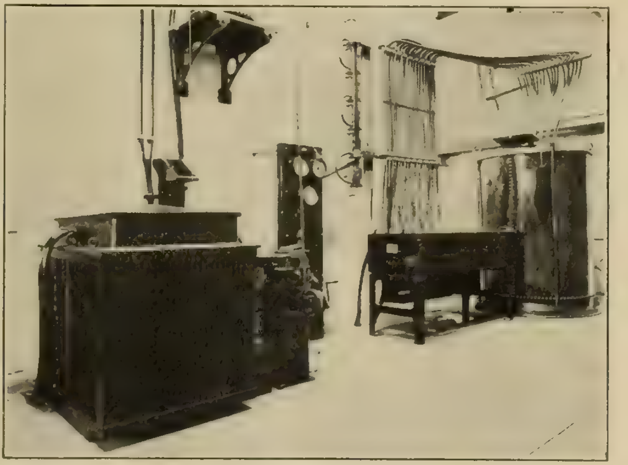

FIG. 6.—GENERAL VIEW OF REGULATION APPARATUS.

Consider first the main transformer and oil-switch regulator. In Fig. 7 it will be observed that there are 10 switches which we shall suppose numbered from left to right. When the handle is turned, one revolution completes one step in the main transformer. Switch No. 1 is closed at the beginning of a revolution of the handle, and then switch No. 3, which connects switch No. 1 with the primary of the transformer. Next, switch No. 2 is closed, throwing in a "preventive resistance," which connects the leads going from switch 1 to switch 3 and switch 2 to switch 4, respectively. Then switch 4 is closed, cutting out the first section in the primary of the main transformer. The "preventive resistance" is shown in Fig. 6, mounted on the wall beside the cables coming from the main transformer. The object of using the "preventive resistance" is to avoid the serious sparking that would occur if the circuit was opened at any time during the revolution of the handle. When the handle of the regulator is in the normal position as shown in Figs. 6 and 7, the resistance is cut out so that no section or sections will be short-circuated permanently. At the next revolution of the handle another section of the primary of the main transformer is cut out, and so on. Thus the oil-switch regulator cuts out eight sections altogether, switches 1 and 2 simply serving to make the changes.

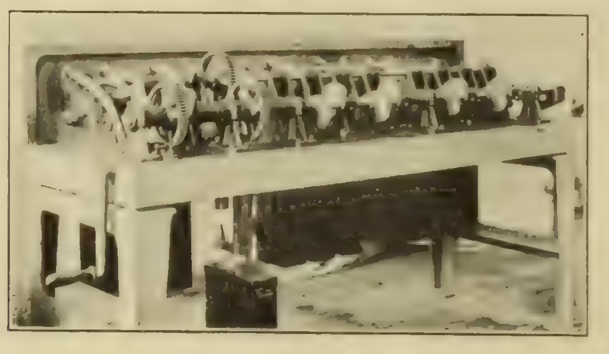

FIG. 7.—OIL SWITCH REGULATOR.

As the change is voltage produced in the secondary of the main transformer by cutting out one of the sections of the primary would be too great for the carborundum furnaces, the auto transformer and series transformer are used to regulate the voltage between the main steps. The sections of the auto transformer are cut out by means of the oil-drum switch shown in Fig. 8. This also is provided with a preventive resistance and the reversing switch, shown in the figure. The reversing switch is provided with an interlocking device which prevents reversing when the main drum is in any but the off position. There are 15 sections in the auto transformer. In this way, by the combination of big steps in the main transformer and a series of small steps obtained with the auto transformer, a large range of voltage by small increments is realized.

In a modified form of the apparatus built for the International Acheson Graphite Company, the oil-switch regulator is worked in connection with an induction regulator on the primary circuit which permits of a continuous regulation of the voltage between steps.

The question as to which of the two is the better is probably largely a question of cost. It is believed that the second method is somewhat the cheaper of the two, and is said to give a slightly higher efficiency. Against this, however, is the complicated system of switches which obviously introduces a large number of possible causes of trouble.

For electric furnaces using currents of relatively low amperage cables may be employed to convey the current from the generator or transformer to the furnace. In all cases the furnace should be as close as possible to the source of energy, for two reasons: First, because of the saving of copper in conductors. Second, because a better power factor is obtained. For furnaces using currents of high amperage, copper bars are used to bring the current close to the furnace, and cables connect the bars with the furnace terminals. Where heavy alternating currents are carried more than a few feet, it is necessary to interlace the bars carrying the current or the power factor of the system will be very low. In the construction of the supports for the bars, closed magnetic circuits must be carefully avoided. Where there is much dust in the furnace room, as often happens, the bars must be inspected at frequent intervals and dust cleared away in order to avoid troublesome short-circuiting with the formation of serious arcs.

The mention of arcs calls to mind a curious experience which may be worth recording, although the conditions necessary for its repetition are not likely to occur. In a certain factory where electric furnaces were used, the transformer was situated in a small building next the furnace room, and the current supplied to the transformer was brought in at 2200 volts to the switchboard, which was provided with marble panels. For a long time no trouble was experienced with the switchboard, till one occasion, when an arc started between two of the high-voltage switches on the board. The arc was finally extinguished, but the cause of its formation was not discovered. Shortly afterwards the same thing happened again, and this time the marble panel of the switchboard was cracked by the heat of the arc and had to be removed. The writer was asked to make an investigation, with a view of discovering the cause of the trouble. It seemed probable that the arc was started by something on the surface of the marble, which destroyed its insulating properties; accordingly the surface of a piece of the marble was washed off with distilled water. Examination of the wash-water showed that it contained a considerable quantity of calcium chloride, which easily explained the formation of the arc. It happened that an appreciable amount of chlorine was given off by the furnaces, and this, in conjunction with moisture had evidently attacked the marble, forming calcium chloride on the surface of the switchboard.

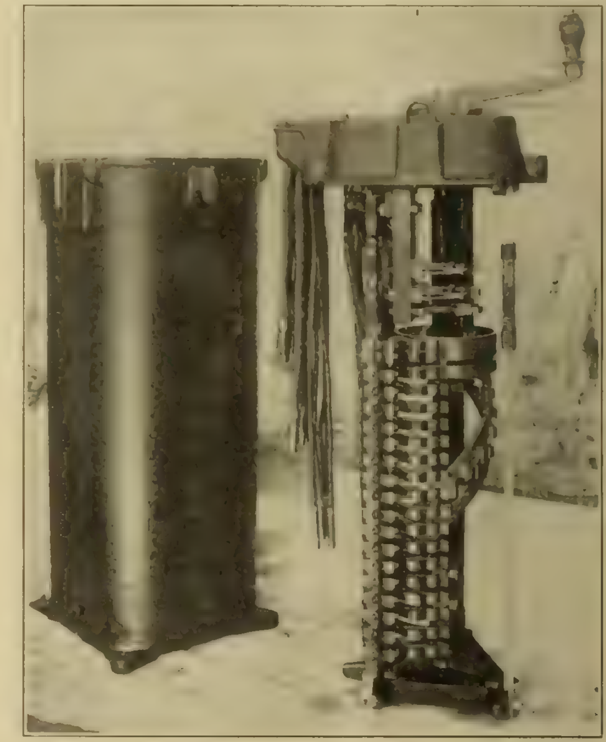

FIG. 8.—DRUM REGULATOR.

One of the most important and troublesome parts of electrical resistance furnaces are the terminals. These are almost invariably built of carbons, with suitable arrangements for connecting the carbons to the cables carrying the current. In special cases, where low temperatures are used, or in small furnaces like those designed for dentists, where the resistor is a platinum wire, metallic terminals may be employed. But in large furnaces using heavy currents, where high temperatures are attained, the only suitable terminal material is carbon. An exception to this may be found in the Gin furnace for the manufacture of steel, which was described by P. McN. Bennie in Electrochemical Industry, Vol. II., No. 1, January, 1904. Here the terminals consist of large blocks of steel which are cooled by water. However, that is an altogether exceptional case.

In America carbons suitable for making the terminals of resistance furnaces are almost invariably made of petroleum coke. The writer only knows of one furnace process where terminals made of other forms of carbon are used, for in most cases objections would be found to carbons containing considerable quantities of ash, and of the cheaper forms of carbon petroleum coke is freer from ash than any other. Indeed, carefully selected petroleum coke, taken from the middle of the retorts so as to avoid all contamination from "scale," is as pure a form of carbon as can be obtained. In Europe, however, terminal carbons made of anthracite coal are used, and in this connection the following extract from a translation by E. Ornstein of a lecture by Julius Zellner will be of interest:

"The recent crisis in the calcium carbide industry has forced the carbide manufacturers to institute economies. Their greatest expense is in the purchase of electrodes, consequently they have done their best to obtain these as cheaply as possible, and have been successful, for in three years the selling price of this product has diminished by 50 per cent.

"The factories producing electrodes have been forced, on account of the cost, to abandon the use of retort carbon and to investigate the treatment of anthracite coal, a much less expensive raw material. The anthracite coal electrodes turned out to-day by all well-equipped factories have a longer life than the retort carbon electrodes made three years ago. While in the early days of the manufacture of calcium carbide, the consumption of electrodes amounted to 10 or 15 kilograms per 100 kilograms of carbide, at present it reaches only to 6 to 10 kilograms at the most, some manufacturers asserting that they consume only 3 per cent.

"The impurities in anthracite coal, at least 3 to 5 per cent of ash, prevent its use in the manufacture of alumina (aluminium?). The electrodes used in this industry are made of petroleum coke, which can be obtained with an ash content of 0.1 per cent.

"So far the employment of charcoal for electrodes has not met with success, for the articles made of this material, which is cheap and low in ash content, have a much greater electrical resistance than those made of other varieties of carbon. Hence, they use too much of the current energy, which precludes their employment"[1]

For much furnace work the presence of the impurities found in anthracite coal would be unobjectionable, but how these carbons would compare with the articles made from petroleum coke when used as terminals, the writer is unable to say, having had no experience with them. It seems probable that one objection to their use might be that they would not allow the use of as high a current density. The writer has made a determination of the real density of a specimen of an anthracite coal electrode and found it to be 1.78, while the real density of a good specimen of a petroleum coke electrode, as made in this country, is about 1.98. In general the electrical conductivity of carbon is greater, the greater the real density, which would indicate that petroleum coke electrodes have the higher electrical conductivity. According to the quotation given above it would appear that the carbon consumption was much less in the case of anthracite coal than in that of retort carbon; but it seems probable that a part of this improvement is due to better design of the furnaces in which the carbons are used.

Another form of carbon for terminals is that produced by the International Acheson Graphite Co. The method of manufacturing these articles is well known; they are made of petroleum coke in the ordinary way, except that a samall amount of some carbide-forming substance is added and are then heated to the highest attainable temperature in an electric furnace. The advantages of these carbons are: Low electrical resistivity, greater resistance to combustion than amorphous carbon articles, ease of machining. The disadvantages are: Relatively high cost, high heat conductivity, up to the present large-sized articles are not manufactured. The last objection refers to length especially, as it is understood that 4 feet is the maximum length attainable, while amorphous carbons can be obtained as long as 6 feet.

In using carbons for resistance furnace terminals the current density that may be allowed varies somewhat according to circumstances; but in general we may use with amorphous carbons a density of 30 to 40 amperes per square inch, and with graphitized articles a density of 80 to 100 amperes per square inch.

The heat conductivity of graphite electrodes is so great that if the temperature of the ends in the furnace is high, the outer end of the carbon will also become so hot that it may be necessary to use some device for cooling, such as a water jacket. In working with small electric furnaces and relatively large graphite terminals, the loss of heat by conduction becomes serious. This heating effect is by no means so marked in the case of amorphous carbon terminals.

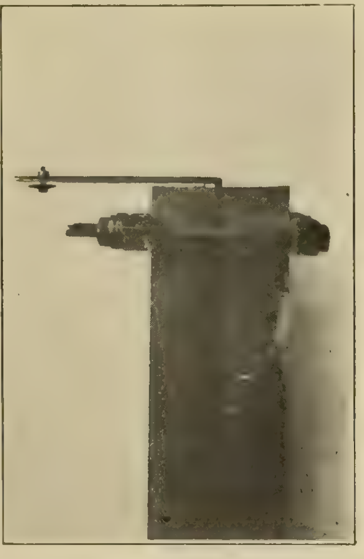

FIG. 9.—ELECTRODE CONNECTION.

The ease with which graphite electrodes can be machined is often of great advantage, for they may be put in a lathe and treated like a metal; drilled, turned, threaded, etc. For a description of this feature the reader is referred to C. L. Collins' article in Electrochemical Industry, Vol. II., No. 7, July, 1904. The machining of amorphous carbon electrodes is by no means so simple, though probably more can be done in that way than is generally supposed.

The problem of making connection between the carbons of the terminal and the cables conveying the current is by no means simple in some cases. In furnaces of the type designed by Acheson for the manufacture of carborundum and graphite the problem is comparatively simple, for the carbons are well protected and last for a long time without renewal. In those furnaces where the carbons are exposed to destructive influences and wear away more or less rapidly, the difficulties met with are great.

A very simple form of connection is shown in Fig. 9. The carbons are each 4 inches square, and between them is clamped a ¼-inch copper plate bent at right angles. The plate is drilled with a hole through which a bolt passes, thus permitting connection to be made with the cable carrying the current. All parts of the clamp should not be made of iron, for in that case a closed magnetic circuit would be formed, which is objectionable, as already pointed out. If the part of the copper plate between the carbons is about 12 inches long, the terminal will have a capacity of at least 1000 amperes, with amorphous carbon and about 2500 amperes with graphite In the latter case, however, if the end of the terminal in the furnace was very hot, it would be necessary to use long electrodes in order to avoid serious heating of the outer end. In Fig. 10 an end view of a terminal built on this principle is shown in place in a furnace.

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[1] Translation from Revue d'Electricité; October 8, 1904; page 49.

Another method of making connections that may be used with graphite electrodes, is to drill out the latter and thread the hole, so that a bolt may be screwed into it. Or the round end of the electrode may be threaded and a metal cap screwed on. The objection to both these arrangements is that the coefficient of expansion of metals suitable for caps or screws is much greater than that of carbon, so that in the case of the bolt there is a possibility of breaking the electrode, and in the case of the cap the contact becomes loose. If the terminal connections are so fitted that they can be sufficiently cooled, these objections are not so serious.

In order to make the best possible contact between the carbon and the metal it is advisable to use graphite between the two. For example, in putting together the terminal shown in Fig. 6, there should be layers of graphite between the copper plate and the carbons. The graphite used for this purpose should be pure, and very soft, so that it will readily compress and adjust itself to all irregularities on the surfaces of the plate or electrodes. A good form of graphite for this purpose is that obtained by the decomposition of silicon carbide at high temperatures.

Where this cannot be obtained a pure grade of Ceylon graphite may be used; but this should first be treated in the following manner: The powdered graphite is put into an iron pot or kettle, such, for example, as the pots used in melting solder, moistened thoroughly with concentrated nitric acid, heated gently at first and then strongly, so as to drive off the last traces of acid. Treated in this way, the graphite swells up greatly and a beautiful soft product is obtained, which is perfectly satisfactory for the object in view.

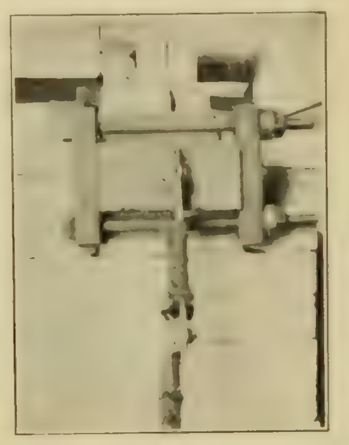

FIG. 10.—END VIEW OF TERMINAL CONNECTION.

In using this material for making connection between the copper plate and the carbon electrodes, as shown in Fig. 6, it should be sprinkled on evenly to a thickness of at least ¼ inch, for when the clamp is tightened the graphite will be compressed to a very thin sheet. A joint between copper and amorphous carbon made in this way will easily carry 10 amperes without serious heating. In applying the nitric acid treatment described above it is important to use Ceylon graphite, because all forms of graphite have not the property of swelling or intumescing when treated in this way.

When a terminal is built up in the manner described, using even amorphous carbon electrodes, the heat conducted from the interior of the furnace may be so great that the outer ends of the carbons are raised to a red heat. It then becomes necessary to protect the carbon from the air, and some modification of the terminal becomes necessary. As the methods used depend upon various circumstances, they will not be discussed here.

Before leaving the subject of the terminal connections, attention should be directed to a very ingenious furnace terminal, patented by Henry Noel Potter. (2) This is designed to overcome the difficulties due to unequal expansion of carbon and metals. Fig. 11 is a section of the terminal. *A* is the carbon of the terminal, *C* is a block of copper or brass, which is held tightly against *A* by the bolts *D*, *D'* passing through *C* and the washer *B*. The principle of the terminal connection is the proportioning of the parts so that the expansion of *A* and *C* by heat just compensate for the expansion of the bolts *D* and *D'*. The copper *C* having a greater coefficient of expansion than the bolts will compensate for the expansion of the latter, thus preserving a constant contact pressure between *A* and *C*.

In using amorphous carbon electrodes for terminals in electric furnaces it is of the first importance that they should be thoroughly baked. Carbon electrodes that are not thoroughly baked have a higher electrical resistivity than well-baked carbons, and are likely to crack when exposed to high temperatures. The fitness of carbon electrodes for furnace work may be tested by determining their electrical resistivity; but such a determination is not easy to make, and requires somewhat expensive apparatus. On the other hand the determination of the density of carbon is comparatively simple, and the desired information can be obtained in that way.

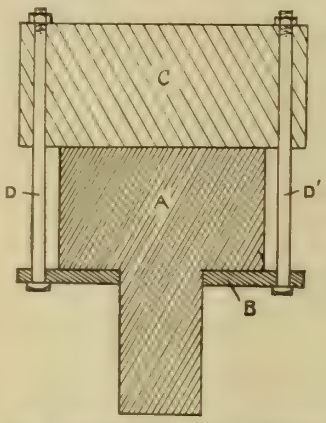

FIG. 11.—COMPENSATED TERMINAL CONNECTION.

A carbon electrode has two densities which may be determined: *Real Density;* that is, the ratio of the weight of the carbon to its *real* volume, and *Apparent Density;* that is, the ratio of the weight of the carbon to its *apparent* volume, which is the sum of the volumes of the carbon and the pores. The higher the temperature at which carbon has been baked, the higher the density, the increase in density increasing continuously till the carbon is converted into graphite. It follows, therefore, that from the real density of the carbon we may form an estimate as to how thoroughly the carbon has been baked. It has been shown elsewhere that the electrical conductivity of carbon increases permanently by heating to high temperatures, hence the importance of baking at high temperatures. But in considering the current-carrying capacity of an electrode the apparent density must also be kept in mind. It is easy to conceive that an electrode might be so porous that although it had been thoroughly baked it would not bear a heavy current density, on account of its porosity.[3]

Notes on the Metallurgy of Sulphur Compounds in the Zinc Retort.

By Woolsey McA. Johnson.

The action of sulphur in a zinc retort is not only interesting from a scientific standpoint, but has a direct practical bearing on the treating of zinc ores in the reduction furnace. A great deal has been written on this subject, and while I have not given the question the rigid experimenting I did the reduction of zinc oxide, nevertheless some account of the experiments I made may prove of interest to zinc metallurgists.

The first point that naturally confronts one is how the sulphur occurs in the roasted ores. How much of the sulphur is bound to the zinc, and how much is bound to the other elements present How much is present as sulphate, and how much is present as sulphide For a fuller discussion of this the reader is referred to Ingall's "Metallurgy of Zinc," chapters on roasting of zinc ores.

However, a few general principles may be laid down here,

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[2] U. S. Patent No. 770,312; September 20, 1904.

[3] For a method of testing carbon electrodes, see Transactions of the American Electrochemical Society, Vol. II., page 43. Electrochemical Industry, Vol. I., page 68.)

although they may be well known to many. To insure a better understanding of some special points of this paper, we will give a short discussion of these general principles.

It may be assumed that in any kind of a properly operated roasting furnace, the percentage of sulphur as zinc sulphate is less than 0.1 per cent. Mr. W. M. Sanders made in our laboratory at Harpe, Kan., several determinations of the decomposition temperature of zinc sulphate. He used for his cell one of the graphite testers we have used in our laboratory.* He found as a mean of several determinations at atmospheric pressure, the temperature of dissociation to be 739 degrees C. This has been confirmed in a private communication by an independent investigator.

As the temperature of the zinc roasting furnace is always 200° C. higher, and sometimes 300° C. higher, than 739° C., it is apparent that the dissociation pressure is always high enough to drive off any SO_3SO_2 and O, provided the ore is rabbled enough in the current of gas. Any amount of sulphate of zinc is thus due to faulty roasting on the part of the men operating the roasting furnace or in the case of a mechanical roasting furnace, due to weak mechanical engineering. Naturally, there is no sulphate of iron present, for the decomposition of iron sulphate is some 100° below that of zinc sulphate.

Lead sulphate is undoubtedly formed, for its decomposition point is high, and with ores containing a large percentage of iron the catalytic effect of the ferric oxide provides for a large percentage of SO_3 in the roast gases.

In many zinc works in the Iola gas belt, reduction is allowed, for sulphur bound to lead, on the supposition that *all* the lead is present as sulphate. This, it is quite certain, is an unwarranted assumption. It was my intention to investigate this thoroughly by chemical analysis, but there were many other things to be done of more importance, and therefore I had no time to give this question the investigation it really deserved. I am convinced, however, that it is extremely important, and that a properly operated hand roasting furnace can control this factor. This will furnish one reason why in certain instances, ores roasted on the hand-roasting furnaces will give much better results in the reduction furnace as regards yield of metal than ores roasted on the machine calciners.

With regard to lime, there is no doubt that it will take up at least 95 per cent of sulphur as sulphate. Chemical analysis, small scale and large scale experimenting by me, as well as by many others before me, have proven this beyond a doubt. Prof. Carl Petraeus, formerly connected with the Lanyon Zinc Company insisted in justice to the men on the roasting kilns who were paid a premium for roasting ores low in sulphur that this fact be taken into account. Consequently, the men are paid on a basis of "Faulty sulphur:" that is, total sulphur less sulphur combined as sulphate of lime. This was a wise procedure, for it is obvious that nobody can do the impossible.

It is also true that below a thousand degrees C. magnesia acts somewhat as does lime. Perhaps, at a high heat, say above 1100° C., it dissociates. At all events, this sulfation undoubtedly occurs to 80 to 90 per cent in the temperatures usually used in roasting zinc ores, namely, 900° to 1075° C.

There remains always from .05 per cent to .50 per cent sulphur, combined with zinc, iron, lead or other similar metals as sulphides. This is due to imperfect roasting, because of lack of proper rabbling, too low a heat, or lack of oxygen in the roasting gases. The question to which of these metals it is combined, is, of course, a most difficult question to decide. That it is already combined with the zinc is impossible to prove.

Assuming, however, that it is combined with the zinc as zinc sulphide, we will discuss the behavior of zinc sulphide in the reduction and distillation process in the retort.

Experiments done by Percy fifty years ago show that zinc sulphide is completely decomposed by *iron* and zinc at a high temperature. When I started to study the reactions used in the metallurgy of zinc I repeated almost all of the experiments given in Percy's metallurgy, sometimes with considerable amplification. We found that iron decomposes zinc sulphide at a temperature of 1167° C. This action is very slow at this temperature, but if the iron is in a molten state, the reaction proceeds with great vigor. We found that 5 per cent of the theoretical quantity of iron would decompose 98.4 per cent of the sulphide of zinc at a temperature slightly above the melting point of cast iron, *i. e.*, 1250° C.

We also found that carbon starts to decompose sulphide of zinc at a temperature of 1200° C., but with the reaction above that temperature, is much slower than is the reaction with iron. A temperature of 1300° C., however, will decompose sulphide of zinc with time.

As there is always a large amount of carbon present in the retort, and in some cases a large amount of iron present and the temperature is usually as high as 1200° to 1300° C., there can be no doubt but that a considerable amount of the zinc sulphide present in the roasted ore is decomposed with the formation of metallic zinc in the eighteen hours of treatment to which the mixture of roasted ore and reducing material is subjected.

The reaction of iron and zinc sulphide is naturally an aid to the condensation of the metal, for it makes a metal vapor undiluted with any neutral gas. The reaction of carbon on zinc sulphide is, however, not so disadvantageous to the condensation of metal, for the reason that in the condenser the reaction reverses to some extent. Thus, we have sulphide of zinc and carbon formed in the condenser. Naturally this is very bad, depositing a kind of a fine soot on the particles of molten zinc and preventing their coalescing. It is quite apparent to one who has watched the zinc furnace when the ore has been improperly roasted and much sulphide of zinc is present in the roasted ore that this reaction occurs at about 11 o'clock in the night when the retorts are subjected to an increasing heat. The flame which shows up at the mouth of the condenser, is of a peculiar greenish color, quite different from the white flame caused by too fast a rate of reduction and imperfect condensation. The formation of iron sulphide as a result of the first reaction is not a good thing, in one way, for iron sulphide is extremely corrosive, and will dissolve fire clay with the formation I believe of a complex sulpho-silicate.

The action of sulphates in the processes is even more complicated than the action of sulphides in the processes. At seven to eight o'clock, at a lower temperature than is the case with the sulphates, these sulphates begin to break up and form also a peculiar white flame of somewhat similar, yet different character than the white flame caused by sulphides. Both these flames are termed by the furnace men "foxy flames." The first "foxy" flame can be prevented, to some extent, by having a large amount of active carbon as reducing agent. There is undoubtedly reduction of these sulphates, but the reaction and "between-reactions" are too complicated and too uncertain to put down on paper. Whether there is reduction from zinc sulphate to the metal with the formation of carbon disulphide and carbon monoxide, or even carbon monosulphide can be neither confirmed or denied. At all events, the facts are known that any amount, say half a per cent of sulphur or sulphates, is bad in causing an imperfect condensation in the retort by a process similar to that described above in the reduction of zinc sulphide by carbon. It is simply a fact that almost all the oxygen and some of the sulphur goes off into volatile compounds. These compounds in the condenser where the atmosphere is at a lower temperature and of not so strongly a reducing nature as in the retort, reoxidize some of the zinc to zinc oxide, and also resulphurize some of the zinc to zinc sulphide. The end result is the same in all cases. The condensation of metal is not as good, more blue powder is formed, and, in certain cases, the losses increased.

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*See Transactions of American Electrochemical Society, April, 1904.

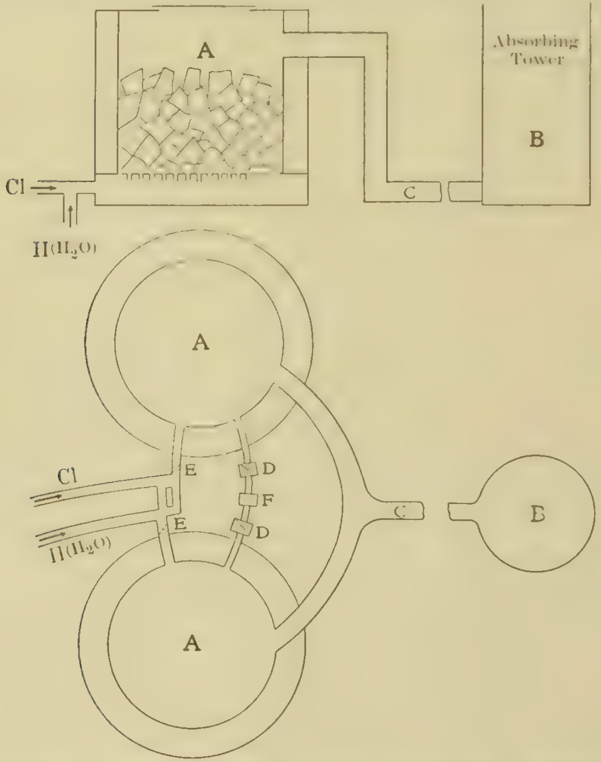
However, a proper mix and proper care to the operation in the distillation furnace will quite prevent this. There is one thing certain, that no one can make the definite statement that "1 per cent of sulphur holds back 2 per cent of zinc." This is the opinion of every smelterman who has given the subject attention. All will agree that sulphur in roasted ores is a bad thing, but how much of the losses amounting to 8 to 25 per cent of the zinc present in the ore is due to sulphur, and how much is due to such as poor coal, improperly mixed ore, poor retorts, careless setting of the condensers, too rapid firing, "poor loom" (which makes bad leaks at the butt of the condenser) or any of the hundred and one factors which contribute to make a low recovery, is a question that passes the capability of human mind. It is hard to diagnose the real causes. There is in the zinc business altogether too much of the fallacy called by logicians "post hoc ergo propter hoc."

One thing is, however, clear to my mind, that properly roasted ore will give much better results in the distillation furnace as regards metal and consumption of retorts than improperly roasted ore, even though sulphur in the first case be as high as one per cent, and the second case as low as 0.2 per cent. The roasting furnaces should be operated, not to produce roasted ore, low in sulphur, but roasted ore that will give the best results in the distillation furnaces. The object of a zinc works is not to show good roasting reports, low in sulphur, but to make zinc.

Utilization of Electrolytic Chlorine.

By Oskar Nagel, Ph. D.

In the September issue, 1904, of this paper, suggestions were made for utilizing electrolytic chlorine, the manufacture of bleaching powder not yielding sufficient profits. Several products were mentioned that could be made by means of chlorine. However, the most important one, for the manufacture of which the largest quantity of chlorine could be used, was omitted, that is, muriatic acid.



PRODUCTION OF HYDROCHLORIC ACID FROM ELECTROLYTIC CHLORINE.

Several experiments were made in Germany some time ago for making muriatic acid by combining hydrogen with chlorine with satisfactory results, but this was at the time when the manufacture of muriactic acid from electrolytic chlorine was absurd, the manufacture of bleaching powder being the more profitable of the two processes. However, since that time conditions have changed and especially in this country the making of muriatic acid is now more profitable than the manufacture of bleaching powder.

The manufacture of muriatic acid from electrolytic chlorine would yield very large profits at the present market conditions, as a much higher price is paid for the chlorine contained in muriatic acid, compared to the chlorine in bleaching powder. Assuming the price for bleaching powder at $1.25, for muriatic acid 22 B with $1.75, we easily see that about 1½ cents more are obtained per pound of chlorine if it is sold as muriatic acid, compared with the chlorine in bleaching powder. If we take the salt used in electrolysis as a basis for this estimate, we find that by making muriatic acid from electrolytic chlorine we are getting about ¾ of a cent more per pound of salt as with bleaching powder.

The reaction $H + Cl = HCl$. is taking place without any difficulty and explosions, if the mixture of these gases is lead through glowing coke. It is to be recommended to have two such vessels filled with coke on hand for using same alternately, as thereby we are enabled to blow the coke hot in one vessel after it is cooled off, and to use meanwhile the other vessel. From these converters, which are filled with coke, the muriatic acid goes by means of a pipe-line, in which it is sufficiently cooled, to the condensing towers, where it is absorbed by water.

Great care must be taken in this process for obtaining a muriatic acid free from chlorine and therefore the following reaction, which works easily and satisfactorily and is always yielding acid free from chlorine, is to be preferred:

$$Cl_2 + 3H_2O + 2C = CO_2 + CO + 2HCl + 2H_2.$$

This reaction is taking place quantitatively at 1,000°C, which temperature is easily obtained by blowing air through coke, analogous to the water gas process. After one minute blowing, chlorine can be made for about seven to eight minutes.

The apparatus to be used for this reaction is practically the same as for the other process.

The gases escaping after the condensation of the muriatic acid can be used for fuel or power purposes.

Muriatic acid made from electrolytic chlorine is far superior to and could command a much higher price than the ordinary acid, the former being free from iron and arsenic.

On the Contents of Carbonate in Electrolytic KOH Solutions and in Solid Caustic Potash.

By F. Winteler, Ph. D.

Since solid caustic potash, as well as 50 per cent solutions are sold according to the content of free KOH, while nothing is paid for the content of carbonate, it is in the interest of the manufacturer to place a product on the market which contains as little carbonate as possible.

The analysis of a whole series of caustic potash products of European electroytic works showed that the content of carbonate often assumes considerable proportions, as will be seen from table 1, the first row representing the number of 19 samples of so-called 100 per cent caustic potash and the next two rows the contents of KOH and K_2CO_3 in per cents by weight respectively.

Table 1.

No.	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
Per cent KOH.	77.0	78.1	76.6	75.2	76.3	73.3	77.1	74.7	76.4	79.5	76.3	80.9	79.7	77.6	78.1	77.4	81.8	77.3	80.8
Per cent K_2CO_3.	11.1	16.6	14.6	16.2	17.2	11.0	12.0	18.4	14.8	6.8	14.8	11.2	11.2	7.8	5.6	4.7	2.4	4.4	3.0

It will be seen that the content of carbonate differed greatly in the various samples and was often more than 20 per cent of the amount of KOH contained in the same.

The caustic potash solution produced by an electrolytic process, using a diaphragm cell or the gravity process ("Glocken" process), is free from carbonate, although dilute and containing potassium chloride as in impurity. The presence of carbonate in the commercial product would, therefore, indicate that the carbonate was introduced during the process of evaporation and casting.

If the vessels which contain the solution before evaporation are not covered, the solution absorbs carbon dioxide from the air. The same occurs in the reservoirs, after coming from the vacuum evaporators. Here the carbonization will go on to a greater extent on account of the higher concentration of the lye (50° B) and of the higher temperature. It is, therefore, in this case of still greater importance to keep the reservoirs closed air-tight. The barrels in which the highly concentrated caustic is heated should be provided with a head-piece as in distilling apparatus. The analyses of Table 1 show that in some European works not sufficient attention is paid to these requirements.

In order to find the maximum quantity of carbon dioxide which a 50° B. caustic potash lye, such as comes from the vacuum evaporators, is able to absorb, I have made the following experiments.

Fifty degrees B. caustic potash was thoroughly shaken, while hot, with solid potassium carbonate for some time, and then set aside for twelve hours and analyzed.

The composition was at a temperature of 22° C. KOH 70.68 per cent by volume, and K_2CO_3 5.53 per cent by volume.

The same lye contained, after five days, at a temperature of 20° C. KOH 71.46 per cent by volume, and K_2CO_3 4.50 per cent by volume.

Another sample of lye of 51° B. contained, after having been shaken hot for some time with K_2CO_3, after forty-eight hours at a temperature of 20°C.: KOH 76.07 per cent by volume, and K_2CO_3 3.73 per cent by volume. After another forty-eight hours KOH 76.3 per cent by volume, K_2CO_3 3.7 per cent by volume.

These figures represent the maximum amount which a concentrated aqueous solution of potassium hydroxide is able to absorb. If cast caustic potash contains a higher percentage of carbonate, this must have been absorbed during the casting process. Exact experiments on the question how the fused caustic can become saturated with carbonate were found very difficult, since temperature and concentration are of very great influence.

However, from the observations of the author, it follows that when caustic potash, strongly saturated with carbonate, is left standing for some time in fused condition, the greatest part of the carbonate settles out at the bottom. If the upper clear portion is then put into iron drums, a pretty satisfactory commercial product of 3 to 5 per cent of carbonate by weight is obtained. The red sediment on the bottom of the pots, which is colored by iron oxide, and consists mainly of carbonate, is then dissolved so as to form a 10 per cent solution of KOH. This solution is then treated with lime in the same way as in the Leblanc process. The caustic potash solution of about 10 per cent, thus obtained, is then passed into the vacuum evaporators, and after having been concentrated, goes to the melting pots.

Corrosion of Metals by Electrolysis.—At the spring meeting, 1903, of the American Electrochemical Society, A. A. Knudson spoke at length on his investigations on the corrosion of water pipes and gas pipes due to stray currents passing from the rails of trolley systems into the earth. (ELECTROCHEMICAL INDUSTRY, Vol. I., page 318.) Mr. Knudson has recently made a study of similar damage in Bayonne, N. J. His results are given in *Eng. News,* November 17.

Investigation of the Properties of Zinc Coatings.

By Prof. Charles F. Burgess

The importance of zinc as an agent for protecting iron and other metals from corrosion is strikingly demonstrated by the fact that over one-half of the zinc produced is used for this purpose. That the efficiency of the zinc coating varies greatly with the purity of the metal, its thickness, continuity and method of application is well known, though the degree to which these various factors affect the efficiency seems to have received but little attention, if one may judge by published data.

Since certain new methods of applying zinc coatings have, to a certain extent, come into competition with the older "galvanizing" processes, the question as to how the protective power of such coatings compares with the older ones becomes of interest and importance.

It was the purpose of an investigation undertaken in the Applied Electrochemistry Laboratory of the University of Wisconsin to determine, if possible, the relative value of zinc deposited by the electrolytic or "cold" processes and the metal as applied by the "hot" or dipping methods. Such determination proved to be somewhat difficult and unsatisfactory, since it involved the devising of methods for determining the properties of zinc coatings. At best, such methods can be only approximations.

While the literature relating to the electrolytic deposition of zinc is extensive, there are but few references to the durability of zinc so deposited as compared with zinc coatings obtained by dipping in molten zinc. The impression is prevalent that electrolytic zinc is much more protective and durable than the other form of zinc. Cowper-Coles makes such assertion in a series of articles on the "Protective Action of Zinc" (*Industries and Iron,* October, 1898), from which the following is an abstract:

"The zinc applied by the hot method usually contains lead, tin and iron. It has been found that iron above 13 per cent makes the zinc too brittle to bend. Lead, up to 1 per cent, is harmless, but above 1.5 per cent, will not dissolve, and the excess collects and forms weak spots. On the other hand, zinc applied by the electric process is very pure, and it is found to resist the corroding action of a solution of copper sulphate very much better than hot galvanized iron. A specimen of hot galvanized iron, having 1.42 ounces per square foot, withstood three one-minute immersions, while a similar piece of electrogalvanized iron having 1.26 ounces per square foot withstood five immersions.

"Results of sulphate of copper tests made on samples of charcoal iron, coated with zinc by different processes:

Process Used to Coat the Iron.	Grains per Square Foot.	Oz. per Square Foot.	Number of One Minute Dips Samples Stood Without Showing Metallic Copper.
Hot galvanizing	648.5	1.48	3
Acid bath, $ZnSO_4$	446.5	1.02	4
Neutral bath, $ZnSO_4$ . . .	552.64	1.26	5

"From this it is seen that steel coated by the electric process will have as long a life as hot galvanized steel, and with much less zinc."

Properties Influencing Value of Zinc Coatings.—The properties upon which the value of a zinc coating depends include the following:

The durability of the zinc and the protection of the underlying metal against corroding influences.

Adherence to the zinc coating.

Toughness and malleability of the coating.

Continuity and density and uniformity in thickness.

Resistance of the zinc to abrasion.

METHODS OF TESTING.

In the determination of these properties for various zinc coatings, recourse could not be had to standard methods of measurement, since no such methods, suitable for the purpose, have been proposed. The methods which were adopted in this investigation are naturally open to criticism, but certain deductions may be derived from them which give approximations to the properties studied.

Durability of Zinc Against Corrosion.—The corrosion of zinc may be defined as a chemical reaction between the metal and the liquid in contact with it. This corrosive agent may be the moisture in the atmosphere, the solutions which impregnate the earth, sea water, or corrosive liquids, such as acids dissolved in water, which may be in contact with it, and the corrosive action may, in the various cases, be considered as differing principally in the rapidity of the chemical action which takes place.

The accurate determination of the life of a zinc coating under practical conditions can be made only after a lapse of months, and for laboratory investigation a more active corroding agent than that afforded by the atmosphere is necessary. The only rapid test which appears to have been applied to zinc-coated iron is the well-known copper sulphate test, which consists in immersing the sample in a saturated copper sulphate solution for periods of one minute or less. The number of such dips which a sample will stand before showing a deposition of copper on the exposed iron, or until all the zinc has been removed, has been taken as an indication of the life of the coating when subjected to corrosion under practical conditions. While this test may be a satisfactory one for roughtly determining the relative thickness of zinc coatings on iron wire or other galvanized ware, it is far from satisfactory as showing the relative values of different qualities of zinc. An objection to the copper sulphate test is that there is no sharp reaction which indicates just when a portion of the underlying iron has been exposed, or the time when the entire coating has been removed.

The question may naturally arise as to whether a rapid corroding test can be applied as a means for determining the durability of a zinc coating when exposed to the much slower corroding influences of the atmosphere, sea water and similar agencies. It is desirable that the rapid method shall approach as nearly as possible to the practical conditions in all respects other than rapidity of corrosion, and copper sulphate seems to be one of the least satisfactory materials which could be chosen as fulfilling this requirement. It acts not only as a strong acid in dissolving the zinc, but the dissolving zinc causes, by galvanic action, a precipitation of copper upon any exposed iron or upon the zinc itself. This deposited copper in turn sets up, with the zinc, a very active galvanic couple, such as is rarely produced under practical conditions, and which causes an abnormally rapid dissolution of the zinc. Copper sulphate is very rarely the corroding influence to which these surfaces will be exposed under practical conditions.

Many tests were made, using the copper-sulphate method, but the only value it seems to have is in being an approximate method for determining the thickness, but not the durability, of a zinc coating. The various kinds of zinc, whether electrolytic or not, or whether attached to iron or separated from it, seem to be corroded by the copper sulphate solution at equal rates. No measurements were made with this solution which will confirm the data previously referred to, as given by Cowper-Coles. The data obtained from tests made with copper sulphate will be given under the discussion of the results of tests.

The action of dilute acids approaches more nearly the practical conditions of corrosion, being similar to the action of impure water upon zinc, and in a certain degree comparable to the action of the various gases and moisture in the atmosphere. Possibly a better test would be one in which the galvanizing is subjected to various gases, such as chlorine, sulphur fumes, and other similar materials, in the presence of moisture, but these tests are naturally very slow, requiring weeks, and sometimes months, to obtain any indications of corrosion. Consequently they are hardly practical for the purposes in hand.

Many trials were made for the purpose of determining the most satisfactory acid to use in the tests of zinc coatings and the most suitable strength of such acid for the production of uniform and comparable results. The data from these tests are not included here, but from them was adopted a solution of 2/3 normal (= 3.2 per cent) sulphuric acid, to be used at room temperatures as the corroding agent to be employed.

The reason for adopting this strength of solution was that the corrosive action was sufficiently rapid to enable a measurement to be made within a reasonable time, not greatly exceeding one hour. It is probable that the more dilute the solution is, the more nearly will it approach practical conditions, but where economy of time is necessary, the stronger solutions must be used. For most practical purposes, where greater rapidity is required, it seems desirable to employ stronger solutions than were used in the tests here described.

The rate at which zinc will dissolve from an iron object depends to a certain extent upon the amount of iron surface which is exposed, simultaneously with the zinc, to the acid. If the iron be completely covered by zinc so that none of it makes contact with the solution, the zinc itself will dissolve very slowly, as shown by the fact that a stick of solid zinc not in electric contact with any other metal, when dipped into sulphuric acid, corrodes at a very slow rate. It would be inaccurate, therefore, to compare a sheet of iron which has been electrolytically galvanized over its surface and edges with a similar sized sample cut from a sheet of galvanized iron. In the latter case the iron is exposed all around its edges, while in the former case it is not, and a corrosion test applied to such sample would show erroneously the advantage to be in favor of the electrogalvanized coating. For better comparison, therefore, in all of the measurements, the samples to be tested were cut so as to expose an edge of the iron all around.

The method employed for conducting the tests was as follows: Thin-sheet galvanized iron was used in most cases. From this, test samples were cut having dimensions 2 inches x 5 inches, and, therefore, a surface area of 20 square inches. Each sample, previous to the test, was scrubbed with vienna lime to remove any dirt or grease which might interfere with the action of the corroding solution. The sample was carefully weighed and then immersed in the standard sulphuric acid solution and left for certain definite periods, after which it was removed, rinsed, brushed, dipped in alcohol, dried, and again weighed. The loss of weight thus obtained gave an indication of the rapidity of corrosion. These dips were repeated until all the zinc had been removed, or until it had been removed sufficiently to determine its relative resistance to corrosion.

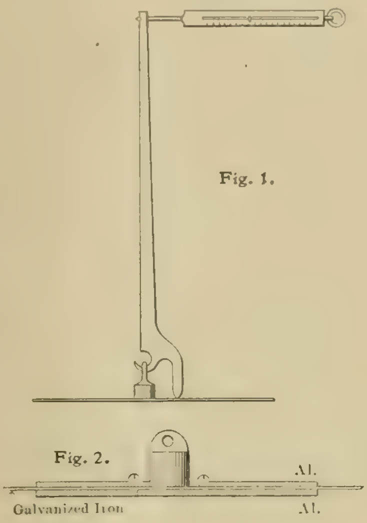
The point at which all the zinc is removed can be quite sharply determined by noting the cessation of the rapid evolution of hydrogen, and also by the very small loss of weight which occurs after the zinc has been completely dissolved. The rate at which the iron itself dissolves in the acid is a very small percentage of the rate at which the zinc in contact with iron dissolves, and the iron can be left in the solution for a considerable time after all the zinc has been removed without losing appreciably in weight. By the weight of the sample before and after the removal of the zinc the amount of zinc which was originally on the surface is readily determined. This can be done with a very small percentage of error, inasmuch as little or none of the iron dissolves while zinc is present, and the process can be stopped as soon as the zinc has been completely removed.

The length of time that a given weight of zinc will last, as shown by such tests, is assumed to indicate, approximately, the durability of the zinc against corrosion in general. Different kinds of zinc and different qualities of coatings show a remarkable variation in durability, as will be pointed out later.

Adherence.

There seems to have been no work previously done in determining the degree of adherence of zinc coatings, and, inasmuch as such determination seemed desirable, a method of testing this property was devised along the following lines:

The method consists in soldering to the zinc surface, by means of a low-melting solder, a copper plug one-half inch in diameter. By noting on a spring balance the pull necessary to separate this plug from the iron, a measurement of the adherence of the zinc to the iron is made. The zinc adheres, by means of the solder, to the brass plug with a greater strength than to the iron. By this apparatus, which is illustrated in Fig. 1, a fair degree of uniformity of measurement was obtained and the relative adherence of different kinds of coatings derived.



Figs. 1 and 2.—Arrangement of Test and Method of Soldering.

An error may possibly come into this test, from the fact that the sample must be heated to a point sufficiently high to cause the solder to adhere. This heating undoubtedly decreases the adherence of an electrolytic zinc coating to the iron, while it does not decrease in like degree the adherence of the coating which has been applied by the hot process. It may also be suggested that a very porous coating will allow the solder to penetrate through the zinc and adhere directly to the iron, thus giving an erroneous value. This was found, however, not to be the case.

There is some tendency for the solder to run out from the edges of the brass plug and thus adhere both to the brass and the surrounding zinc, increasing the force necessary to pull the plug away. This error was eliminated by cutting away such surrounding solder. To avoid that operation, the method of soldering shown in the accompanying sketch (Fig. 2) was adopted. The galvanized iron was held between two aluminium plates, one of which had an opening into which the brass plug fitted. By applying the solder in this opening, and then applying the brass plug, the adherence was effected without the solder flowing on the zinc surface not covered by the plug.

An example of the failure of galvanized coatings is seen on galvanized pipes leading from heating furnaces. The alternate heating and cooling of these pipes frequently causes the zinc to become detached in large scales. To determine whether electrolytic zinc behaves in a similar manner, samples of various kinds of galvanizing were heated and cooled alternately and the effects thus produced observed.

The toughness and strength of the coatings were tested by passing various samples between iron rollers. In some cases the coatings were so brittle as to be cracked off in scales by this treatment, and in other cases the zinc seemed to be pressed into the iron, and to preserve a perfectly continuous coating. By bending such samples at a sharp angle after having been compressed by the rollers, the coatings could in most cases be peeled off. Such coatings were then examined for brittleness, ductility and porosity.

The continuity of the coatings was determined by examination of samples removed in the manner previously mentioned, and also by the use of a microscope.

The method employed for testing the resistance to abrasion consisted in placing samples of galvanizing in a porcelain-lined tumbling barrel, together with quartz sand. By revolving this at slow rate for a number of days, weight determination showed the amount of zinc which was removed per unit area of surface.

Another test consisted in determining the degree to which galvanized iron could be bent before developing cracks in the zinc. This was done by placing a strip of the iron in a device especially constructed for the purpose, by means of which the amount of deflection that is required to produce cracks in the coating can be measured, the cracks being detected by means of a microscope.

Results of Tests

Resistance to Corrosion.—To show the relative values of different zinc coatings as based upon their power to resist the action of corroding influences, twenty samples were prepared and the tests conducted in the manner previously described. Immersion was made for five-minute periods, until the total time of immersion had reached one hour; ten-minute periods for the second hour, and twenty and thirty-minute periods later. The loss of zinc after each dip was determined to a fraction of a milligram. From such measurement the rate of corrosion of zinc at different times was determined, and the results are plotted in accompanying curves, the time in minutes being measured on the horizontal axis, and the rate of corrosion in milligrams per square inch of surface per minute on the vertical axis.

Four determinations were made upon hot galvanizing and sixteen determinations on various thicknesses of electrolytic deposits, and upon deposits obtained from neutral and slightly acidified solutions. The electrolyte from which the deposits were obtained consisted of an ordinary plating solution of zinc sulphate and aluminium sulphate. In the practical operation of such a solution it becomes desirable to add occasionally a small amount of free sulphuric acid to prevent the formation of a precipitate of basic zinc salt, and the tests were designed to show the effect upon the durability of the zinc caused by the presence of this free acid in the plating solution.

It was found that the amount of zinc applied by the hot process is about 25 to 28 grams (one ounce) per square foot, and the electrolytic coatings used ranged from 8 to 34 grams per square foot.

The tests were all run in duplicate; that is, two samples, as nearly alike as possible, were cut from each plate. Where the weights of zinc are alike on the two samples of similarly galvanized iron, the curves of corrosion are quite similar. In some cases, where samples were cut from the same sheet, there is considerable difference in the amounts of zinc, which is due to unequal distribution of current on the cathodes, although care was taken to make the distribution as uniform as possible.

The samples of hot-galvanized iron, which are designated 1, 2, 3, 4, respectively, were obtained from two different dealers, and represented two different grades. One grade had much larger crystals or spangles on the surface than had the other. It is seen from the curves for these samples that the corrosion begins immediately upon immersion, and the rate increases rapidly to a comparatively high value, and then falls off as the iron becomes more exposed.

These tests, toegther with many others not here recorded, show that the hot process zinc was in all cases completely removed in less than one hour's time. The various samples were found to be fairly uniform as to the weight of zinc per unit area, and the rates of corrosion are similarly uniform. These curves may, therefore, be taken as fairly representative of galvanized sheet iron as ordinarily obtained on the market.

It should be noted that the usefulness of the zinc coating must be judged, not by the length of time it takes for the zinc to be completely dissolved, but rather by the length of time for the curve to become maximum. There are two explanations for the fact that the rate of corrosion increases to a maximum, [illegible] On the supposition that the corrosion is due to small galvanic couples set up between the zinc and small impurities imbedded in it or between the zinc and exposed portions of the iron, as the dissolving of the metal proceeds, an increasing number of particles of impurity come to the surface and furnish more galvanic couples and consequently increased corrosion. Another explanation is that the corresion produces a roughness of the surface, and thereby an increased area of surface in actual contact with the acid. This increase cannot go on indefinitely, since after a certain amount of the zinc has been dissolved, its surface decreases, and it is after this point is reached that the iron becomes rapidly exposed.

Table I. gives the methods of preparing and data for the test samples.

TABLE I.

No of Samples.	Wt. of Zn per Sq. Ft.	Amp. per Sq. Ft.	Time of Deposition, Min.	Solution.
1	25 g.	Hot galvanized.	.	......
2	25.6 "			......
3	27.98 "			......
4	27.39 "		..	......
5	8.48 "	14.4	30	Neutral
6	7.87 "	"	30	"
7	9.9 "	"	30	Acid
8	8.1 "	"	30	"
9	17.1 "	"	60	Neutral
10	14.7 "	"	60	"
11	16.7 "	"	60	Acid
12	13.8 "	"	60	"
13	28.38 "	"	90	Neutral
14	23.69 "	"	90	"
15	25.64 "	"	90	Acid
16	21.7 "	"	90	"
17	33.9 "	"	120	Neutral
18	29 "	"	120	"
19	34.14 "	"	120	Acid
20	30.16 "	"	120	"

The curves for corrosion of 5, 6, 7, 8 are somewhat similar to those for the hot galvanizing, though showing a somewhat lower durability, and demonstrating that 8 or 9 grams of electrolytic zinc is not sufficient to be the equivalent of the hot galvanizing coat. The subsequent curves show in a striking manner the advantage of increasing the thickness.

A one-hour coating is seen in curves 9, 10, 11, 12 to be more durable than the hot-process coating, even with much less zinc. Curves 13, 14, 15, 16 show that the one and one-half hour coating corrodes vary slowly for the first two hours, and then increases its rate, though it corrodes at no time with the degree of rapidity of the first four samples. The two-hour coating, as shown in curves 17, 18, 19, 20, corrodes almost imperceptibly for a period of five or six hours, and its protective action is by far the best of all the coatings tested.

A summary of the results of these tests is given in Table II., in which column 1 gives the number of the samples; column 2, the weight in grams of the zinc on the samples; column 3 gives the weight in grams per square foot; column 4, time in minutes which it took for the zinc to be completely removed; column 5 gives the protective action of the zinc, expressed as "minutes per gram per square inch;" column 6, the weight of zinc which should be deposited with a current efficiency of 100 per cent, and, assuming that the current density was exactly 14.4 amperes per square foot, uniformly distributed; column 7 gives results of tests on similar samples in [illegible]te solution.

[illegible] 2 it is seen that it takes one and one-half [illegible] at the current density of 14.4 amperes per square [illegible] e thickness as the [illegible] galvanizing. It will be [illegible] vever, in column 4 [illegible] mersion, while the galvanized varies between 50 and 60 minutes. With 8 or 9 grams of zinc per square foot deposited electrolytically, the time of corrosion is 30 to 90 minutes, showing that with even that small amount of zinc it is possible to get a protective action equivalent to that of three times as much zinc obtained by the hot process. From column 5 may be obtained the best idea of the relative durability of various kinds of zinc coatings expressed in time per gram per square inch. These values were obtained by dividing the length of time it took for the coating to be removed by the amount of zinc on the unit area.

An explanation, which is suggested for the remarkably favorable showing of electrolytic zinc over zinc deposited by the hot dip, is that the hot galvanized has a certain amount of impurity in the form of iron, lead and other materials, and is therefore really an alloy. Such an alloy will dissolve very readily in an acid solution, owing to local action, which is set up between the particles of the different metals. This results in a more rapid corrosion than where a single metal is immersed in a corroding solution, as is the case with the more pure electrolytic zinc. It is well known that if absolutely pure zinc be immersed in a pure acid, little or no corrosion will take place unless contact is made with platinum, iron or some other electronegative element. If electrolytic zinc, deposited upon iron, so completely covers the iron that none of the underlying metal can come in contact with the solution, then the galvanizing will act as a pure piece of zinc. If it is porous, however, there are galvanic couples set up wherever the acid penetrates these pores, resulting in a rapid corrosion of the zinc. This is probably the condition when the zinc coating has insufficient thickness. By increasing the thickness of the coating, these pores are gradually bridged over until there is little or no chance for the corroding agent to get through to the iron, and thus the rate of corrosion of the zinc is greatly reduced.

It will be seen from curves 17 nd 18 that the corrosion of the standard sulphuric acid solution upon zinc deposited to a thickness of 30 grams per square foot was less than one gram in six hours. After that amount had been removed, however, the iron became exposed to such an extent as to cause a more rapid corrosion, and the rate of dissolving increased at a much more rapid rate during the succeeding time.

TABLE II.

Sample.	Wt. of Zn. on 20 Sq. In.	Wt. of Zn. per Sq. Ft.	Time in Minutes.	Time per Gram. Sq. In.	Wt. as Calculated from Current.	No. dips in $CuSO_4$ Before all Zn was Removed.
1	3.4846	25.09	50	.72	....	12
2	3.6950	26.60	60	.86	....	12
3	3.8865	27.98	55	.71	....	12
4	3.8054	27.40	50	.66	....	12
5	1.1786	8.49	35	1.5	1.21	5
6	1.0931	7.87	30	1.35	1.21	5
7	1.3776	9.92	90	3.25	1.21	..
8	1.1251	8.10	45	2.	1.21	.
9	2.3759	17.11	240	5.05	2.42	8
10	2.0747	14.94	200	4.85	2.42	7
11	2.3214	16.71	240	5.2	2.42	..
12	1.9204	13.82	185	4.85	2.42	..
13	3.9438	28.39	570	7.25	3.63	9
14	3.2900	23.69	450	6.85	3.63	8
15	3.5624	25.65	430	6.05	3.63	..
16	3.0139	21.70	220	3.65	3.63	..
17	4.7091	33.90	870	9.25	4.84	12
18	4.0275	28.99	890	11.05	4.84	11
19	4.7424	34.14	740	7.8	4.84	..
20	4.1900	30.17	570	6.8	4.84	..

By a comparison of columns 2 and 6 in Table II., it may be noted that in some cases the actual amount of zinc deposited was very nearly equal to the amount calculated from the current and the electrochemical equivalent. In other cases, the amount was somewhat less. These variations are due to slight errors in the measurement of the current, but more largely to the unequal current density on different parts of the plate, which had received the deposit. Samples 4 and 20 were cut from cathodes 10 inches x 10 inches, which had been plated by surrounding them with a wooden frame to prevent an excessive current flowing to the edges of the plate, and by this means it was attempted to get the current density as uniform as possible. The odd numbered samples were taken from near the edge of the plate, and the even numbered ones, near the middle. It will be seen that the samples nearest the edge of the plate show a greater amount of zinc than do the even numbered ones. An idea of the distribution of the current can be obtained by comparing these values.

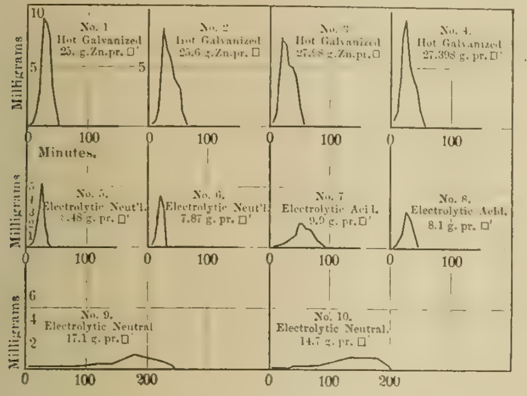


FIG. 3.—RESULTS OF TESTS 1 TO 10.

Tests of Zinc Coating with the Copper Sulphate Solution.—To determine the showing which the copper sulphate test gives for the different kinds of coatings, samples similar to 1 to 4 and 5 to 12 were dipped for one-minute periods in saturated copper sulphate solution. After each immersion the plates were thoroughly scrubbed, dried and weighed to determine the amount of zinc lost. The plotting of this data gives curves of an entirely different form from those obtained with the dilute acid solution, showing that the character of the corrosion is materially different. In Table II., column 7, is given the length of time each sample stood this test. It will be noted that in all of the samples the durability of the coating against copper sulphate was approximately proportional to the amount of zinc in the coating. These tests show only a slight difference between different kinds of zinc coatings of equal thickness, the advantage being slightly in favor of the hot-process product.

It was found also that it makes little or no difference in using this copper sulphate method of testing, whether the zinc is in contact with iron or not. That is, a piece of solid sheet zinc corrodes as rapidly from its surface as though it were in contact with iron. It is believed, therefore, that the sulphuric acid test is one which approaches much more nearly practical conditions, because the ordinary corroding influences to which galvanizing is subjected are such that zinc in contact with iron corrodes more rapidly than does sheet zinc. In fact, it has been demonstrated in practice that solid sheet zinc will last indefinitely, while zinc in contact with iron has a limited life. The only useful purpose that the copper sulphate dip will serve is as a rough method for determining the amount of zinc which a layer contains, the number of dips being proportional to the thickness.

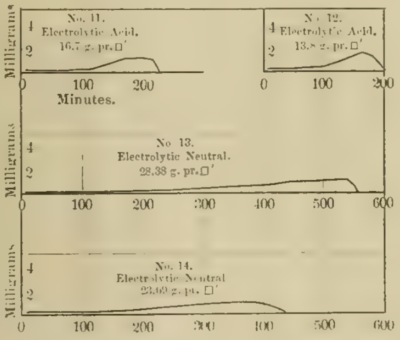


FIG. 4.—RESULTS OF TESTS 11 TO 14.

Samples of corrugated sheet iron which had been galvanized by both the hot and the cold processes were tested for corrosion by the acid solution, and it was found that they were approximately equal in durability, though the electrolytic product had much less zinc than the other. A noticeable defect of the former was that the zinc was not evenly distributed, this being due to the fact that the convex portions of the service received a greater current density than the concave portions, which were further from the anode.

Pieces of the corrugated galvanized iron were subjected to the corroding action of sulphuric acid fumes, such as are liberated directly over a storage battery, and also to chlorine liberated by the electrolysis of sodium chloride. The action of these gases was continued for six weeks. With the sulphuric acid fumes, this length of time was not sufficient to produce any marked results, but with the chlorine, corrosion of the underlying metal took place. This test demonstrated that the electrolytic zinc was fully as durable as the thicker hot-process coating.

Adherence of Zinc Coating.—The tests for the degree of adherence of zinc coatings, as carried out in the manner previously described, show that it takes almost twice as great a force to separate the electrolytic zinc from the iron as it does the hot galvanized. The average of a considerable number of determinations gives the following values:

Zinc deposited from a zinc sulphate plating solution, 482 pounds per square inch.

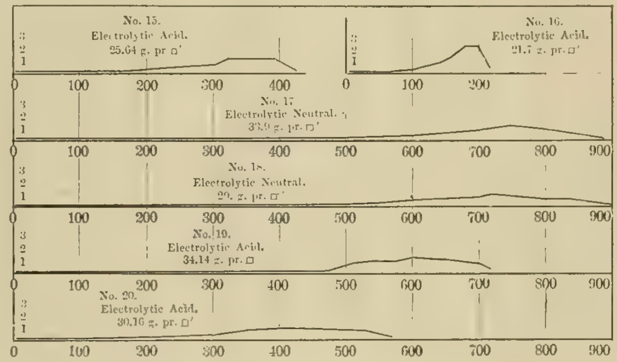


FIG. 5.—RESULTS OF TESTS 15 TO 20.

Galvanized iron obtained from local dealers, 280 pounds per square inch.

Tests were also made on zinc which had been deposited from a cyanide plating solution, and gave values around 230 pounds per square inch.

It was found that a considerable variation in different tests on a single piece of material existed, showing that both the hot and cold galvanizing were not uniform in their degree of adherence over the entire surface. The variation was found to be greater with the hot galvanized than with the cold.

While these measurements of adherence show decidedly to the advantage of the electrolytic process, they were not sufficiently extensive to warrant a definite statement as to relation. The adherence undoubtedly depends upon the care with which the surface has been prepared previous to receiving the zinc. This is especially true as regards the electrolytic deposit.

The Effect of Heating and Cooling.—One of the disadvantages of the galvanized iron is that by rapid heating and cooling, the coating becomes non-adherent and separates from the iron in large flakes. A number of tests were made to determine which of the two methods of galvanizing has the advantage in this respect, but the length of time required to make these tests has prevented this question from being definitely determined. It was observed, however, that electrolytic zinc shows a decided tendency to blister on heating, and that this tendency is not manifested in the hot galvanizing. This blistering seems to be due in part to the unequal expansion of the [illegible] the greater toughness of the electrolytic zinc [illegible] away from the surface, whereas a more [illegible] does not show this tendency. Another cause of blistering is the expansion of enclosed portion of solution. The conclusion may be drawn that where the [illegible] the electrolytic zinc is inferior to the [illegible]

Toughness and Malleability of the Coatings.—It is desirable that the zinc coating have a certain degree of toughness and malleability, especially where galvanized sheet iron is to be bent into various desired shapes. If these properties are lacking, it tends to crack or crumble when subjected to bending stresses. Tests were made by passing samples of galvanized sheets between iron rollers in such a way as to reduce the total thickness by about .01 inches. The effect of this treatment upon the hot galvanized sheet was, in certain cases, to cause it to flake away from the iron, while the electrolytic sample showed no such tendency. By bending the samples back and forth, after having passed them through the rollers, it was found possible to separate the zinc coating from the iron in strips of considerable size. Electrolytic zinc was found to be perfectly uniform and tough, and with a high degree of strength, while the hot galvanizing was very brittle.

The results of tests to determine how much bending a sample will stand before developing cracks in the surface, show that the amount depends to a great degree on the thickness of the coatings, the thin coatings naturally being decidedly superior to the thicker in this respect. No decided difference between hot and cold-process coatings of equal thickness could be detected.

Continuity, Density, and Uniformity of Coating.—The porosity of the coating is a factor determining to a large extent the rapidity of corrosion. Various kinds of galvanizing were examined by a microscope to determine this property, and it was found that the hot galvanizing is very porous, being perforated by minute holes, which are quite apparent by stripping off some of the coating and holding it to the light. It was noted that the lines of contact between the crystals are marked by a large number of perforations.

Electrolytic zinc also has been found to be quite porous when in layers of less than 10 grams per square foot, while coatings of twice that thickness show no perforations.

As is well known, a hot galvanized coating is not uniform in thickness. The crystalline nature shows the manner in which it varies, and there is also a tendency for the zinc to be thicker around the edges, especially in that portion which leaves the bath last, than in other parts of an article. The uniformity, however, is greater with hot galvanizing than with the electrolytic processes, and where irregularly shaped objects are plated, the lack of uniformity on the electrolytic produce is very decided. This is due to a non-uniform distribution of current over the surface of the objects plated, a higher current density existing around the edges than at the center, and at parts of the object nearest the anode.

Resistance to Abrasion.—To determine whether there is any relative advantage as regards the resistance to wear, a number of samples of similar size were placed in a tumbling-barrel with quartz sand. The amount of zinc removed after being subjected to the wearing action for several days showed that there is very little difference in the mechanical wearing quality of hot and cold-process galvanizing.

Conclusions.

The principal fact brought out in these investigations is that with the corroding agent adopted, electrolytic zinc has resistant properties far superior to zinc applied in the molten condition. It may, with some justice, be inferred that a like [illegible] natural corroding influences prevail. [illegible] tion the exact value of this test will remain undetermined for some years.

[illegible] coating shall have to compete [illegible] with the other coating. It is found that for equal durability the electro zinc should be present in an amount equal to about 12 grams per square foot, requiring about 10 ampere-hours for its deposition. To attain the advantage of great durability the electrolytic coat should be as thick as the hot galvanized deposit, or about 28 grams per square foot. This can be produced by 28 ampere-hours or 14 amperes per square foot for two hours.

It is seen that to obtain the best quality of coating the disadvantages of the electrical method are brought out, the principal one of which is the great length of time required to apply the deposit. This can be remedied only by the use of high current densities, and with solutions at present available, it seems impracticable to run the density beyond 18 or 20 amperes per square foot on account of the liberation of hydrogen and undesirable physical quality of the deposit produced. In the tests made, 14.4 amperes per square foot was taken as best representing practical conditions where a cold, unagitated solution is used. With this current density the metal may be deposited with a current efficiency of nearly 100 per cent.

A serious limitation upon the electrolytic process as at present applied is that after the coating has reached a certain thickness there is a tendency for it to become rough and crystalline, and this tendency makes it extremely difficult to apply a coating having more zinc than 30 grams per square foot.

The conclusions regarding electrolytic zinc are based upon data obtained by the use of the one zinc plating solution described. This particular solution was chosen as being extensively used in practice, and tests made on a great many different compositions of bath showed this one to yield the best quality of deposit.

Even with this solution the quality of deposit was found to be sensitive to slight changes within the tank, a somewhat surprising observation being that with a decrease in the amount of anode surface, the appearance and durability of the deposits were improved. The amount and also the distribution of the anode surface have therefore a direct bearing on the quality of deposit.

A great difficulty lies in securing an even distribution of current over the object being plated, as demonstrated on corrugated sheet iron and other irregularly shaped articles. This causes, also, greater thickness and roughness around the edges.

The tests of durability show that the best results are attained with solutions operated as nearly neutral as possible, and that the addition of free acid, which seems desirable for keeping a clear solution, not only decreases the current efficiency, but also materially reduces the efficiency of the zinc as a protection.

There seems, therefore, abundant opportunity for improvement in zinc plating solutions, even though present methods are giving good results industrially. Such improvements may come through the discovery of new compositions for the bath which will enable higher current densities to be used, thicker deposits to be obtained, and greater stability and uniformity of operation acquired. Circulation has been suggested as an improvement, and it is to be expected that the advantages which efficient agitation gives in copper depositions may also be realized with zinc plating vats.

The adherence of zinc coatings to iron is frequently ascribed to an alloying which is supposed to take place. The tests for adherence indicate that no such alloying occurs, either with hot or electrolytic galvanizing, since the tenacity with which the coatings adhere is far less than would be expected if alloying existed.

Laboratory of Applied Electrochemistry,
University of Wisconsin.

Canada.—In a recent consular report we note that the first lead pipe ever made in Canada from Canadian lead is now being produced at Trail by the Canadian Smelting Works; any dimensions required are made up to 4 inches in diameter. It will be remembered that the Betts electrolytic refining process is in use in Trail.

Notes on Electrochemistry and Electrometallurgy in Great Britain.

(From Our Special Correspondent.)

Presidential Addresses and Electrochemistry.

Mr. Robert Kaye Gray's presidential address to the Institution of Electrical Engineers last year was notable for its comprehensive review of the entire electrical situation. In this, of course, the latest developments in electrochemistry and in electrometallurgy were touched upon. Mr. Alexander Siemens' presidential address, delivered on the 10th of November, was wholly of a different order, being in the mature partly of a review of the economic conditions, other than fiscal, which contribute to the success or failure of engineering works, partly of a recitation of views on the metric system diametrically opposed to Mr. Gray's, and a criticism of the latest British patent laws.

Mr. Fletcher Moulton, whose combination of legal talent and technical skill is alike the wonder and envy of many, delivered an interesting and able address before the Society of Chemical Industry. His address was an able resumé of the present outlook of the world's chemical industries. Of the lack of success which has attended many of the attempts to electrolytically produce organic compounds, this "was most disappointing but intelligible, multitudes of reactions were possible, and not easily stopped at the right moment." Mr. Moulton was very hopeful concerning the outlook of electrometallurgy, which is already giving us sodium, phosphorus, calcium carbide, carborundum and graphite. One point may be cited as bearing on the effect of accident upon scientific research. It was mentioned that in one of the greatest of the triumphs of modern chemistry—the synthetic preparation of indigo—an unexpected increase in the yield had been observed after the accidental breakage of a mercury thermometer, which added mercuric sulphate to the reagents present.

At the Civil and Mechancal Engineers' Society, the president, Mr. C. T. A. Hanssen, chose for the subject of his address, "The Effect of Patent Law on Modern Civilization." Throughout the speaker pleaded for the better remuneration and protection of the inventor. Vast benefits have been derived from the work of inventors, who themeselves had shared to a small degree only in the resultant prosperity. A longer duration was suggested in order to repay the heavy expenditure usually incurred in developing and pushing a new enventia. It was also suggested that patents, instead of being issued at a uniform rate of taxation, should be taxed according to the income derived from the monopoly. This would result in a considerable revenue from successful patents, while the charges for undeveloped inventions might be considerably lowered. Most electrochemists would probably welcome changes of this novel and interesting order.

Sir William Abney, at the Society of Arts, dealt mainly with the relations existing between the State and Science. The National Physical Laboratory should be transferred from the Royal Society to a special Government department as is the case in the United States, Germany and France.

Lastly, I must allude to the startling address which Mr. Swinburne delivered to the students of the Institution of Electrical Engineers, entitled "Some Difficulties in Getting On." As reprinted in the *Electrician*, those not present may read five columns of brilliant epigrams and genial chaff of those who devote themselves to scientific research at the expense of developing their business instincts. From the charter of the Institution of Civil Engineers down to the general pursuit of knowledge very few subjects went untouched. For instance, "Science, for which no use has been found, or which is not applied, is called 'Pure Science,' whereas it is really the raw material, and should be called 'Raw' or 'Crude Science' * * * No knowledge is worth obtaining for it own, or any other sake, unless it is, or will probably be, useful to man. * * * A genius does not work for a given employer; he works for the world at large, and the world does not pay him." As a broad rule, Mr. Swinburne would measure the worth of the average man by his earnings. All this is very iconoclastic. Professor Perry and others will have much to say for the other side. It is well, however, that one of Mr. Swinburne's scientific attainments, and practical knowledge should urge the development of the business instinct, although much of what has been said is more suggestive, say, of George Bernard Shaw than, say, of Karl Pearson.

Hypochlorites and Sewage Purification.

The past summer has seen a revival at Guilford of the attempts to sterilize sewage effluents with solutions of hypochlorite of soda electrolytically prepared. This application of the hypochlorites is by no means new. Hermite's plant at Worthing, Woolf's installation at Riker's Island, Danbury, Conn., and at Brewsters, Putnam County, N. Y., were the earliest attempts. The Hermite experiments at Worthing attracted considerable attention. The electrochemical efficiency was, however, low; much trouble was experienced with the insulation of the anodes, while the solutions produced were, according to Roscoe, very unstable. Indeed, a depreciation of 25 per cent of the strength of the available chlorine produced occurred in twenty-four hours. Of Woolf's American demonstrations that at Riker's Island consisted in deodorizing about thirty acres of putrefying garbage. The plant at Danbury, as the writer remembers it, merely provided for the admixture of a certain amount of "Electrozone" (as Woolf called his solution) to crude sewage. At Brewsters, electrozone was added to the water flowing in a small brook, which ultimately made its way into the Croton Reservoir. Here, an almost complete sterilization was effected. As to how long these plants remained in operation, or as to the amount of success which attended the Havana plant, the writer is unable to say. In 1895 a company was formed in England to work the Woolf patents, and at first a temporary and then a more permanent plant was erected at Maidenhead. A long series of experiments showed that the hypochlorite solutions prepared would sterilize effluents from any ordinary filter or even an effluent from a precipitation tank. It was also found that to oxidize the organic matter present would require too great a consumption of free chlorine. A gallon of electrozone containing 4 grammes of free chlorine to the litre would sterilize 1000 gallons of tank effluent, or 1600 gallons of filtered effluent. Reckoning electricity at 4 cents a unit, and salt at $6.00 a ton, 1000 gallons of tank effluent could have been sterilized for 0.7 cents. Such a figure leaves out of calculation royalty, supervision, and repair and renewal of electrodes.

The experiments of the past summer at Guilford were carried out by the Oxychloride Co. The writer believes that an Atkins electrolyzer was used, in which a revolving drum of lead acts on the cathode, while the anode consists of a semi-circular trough of carbon. No figures have yet been published of the electrochemical efficiency obtained, cost of salt used, working strength, or stability of the electrolyte. Dr. Rideal, who has carried out a long series of experiments at Guilford, reports that 3¼ galons of oxychloride are needed to sterilize a filtered effluent. It is a pity that no figures of the cost of production are given.

The new company will have to fight against the apathy of British municipalities and water supply authorities. At present only chemical standards of purity of a sewage effluent are recognized. When bacteriological standards are laid down, it will speedily be apparent that treatment with ozone or treatment with hypochlorite solutions are the only alternatives which are reasonably cheap.

The Faraday Society.

The Society's meeting on the 23rd instant was fairly well attended, several members who are not regular attendants [illegible] the advanced proof of [illegible] on the dissociation theory of Arrhenius and van't Hoff. The enjoyment of the evening was greatly marred by Professor Kahlenberg's absence. Your correspondent, remembering vividly a celebrated heresy hunt at a northern Synod, in which there was war to the knife between two theological schools, and remembering also that the debating of opposing scientists is second only to that of theologians in order of rancor, went, expectant of hard hitting. In this he was not disappointed. A lengthy abstract of the paper having been read, Dr. Lowry was the first to speak. Speaking with extreme rapidity, never at a loss for a word or phrase, the speaker expressed his sense of a temptation to differ from Professor Kahlenberg at nearly every point, the paper being chiefly founded on a misconception of the theory as held at the present day. Point after point was raised and dismissed in a few words, the influence of viscosity being urged as an important factor in such investigations. Communications were then read from Mr. Wetham, Professor Abegg, Dr. Rudorf, Dr. Sand, and Dr. Desch. The general trend of their opinion was that, while the Arrhenius theory was not all-explaining, yet it was so successful that a mere citation of difficulties is not sufficient to occasion its rejection. What was really needed was an alternative theory leaving fewer unexplained difficulties. The debate was continued by Dr. Borns and Mr. Spiers, and closed by the vote of thanks moved by Mr. W. R. Cooper, who was in the chair. (A rather full abstract will be found on another page of this issue under the title Faraday Society Meeting.—Ed.)

METALLURGICAL PAPERS READ IN NOVEMBER

While neither electrochemical nor electrometallurgical in its scope, mention must be made of the paper read by Messrs. A. E. Seaton and A. Jude at the Institution of Mechanical Engineers on November 18th, entitled, "Impact Tests on the Wrought Steel of Commerce." The authors point out that tests as to the elastic limit and tensile strength, while in themselves valuable, do not afford all the indication which is desirable in the case of structures to rapidly repeated loads of one kind more or less suddenly applied, as bolts, studs, rails, etc., or for structures subject to alternating loads, such as the fixed and moving parts of machinery. Whereas tensile testing machines make practically no distinction between the steel that is utterly unfit and that which is eminently fit for the parts of a machine subject to shock or vibratory stresses, the drop-testing machines does most certainly and without fail make the discrimination. Further, wherever the drop-testing machine shows a steel to be good, the tensile testing machine corroborates it.

The subject of Mr. E. A. Weinberg's paper, read before the Institution of Mining and Metallurgy on November 17th, is probably more familiar to American engineers than to their English confreres who listened to the paper. Its title was "The Blake-Morscher Electro-Static Separator," and included, with its description and a narration of the practical working results obtained in Colorado on account of the Wagner mica plate static machine, used by the author in his own [illegible]

[illegible]

Up to the date of mailing this letter, very little attention [illegible] and Mr. F. W. Harbord (otherwise described in Canadian official documents as "The Commission") on this subject. *The Electrician* has published a leading article on the subject of a recitative rather than a critical order. Mr. Harbord is, however, to read a paper on the subject before the [illegible] (The report was abstracted and discussed at great [illegible])

MARKET QUOTATIONS

So far as chemicals are concerned there has been practically no change during the month of November. Apart from copper sulphate, which has now risen to £23 per ton, prices on November 30 are almost identical with those quoted on November 2. Bleaching powder is fetching £5.5.0 per ton, caustic soda (white 70 per cent) £10.15.0 per ton, and lead peroxide is £27.10.0.

Fine Para rubber is still rising 5s. 2d. to 5s. 5½d. per pound. The metals generally are advancing in price. Platinum is now quoted at £4.1.6 per ounce—an advance of 1s. 6d. during the month. Aluminium is stationary, ingots fetching £130 per ton, wire and sheet £168 per ton. Electrolytic copper bars have risen from £66 to £70.10.0 per ton ,electrolytic sheet and rod fetching, respectively, £87 and £81 per ton. H. C. wire has risen three farthings per pound to 9¼d. Cleveland pig iron has risen 3s. 0d., now reaching 47s. 8d. Block tin is now being sold at £137 to £138 per ton. Zince sheets have risen £2.10.0 to £29.7.6. Brass sheets and tubes have risen slightly in sympathy with the rises in copper, tin and zinc.

Faraday Society Meeting.

The ninth ordinary meeting of the Faraday Society was held in London, November 23, 1904, at the Institution of Electrical Engineers, Mr. W. R. Cooper being in the chair.

ELECTROLYTIC DISSOCIATION THEORY.

The first paper presented was by **PROFESSOR L. KAHLENBERG** of the University of Wisconsin. In the absence of the author it was read in abstract by the secretary.

Dr. Kahlenberg starts with the remarks that in creating the theory of electrolytic dissociation, the phenomena of actual electrolysis have played a minor part, and that the hypothesis of Clausius, which displaced the old Grotthus theory and brought conceptions of electrolysis into harmony with thermodynamic requirements, is by many chemists and physicists still regarded as the best mechanical explanation of the electrolytic process when all facts are fully considered. Dr. Kahlenberg then points out that the dissociation hypothesis is widely connected with van't Hoff's theory of solution. Upon the assumption of Arrhenius that free ions, part-molecules charged with electricity exist in an electrolyte and act like so many molecules, the general statement which van't Hoff had made for solutions was upheld,—namely, that dilute solutions having the same temperature and the same osmotic pressure contain the same number of dissolved molecules.

Dr. Kahlenberg then contends that the facts presented by Arrhenius in his original article in 1887 are not sufficient to serve as a basis for the assumption of electrolytic dissociation in solutions, and that while isolated facts may here and there be interpreted by means of the dissociation theory, the bulk of experimental evidence gathered since 1897 speaks strongly against the theory. Dr. Kahlenberg refers to his own work published in 1901, in which "a comparison of the feezing-point values with the molecular conductivity at 0°, and also of the boiling-point values with the molecular conductivity at 95°, revealed the fact that there is no such connection between freezing-points and boiling-points of solutions on the one hand, and their conductivity on the other, as is claimed by the theory of Arrhenius. In numerous cases not even a qualitative agreement exists." The objection that all the solutions employed were too concentrated to enable one to use the data to confute the dissociation theory, is met by Dr. Kahlenberg by the statement that the adherents of the theory, among them Arrhenius himself, have repeatedly used solutions as strong and even stronger than the most dilute solutions employed by Kahlenberg.

"Not all the adherents of the theory of electrolytic dissociaciation would be ready to admit with Whetham that the only satisfactory cryoscopic measurements at the extreme dilutions necessary to test the theory are those of E. H. Griffiths for cane sugar and potassium chloride. It certainly is demonstrated by Griffiths' work that the so-called molecular lowering of the freezing-point of potassium chloride is within very small limits of experimental error double that of cane sugar. This is merely one isolated experimental fact, and it has not yet been shown that the curve representing the change of the molecular-weight with the concentration runs parallel with the curve indicating the change of the freezing-point with the concentration in the case of potassium chloride within the range investigated by Griffiths. Nor has this parallelism between freezing-point and molecular conductivity been rigidly demonstrated for the solutions cryoscopically examined by Loomis. If one were therefore to take the attitude of Whetham, there would be no unobjectionable experimental evidence whatever on hand at present upon which to base the claim that freezing-points and molecular conductivity are related as held by Arrhenius."

"No one expects the gas equation to hold strictly for a normal solution or even for one considerably more dilute; but what one has a right to expect from the modern theory of solutions is, that with increasing concentration a solution should behave at least qualitatively as a gas does with increase of pressure. And this requirement is clearly not met, since while all gases behave alike under increase of pressure (so that van der Waals has been able to express their behavior by means of his well-known equation) solutions, as has been shown, often behave in a manner opposite to that of gases, and this, too, frequently in solutions that cannot be termed concentrated. This demonstrates, then, that the van't Hoff law is at best only approximate and must be supplied with great care."

Dr. Kahlenberg then calls attention to the fact that, while the molecular conductivity increases with the volume in very many cases, yet in some instances, like that of the caustic alkalies, the molecular conductivity first increases and then decreases with the volume. In still other cases, the molecular conductivity diminishes or remains practically constant as the volume increases. The only cases which conform to the requirements of the dissociation theory, even in general trend, are those in which the molecular conductivity increases with the volume.

Concerning the statement that it is the high specific inductive capacity of the solvent that causes electrolytic dissociation, Dr. Kahlenberg say that "in the face of the fact that numerous exceptions to the Nernst-Thomson rule are known, and that no quantitative relation between the conductivity of a solution and the dielectric constant of its solvent has ever been established, it would appear that the Nernst-Thomson rule is really untenable." Whether a solution will conduct electrolytically or not depends upon the specific character of both solute and solvent, and is quite independent of the dielectric constant of the solvent.

The approximately additive properties which some solutions exhibit has frequently been regarded as evidence in favor of the dissociation theory. "Additive properties, however, are known to occur where electrolytic dissociation is out of the question, and hence these properties cannot be used as an argument in favor of the Arrhenius hypothesis. In the case of solids the molecular heat is approximately equal to the sum of the atomic heats; in liquids, insulators as well as electrolytes, the molecular volumes and molecular refractions are approximately equal to the sum of the atomic volumes and atomic refractions respectively, and yet who would assume dissociation in these cases on that account?"

The color of aqueous electrolytes has been ascribed to the color of the ions. "Copper ions are blue; nickel ions are green; cobalt ions are red, etc. But I have shown that benzene solutions of copper oleate, nickel oleate, and cobalt oleate are also blue, green and red respectively, and that these solutions are as good as insulators as benzene itself. Furthermore, the cobalt oleate solution turns blue when heated, and on cooling turns red again, exactly as aqueous conducting solutions of cobaltous salts. It has, therefore, been demonstrated that the color of solutions is independent of their power to conduct the current, and cannot be used as an argument in favor of the dissociation hypothesis."

Instantaneous reactions sometimes occur between insulating solutions, and hence cannot be regarded as taking place between the free ions present.

A strong argument against the hypothesis is that it cannot be harmonized with the law of mass-action. The agreements obtained by Ostwald in his application of the dilution law to solutions of weak organic acids are only very rough, and to electrolytes *par excellence* it cannot be applied at all; Rudolphi's and van't Hoff's amended formulæ are purely empirical.

The power of coagulating colloids has been ascribed to free ions, but such power is not confined to solutions of electrolytes.

It is in the realm of non-aqueous solutions that, according to Dr. Kahlenberg, the theory proves especially impotent. We now know of such solutions that conduct electricity even better than aqueous solutions, and yet yield higher molecular weights than those computed from the formulæ of the solutes. The researches of Walden or liquid sulphur dioxide are quoted as being particularly conclusive.

The theory has been applied to fused salts. The author questions the validity of such application to what are practically 100 per cent. solutions; the hypothesis is avowedly only strictly true for infinitely dilute solutions.

Dr. Kahlenberg then develops his own views in the following paragraphs, which are quoted in full:

"I should not like to close these pages without indicating the direction which in my opinion further investigations on solutions must take. All treatises on physics and chemistry seek to draw a sharp distinction between the processes of solution and chemical action; the former is commonly described as purely physical in character, the latter as more deep seated and as caused by specific attraction termed chemical affinity. It is well known that some substances react chemically (using that term in its usual sense) with each other, and again others do not; and that the rate of interaction and the final condition of equilibrium reached are subject to influences of temperature especially, but also to pressure and concentration. The same may be stated of the process of solution. Some substances dissolve each other, others do not; and again, the process, like that of chemical action, is subject to influences of temperature, pressure, and concentration. Energy changes, such as thermal changes, electrical changes, expansions and contractions, and evolutions of light are known to accompany the processes of solution as well as those of chemical action. With the exception of the mass, the properties of chemical compounds are never exactly equal to the sum of the properties of the constituents entering into them; the same is true of solutions. Furthermore, those properties which are approximately additive in character in the case of chemical compounds are also approximately additive in the case of solutions. Examples of such properties are the molecular volume, molecular refractive power, and specific heat. Again other properties like, for example, the optical rotatory power, the specific inductive capacity, which are not additive in chemical compounds, are also not additive in the case of solutions.

"In fact the only distinction between a solution and a chemical compound is that the latter conforms to the laws of definite and multiple proportions and the former does not. But it must be borne in mind that so-called pure chemical compounds are obtained only by subjecting the total reaction products to certain purifying processes, such as evaporation, distillation, sublimation, crystallization, washing, extraction, etc. All of these processes put the reaction products, which are in general solutions (using that word in its broader sense) under special duress, as a result of which there are obtained as cleavage pieces of these solutions, as it were, phases whose [illegible] through a greater or lesser range of temperature, pressure and contact with other phases. *The* [illegible] *'identical' in* [illegible] *chemical compounds are merely the* [illegible] *under special stress or duress repre-* [illegible] *process.* The process of solution is thus the general case of the interaction of bodies, union resulting when the specific attraction, commonly called chemical affinity, between them, is under the existing conditions sufficient to cause an interpenetration, a fusion or blending of their masses, as it were. Furthermore, adhesion, imbibition, absorption, adsorption and imbibition are also due to the same specific attraction which causes solution and chemical action. Adhesion is really to be regarded as an unsuccessful attempt at solution. Thomas Graham was unquestionably right when he stated that from adhesion to solution and chemical action there is every stage of gradation. It is scarcely necessary to add that these views were also entertained by numerous other scientists of note, among whom Bunsen was especially prominent. The truth of this view will force itself upon the mind of any one who seeks to work out the problem of solutions in the laboratory rather than at the writing-desk.

"It need hardly to be emphasized that the recognition of the importance of the laws of definite and multiple proportions, which have been found to hold in so many cases of phases that maintain their composition for considerable ranges of alterations of temperature, pressure and nature of co-existing phases, is not at all affected by the above considerations. The discovery of these laws has been of inestimable value in analytical and synthetic work, and in systematizing our knowledge; but it is a mistake to think that they necessitate the conclusion that the processes of solution and chemical action are different in character and are caused by different agencies. The fact that the quantities of similar solutes which must be added to equal quantities of a given solute to produce solutions of the same vapor tension, are to each other as the quantities in which such solutes unite with other substances to form stereotyped chemical compounds, constitutes a strong argument for regarding the processes of solution and chemical action as identical in character.

"It thus becomes evident that in investigating solutions we must begin with the most concentrated and end with the most dilute; the latter will appear simply as a limiting case. So, for instance, the change of the vapor tension of solutions with the concentration must be studied from the strongest solutions obtainable to the dilutest that can still be measured throughout the range of temperatures at which the solutions can exist at all. And this work must be done for a large number of solutes in a large number of solvents. Such data being at hand, the equations expressing the changes of vapor tension with temperature and concentration may be written. Work of this character has only barely begun; but judging from the results at hand, analogous substances will exhibit similar behavior; and though it is not to be expected that one equation will serve for all solutions, similarities between the equations holding for different solutions will not be lacking."

The abnormal values of molecular weights of substances in solution as determined by change of vapor tension, freezing or melting-points is according to Kahlenberg, merely a measure of the affinity between solute and solvent, and has nothing to do with any supposed dissociation; the calculation of mole- [illegible] liquids, and are by no means in accordance with the requirements of the gas laws. Here, again, as in solutions, *affinity* is [illegible]

[illegible] chemistry has been largely [illegible] of the effects of temperature, pressure and concentration on the progress of chemical reactions, and properly so. In this intensive work it has at times been forgotten, however, that affinity must exist before union can take place, and that temperature, pressure and concentration are simply to be regarded as modifying factors aiding or retarding the tendency of affinity. By some it is even deemed as somewhat old-fashioned to talk about affinity. But upon the distinct recognition of affinity, and a careful investigation of the laws governing it, clearly depends the future progress of chemistry, physical chemistry, and physiology. We must learn to measure affinity quantitatively; but in order to take into account the phenomena of morphology as they confront us in crystals, but particularly in living beings, we need to learn to study the direction as well as the strength with which affinity acts under given conditions. Finally, the question why certain solutions, molten salts, etc., conduct electricity and others do not, will probably not be answered until we can tell why a stick of silver conducts electricity and a stick of sulphur does not. These questions really involve a better understanding of the relation between electricity and gross matter, a problem which is apparently being attacked with promise by J. J. Thomson and his co-workers. Until we have more light on this subject we can hardly hope for very material improvements of our views of the nature of electrolytic processes."

* * * * *

The paper of Dr. Kahlenberg elicited a long and animated discussion. W. C. D. Whetham, in a written communication, pointed out that the problem of the nature of solution was independent of the difference between electrolytes and non-electrolytes, which was all the theory attempted to explain. The ions must be conceived as being free from each other, not from all chemical combination, in order to explain the observed phenomena of conduction.

Prof. Abegg also sent in a written communication. Prof. Kahlenberg's reasons for doubting the theory are merely facts which cannot be explained by the Arrhenius theory alone. The latter explains the behavior of dilute aqueous solutions, where the non-ionized molecules are, as a rule, not associated, and not cases where association occurs, such as in non-aqueous solutions. The author further overlooks the fact that conductivity depends on the mobility of the ions, as well as on the degree of ionization. The theory can be harmonized with the law of mass action, as the work of Rothmund and Drucker and Jahn has shown.

Dr. G. Rudorf referred to the question of the color of solutions, and showed that on the assumption that the molecule of copper sulphate has nearly the same light-absorbing properties as the ion, all difficulties disappear. The theory quite explains the reactions mentioned in benzine solution, if only very few ions are assumed to be present.

Dr. T. M. Lowry said that Prof. Kahlenberg's criticisms were mostly expended on a theory which was not the dissociation hypothesis as conceived to-day. The theory does hold good for concentrated solutions, only we are not yet able to measure the degree of ionization in such cases. The gas laws are not quite true for moderately dilute solutions, because some of the molecules of the solutes are in combination, and not free. He dealt at length with the various points raised by the author.

Dr. H. Borns showed that Prof. Kahlenberg's capital mistake was that he attacked everything that had ever been said or done with the theory. He ignored recent modifications of it that the ideas of hydrolysis, complex ions, and association have caused.

Dr. H. Sand, (in a communication) dealt with the differences between the theory of Clausius and that of Arrhenius, and discussed some of the approximate assumptions that lead to numerical disagreements in the applications of the latter.

Dr. C. H. Desch discussed Prof. Kahlenberg's experiments with copper oleate in benzine solution, and described some experiments of his own bearing on the subject. He thought the difficulties raised by non-aqueous solutions very serious ones.

F. S. Spiers drew attention to the theory of Traube which gave numerical expression to some of the conceptions of solution apparently held by Prof. Kahlenberg, and explained the Arrhenius coefficient i in terms of the affinity between solute and solvent.

Hydrogen-Oxygen Cell.

F. J. Brislee communicated a paper entitled "The Potential of the Hydrogen-Oxygen Cell," which was read by Dr. F. M. Perkin. Measurements of the potential of a hydrogen-oxygen gas cell with platinum electrodes and a normal acid solution show different results when the oxygen electrode is charged with oxygen electrolytically, or charged by passing oxygen gas through the liquid. Experiments were made to determine the influence or hydrogen peroxide, ozone, and persulphuric acid upon the oxygen potential. The results obtained showed that the addition of hydrogen peroxide lowered the oxygen potential, while addition of ozones and potassium persulphate raised it.

By employing platinum electrodes of extreme thinness (prepared by depositing platinum on glass), the same potential of the combination was obtained independently of the method employed for charging the electrodes. This potential agreed well with that calculated from the Helmholtz formula.

In the discussion which followed, H. L. Joly referred to the effect of persulphates, and said that in accumulators it had been proved that their presence did not account for the rise of E. M. F. on charge.

Some Present Problems in Technical Chemistry.*

By Prof. W. H. Walker, Ph. D.

Technical chemistry may be regarded as the performance of a chemical reaction or series of reactions on a scale sufficiently large and by a method sufficiently economical to enable the product to be sold at a profit. The problems which confront the investigators in this field of endeavor may, therefore, be divided into two classes, according as they pertain to the chemical reaction involved or to the process to be employed in carrying on this reaction. The first division is pure chemistry, even though the results of the solution be utilitarian; the second is chemical engineering. There is in reality no dividing line between the two. It would be difficult to find an investigator in the field of pure science who does not hope, and indeed believe, that the results of his labor will at some time prove of value to humanity; may ultimately be utilitarian. On the other hand, few, if any, chemical manufacturers would admit that in solving their chemical problems they do not utilize the most scientific methods at their command. The research assistant is in the last analysis utilitarian; while the successful chemical engineer is pre-eminently scientific.

Probably in no country have the problems confronting the chemical industries been so successfully met as in Germany; yet Germany does not excel in chemical engineers. Engineering enterprises, mechanical, civil and electrical, as well as chemical, are carried on as successfully in England and America as they are in Germany, and still the latter leads the world in her chemical manufacturers. The explanation for this lies in the fact that Germany pays the greatest attention to the first class of problems, as above divided, and recognizes that pure chemistry is inseparably connected with her industries; that the application of new facts and principles follow rapidly when once these facts and principles are known. Most of her problems in technical chemistry are first considered problems in pure chemistry and studied in accordance with recognized methods of modern research by men fully trained in pure science. If these men are also chemical engineers the ultimate solution of the problem is proportionately hastened; but they are first of all men trained in the spirit and methods of scientific research.

In general, an investigation may be prompted by either or both of two incentives; either by the pleasure to be derived from achievement and the love of scientific study for itself, or, by the hope that from the investigation some immediately useful result may be obtained. Yet between the product of the first motive—pure chemistry—and the ultimate result of the second—technical chemistry—a difference does not necessarily exist. The fact that a piece of work is undertaken and carried on with the predetermined purpose of applying the results to a practical or commercial end does not in itself render it any the less a study in pure chemistry. The method of thought and action employed will be that of the investigator in pure science, whatever the ultimate object may be. To make the result of the work an achievement in technical chemistry an important contribution must then be made by the chemical engineer, in order that the conditions forming the definitions of the term "technical chemistry" as already stated may be fulfilled.

In trying to point out, therefore, some of the important problems in technical chemistry, no attempt will be made to distinguish between the part which must first be played by pure chemistry in their solution, and that which will still remain to be done, by the chemical engineer, to make this contribution utilitarian.

There is always a tendency to measure the importance of a subject by the extent of one's knowledge of it and the depth of the interest one has in it. In order, therefore, that we may obtain a proper perspective, we must consider a problem important in proportion as it affects the greatest number of people; of moment according as the results of its solution will be far-reaching in their effects, or be but of local benefit.

From this point of view the first industry to demand attention is the manufacture of fertilizers. In the last ten years the product of this industry in the United States alone has increased from 1,900,000 tons to 2,900,000 tons, an increase of over 50 per cent. This increase is probably more marked in America than in the older countries of Europe, because the necessity of replenishing the virgin soil was there reached long ago, while with us it is only begun. The magnitude of the industries which are dependent directly or indirectly upon agricultural products is so well recognized that it needs no discussion here. That the supply of crude material from which plant life derives its nourishment should be maintained is therefore a source of responsibility for the present, as well as for future generations. Of this, as of every great industry it may be said that the supply of raw material for to-morrow is a problem for to-day.

Dr. H. W. Wiley, of the United States Department of Agriculture, has pointed out the surprisingly large amount of potash, phosphoric acid and nitrogen which is yearly taken up by the agricultural crops alone. The average percentage of ash in all of the important crops has been accurately determined and their percentage composition in respect to potash and phosphoric acid is known. In addition to this we have a satisfactory knowledge of the percentage of albuminous matter contained in the more important agricultural products. From these figures and the reports of the United States Department of Agriculture we can calculate the amount of potash, phosphoric acid and nitrogen consumed each year. Allowing a value of 4 cents a pound for potash, 5 cents for phosphoric acid, and 12 cents for nitrogen, the total value of these ingredients for a single year amounts to the enormous sum of $3,200,000,000. To be sure this is not all removed from the farm and lost to the soil; but that which remains in the form of straw and manure is but a small percentage of the whole. Straw is generally burned, while the soluble salts of the manure heaps are often allowed to leach out and go to waste. When in addition we consider the [illegible] [illegible] involved in the modern methods of sewage [illegible], [illegible] instead of being returned to the soil, these [illegible] [illegible] are carried to the ocean, the net loss of these [illegible] has [illegible] aggravated.

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* An address delivered before the International Congress of Arts and Sciences at St. Louis.

Of the three most important ingredients making up a fertilizer for [illegible] purposes, phosphoric acid alone seems to be at hand in practically inexhaustible quantities. Slag, rich in phosphoric acid from certain metallurgical processes is already much used as a source of the material. Fresh deposits of phosphate rock of such enormous extent are being brought to light almost every day that our supply of this material may give us little immediate concern.

Although the Strassfurt region of Germany may continue to ship undiminished quantities of potash salts, the second important ingredient of a fertilizer, the world's supply cannot be said to be on a perfectly satisfactory basis until independent sources are developed. In the year 1902 the value of the potash salts imported into the United States amounted to 4½ million dollars. The recovery of potash from wood ashes, while once an important industry, must diminish as the value of hard wood increases. While there are doubtless natural beds of potassium salt still to be discovered, the time seems rapidly approaching when we should render more readily available the great amount of potassium distributed throughout the mineral kingdom. Rhodin had already accomplished much toward this end when he showed that feldspar could be made to yield the greater part of its potash when it was heated with lime and common salt. Clark has found that when the mineral leucite, with its 21 per cent potassium oxide is heated with ammonium chloride, the potassium is converted into chloride and is easily separated from the melt. If this reaction could be extended to orthoclase and the ammonia recovered by treatment with lime, the enormous quantity of potash contained in this mineral would be at our service.

It is, however, to the supply of available nitrogen that the greatest importance attaches. The sodium nitrate producing countries of South America exported last year 1,300,000 tons, a large percentage of which came to America. Egypt and the Southwestern United States have nitrate deposits, but of their extent and value little is as yet known. Of the other form of available nitrogen, ammonia, our main supply is at present from the destructive distillation of coal. Although the introduction of by-product coke ovens has increased this supply, our domestic production is now not over 40,000 tons a year.

In the atmosphere, however, we have a never-failing source of nitrogen which needs only to be converted into other forms to be of the greatest value. It is interesting to note that even as long ago as 1840 this same problem was the subject of considerable experimentation and the basis of several technical processes. In this year there was erected in France a plant for the manufacture of potassium ferro-cyanide, which depended on the atmosphere for the supply of nitrogen, and which at one time turned out almost a ton of product per day. From this time until the present, the utilization of this inexpensive and inexhaustible supply of raw material has been an attractive field, and has held the attention of many investigators. It had long been known that while carbon and nitrogen [illegible] could not be made to unite, the union was [illegible] when these elements were brought together in the presence of a strong alkali. The technical difficulties in the way of successfully applying this reaction seem to have been [illegible] [illegible] [illegible] of the [illegible] and the [illegible] of alkali through volatilization. With the advent of cheap electricity and the consequent development of the electric furnace, this [illegible] [illegible] [illegible] [illegible] [illegible] [illegible] [illegible]. The destruction of the [illegible] [illegible] [illegible] [illegible] by [illegible]ting the heat with[illegible] [illegible] [illegible] [illegible] [illegible]. When [illegible] [illegible] [illegible] [illegible] [illegible] [illegible] [illegible] [illegible] and a higher temperature maintained, it was found that a carbide was formed as an intermediate product and that nitrogen readily reacted with the carbon thus held in combination.

Among the investigators who have thus far taken advantage of this reaction may be mentioned the Ampere Chemical Co., located at Niagara Falls, and the group of men represented by the Siemens and Halske Co., of Berlin. The former first produces a carbide of barium and then converts it into barium-cyanide by passing over it air from which the oxygen has either been removed or converted into carbon monoxide. Robert Bunsen long ago showed that by using steam the nitrogen in an alkaline cyanide may be converted into ammonia. In this case barium oxide would be left to be returned to the furnace, and to continue the cycle. When advantage is taken of the process discovered by Professor Ostwald, by which ammonia is converted into nitric acid through the medium of a catalyzing or contact agent, the production of nitrates by way of the cyanide reaction is easily foreseen.

The Siemens and Halske Co. prepared, in addition to cyanide and ammonia, by use of the carbide-nitrogen reaction, a new compound in technical chemistry, calcium cyanamide. In contradistinction to the cyanides the nitrogen of this compound is available for plant food and can take the place of the more common nitrogen salts in commercial fertilizers. The technical difficulties in the way of the economic application of these processes are doubtless very great, but when one considers the advance which has been made in the last five years he has ample reasons to believe that it will not be a great while before the synthetic preparation of the cyanides, ammonia and nitric acid from atmospheric nitrogen will be on a commercial basis.

The old reaction by which nitrogen and oxygen were made to unite through the agency of a high potential electric discharge has been made the basis of a process for the manufacture of nitric acid by the Atmospheric Products Co., operating at Niagara Falls. For agricultural purposes it is proposed to absorb the nitric acid thus formed in milk of lime, and so produce an exceptionally cheap product. There still remains much to be done before this can be called a technical process.

A very much less technical, but so far as our knowledge at present goes, a more promising method of fixing atmospheric nitrogen in the form of nitrates is through the agency of bacteria. While it is true that one group of bacteria has the power of breaking down nitrates with the production of nitrogen gas; there are other groups which are equally able to absorb elementary nitrogen with the production of nitrates. A great deal of excellent work has recently been done by the United States Department of Agriculture with the result that cultures for the artificial inoculation of the soil may now be obtained in considerable quantity. It has been found that these bacteria when grown upon nitrogen freemedia may be dried without losing their high activity. When immersed in water they are easily revived. A dry culture similar to a yeast-cake, and of about the same size, can thus be sent out and used to prepare a fluid in which the original nitrogen fixing bacteria may be multiplied sufficiently to inoculate a number of acres of land. The amount of material thus obtained is limited only by the quantity of the nutrient water solution used in increasing the germs. Field experiments have shown the wonderful activity of these bacteria in fixing atmospheric nitrogen and the splendid crops which may be grown upon what would otherwise be almost sterile soil.

In this one problem of our future supply of available nitrogen for agriculture as well as general manufacturing purposes, we note the aid which technical chemistry draws from the other departments of natural science. The electrical engineer and biologist have already contributed a great share in its solution. There remains, however, no small amount of work for the technical chemist to perform before the desired end is reached.

(To be continued.)

ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS.

By George P. Scholl, Ph. D.

Electric Furnaces and Furnace Products.

Process of Reducing Aluminium or other Metals.—H. S. Blackmore, Mount Vernon, N. Y. Patent 775,060, Nov. 15, 1904. Application filed August 22, 1904.

The invention relates particularly to the production of aluminium from its oxide, but covers in general the idea of liquifying refactory oxides by dissolving them in a readily fusible metallic oxide or mixture of such oxides, and then subjecting the liquid bath thus obtained to electrolysis in such a manner that only the metal desired is obtained, while the oxides acting as a solvent are not decomposed. In the case of aluminium in particular, the inventor takes oxide of lithium and oxide of calcium in the proportion of four of the former to one of the latter, which he fuses by the action of an alternating current, so as not to decompose them. After the fusion of these oxides is complete, he adds oxide of aluminium to the bath, which is stated to readily dissolve, and then subjects the bath to the action of a direct electric current for the purpose of producing metallic aluminium. The apparatus for carrying out the electrolysis is shown in longitudinal vertical section in Fig. 1. It consists of an iron box *A*, lined preferably with carbon *B*, and covered by an insulating cover *X*, through openings in which pass the carbon electrodes *G* and *T*. The box forms the cathode, while the electrodes *G* serve as anodes. In starting the apparatus an alternating current is passed between the alternate carbon electrodes *T*, and the carbon lining of the box *A*, and arcs are produced by bringing the electrodes close to the carbon lining. Lithium and calcium oxide in the required proportion are then fed in and fused, the electrodes being gradually separated further and further, as the molten material accumulates in the vessel. Aluminium oxide is then introduced into the bath, and a direct electric current is passed between the carbon anodes *G*, and the carbon walls of the vessel *A*, for the purpose of electrolyzing the aluminium compound. The metal accumulates on the bottom and is withdrawn through tap hole *K*, by moving plug *T*. Instead of using the above solvent bath for the oxide of aluminium, fused lithium-glucinum oxide can be used in cases where low specific gravity of the bath is required. It is stated that when lithium and calcium oxides are fused together as described above to form the solvent bath for aluminium oxide, the solution of the latter takes place without formation of aluminates. By adding copper oxide with aluminium oxide to the solvent bath, copper aluminium alloys may be produced by electrolysis. The inventor claims that by the above described method he effects an economy in the cost of reduction by substituting the less expensive alternating current for melting purposes, in place of the direct current, and that the solvent mixture, viz., lithium and calcium oxides, are fusible with the expenditure of less energy than is required to maintain the fusion of the more expensive and more resistant fluoride compositions employed heretofore. Another advantage claimed is that the solvent action of the molten oxide bath is more than three times that of the fluoride bath, whereby the volume of solvent necessary to liquify a given quantity of oxide of aluminium is largely reduced, with the further advantage, therefore, of a saving in energy to keep a given quantity of the solvent in the state of fusion. Particulars about the yield of the operation, current density ,etc., are not given.

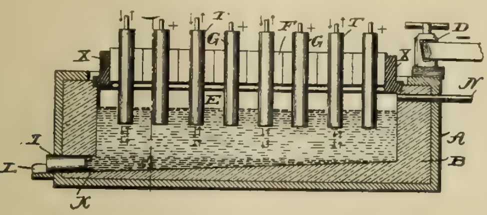


FIG. 1.—ALUMINIUM APPARATUS.

Process of Electrically Treating Materials.—W. S. Franklin, South Bethlehem, Pa. Patent 775,031, Nov. 15, 1904. Application filed Dec. 3, 1900.

The process is described as applied to the treatment of iron ore and glass, and it is carried out in a furnace, the lower portion of which is formed in the shape of a pear-shaped chamber. The furnace was described and illustrated in Electrochemical Industry, Vol. II., page 32. Resistance heating is used by filling the furnace chamber with a molten electrical conductor, such as slag or glass, and passing the current between a carbon electrode arranged in a layer around the circumference of the furnace chamber and another carbon electrode in the form of a rod, which is arranged vertically, and either dips into the molten conductor, thus heating it entirely by resistance, or is raised above its upper surface, in which case the heat of the arc developed between the electrode and the surface of the conductor will be added to that developed by the resistance of the conductor. When applied to the reduction of iron ore, the molten conductor is preferably slag, the charge of ore, coal, limestone, etc., being fed in separately, and the reduced iron collects on the bottom of the furnace, whence it can be drawn off by a suitably arranged tapping hole. When the furnace is used for the purpose of melting glass, glass itself serves as the molten conductor, and the raw materials are fed in in suitable proportions. They become melted on their way down and are fused and form glass when they reach the surface of the molten conductor. The form of the furnace as shown in the specification, when used for the manufacture of glass in the manner described above, is open to the objection of the contamination of the glass by carbon from the electrodes.

Electric Furnace.—R. Raddatz, Milwaukee, Wis. Patent 775,282, Nov. 15, 1904. Application filed June 23, 1899.

The electrodes in this furnace are arranged at an obtuse angle to each other and are movable. Their ends are covered by a hood formed of a highly refractory material, so that the arc between them, the substance immediately under treatment, and the supply of incoming material are protected. The material under treatment is supplied to a hopper from which it escapes on an endless belt which carries it into the vicinity of the electrodes. An electromagnet is used to deflect the arc upon the material under treatment. Provision may also be made for an opening in the cover for the purpose of introducing a stream of air, gas, oil or vapor for deflecting the arc. The stream of air, gas, etc., may carry solid particles with it, either of the same material as that under treatment or others, the purpose of introducing these particles with the current of air or gas being primarily to insure the perfect formation and the persistence of the arc, and to permit a greater separation of the electrodes than would otherwise be feasible. Details of mechanical arrangements are described and illustrated for the automatic regulation of the electrodes to compensate for their burning away, in order to render the action of the furnace perfectly uniform. The material is intended to be fed in a comparatively thick or heavy layer, so that the unfused portion laying beneath that which is fused would protect the bed or apron from the excessive heat. The furnace may be used for either direct or alternating current. When direct current is used, however, the positive electrode wears away more than the negative one, and it is therefore [illegible] to heat the material under treatment thoroughly or [illegible] the arc from the negative toward the positive electrode on account of the positive being the hotter, the greatest heat was available for the resulting work of [illegible] of the material.

Electric Furnace.—A. C. Higgins, Worcester, Mass. Patent 775,654, November 22, 1904. Application filed April 4, 1904.

The invention is primarily to be applied for the intermittent fusion of highly refractory materials, where the furnace has to be torn down after each batch of material is finished. Two serious disadvantages have manifested themselves with this type of furnace. The first was the question of providing a satisfactory refractory lining, for which purpose carbon was mostly used, which was, however, expensive, inasmuch as the lining had to be destroyed each time, and it was necessary to carefully and laboriously build it up again for the next run. There was also considerable time lost in discharging the furnaces and getting them ready for the next run. As shown in Fig. 2, the furnace is of the arc type, the electrodes 10 and 11 being held stationary in heads 12 and 13, to which the conductors 14 and 15 are connected. The crucible is constituted by the shell 16, which fits into the base ring 17, the latter being mounted on the base, 18. The base is raised or lowered by a suitable mechanism, such as a direct-plunger hydraulic elevator, with plunger 19. As the material under treatment is melted, the carbide is lowered until it is full, or the run is completed. A hood, 20, is provided for the purpose of confining the gases and forcing them to escape through the chimney 21, feed opening being provided. The shell 16 is made of boiler plate, and is slightly smaller at the top than at the bottom. A water-distribution pipe, 25, is mounted near the top of the furnace on brackets, 26, and is provided with a number of small holes through which water for cooling purposes is sprayed on the shell. The waste water escapes through pipe, 24. The numerous small streams of water playing upon the shell thus flow over the latter's surface in a continuous sheet, which covers the entire surface of the shell, and chills whatever molten material comes in contact with it. The layer of chilled material thus serves as a lining for the molten mass, the temperature of the lining being kept so low that it will not injure the shell. The water is therefore not confined at the points of the greatest heat, and the danger of explosions is reduced to a minimum. Moreover, the entire flow of the water is constantly exposed to the supervision of the furnace operator, who can regulate the flow of water to suit the conditions of the run, and receives ample warning of any danger to the shell. At the end of the run the furnace is easily taken apart by lifting the shell, with the fused and hardened mass in its interior, off the base and replacing it by another, after which the operation can be commenced again. The furnace seems to be primarily intended for the production of the artificial abrasive, alundum, in which case [illegible] of alumina is fused in it.

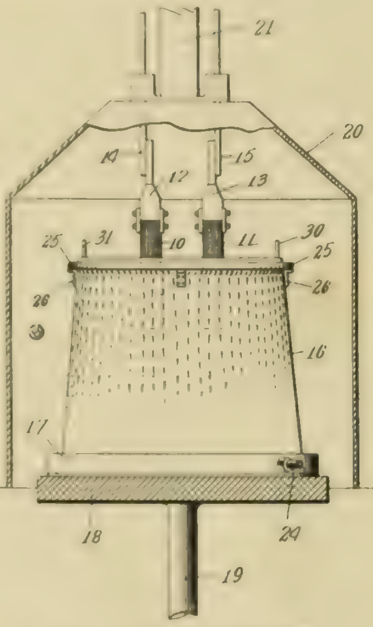


FIG. 2.—ALUNDUM FURNACE.

Electric Heater.—L. F. Weggen, Jefferson City, Mo. Patent 775,714, Nov. 22, 1904. Application filed April 22, 1904.

The heater described is stated to be of peculiar value to boot and shoe factories for the purpose of heating the tools used for removing wrinkles from the toes or tips of boots and shoes. It comprises a casing filled with asbestos, within the head of which is mounted a cap-shaped vessel of iron, lined with copper, and surrounded by a coil of resistance wire of German silver. Suitable arrangements are made for introducing different lengths of this resistance wire into the circuit, and thus regulating the temperature.

ELECTROLYTIC PRODUCTION OF METALS AND COMPOUNDS.

Manufacture of Barium Hydrate.—F. Jahn, Ridley Park, Pa. Patent 775,752, November 22, 1904. Application filed Sep. 5, 1903.

The process depends upon the electrolysis of a solution of barium sulphide in a cell with three compartments divided by porous diaphragms, the outside compartments being anode, and the middle one the cathode compartments. The vessel is lined with an insulating material, such as cement, and the electrodes are preferably iron plates. The inventor has found that the anode plate should have a much larger surface than the cathode plate, good results being obtained with an anode about four times as large as the cathode. This is the reason for providing two anode compartments and only one cathode compartment. In carrying out the process a strong solution (about 35 per cent.) of barium sulphide is introduced into the anode compartments, while the cathode compartment is filled with a weak solution of barium hydrate. The surface of the solutions is preferably covered with a layer of coal oil, in order to exclude the air. The current density should be rather high, about 9 to 15 amperes per square foot of anode surface, and 40 to 70 amperes per square foot of cathode surface. As a result of the electrolysis, sulphur is stated to be precipitated at the anode in an insoluble state, thereby converting the barium sulphide into hydrate. At the beginning of the process the sulphur thus precipitated at the anode is redissolved by the sulphide liquor with the formation of polysulphides until the whole of the sulphide is converted into the polysulphide. After this has taken place, the sulphide is precipitated and not redissolved. When the anode liquor contains from 75 to 80 per cent. of the dissolved barium as hydrate, the electrolysis is stopped. With electrodes of the above size, the inventor finds that he can obtain about 30 per cent. of the hydrate, which is formed in a very pure state, from the cathode compartment, the other 70 per cent. of the hydrate being at the anode. From the anode liquid the hydrate can be obtained by crystallization, the precipitated sulphur being also recovered.

Process of Extracting Gold Ores.—H. R. Cassel, London, England. Patent 775,597, November 22, 1904. Application filed May 22, 1903.

The process aims at providing means for the solution and extraction of precious metals from their refractory ores, such as tellurides and sulphides, or from slimes containing them, without previous roasting. The process is carried out in a round wooden tank, provided with a vertical central shaft, which carries a stirrer. Around the periphery of this vat there are arranged a series of carbon rods or plates, placed in a vertical position. They are alternately connected to the negative and the positive pole of a source of current. The pulverized ore is converted into a pulp by mixing it with a solution consisting of water containing either a cyanide and a bromide, or a cyanide, a bromide and a chloride. Bromates and chlorates may also be added, the different ores requiring variations in the composition of the electrolyte as well as in the current. The following composition of the electrolyte is stated to be usually employed with good results for the treatment of one ton of ore: one ton of water, 100 pounds of sodium chloride, 3 pounds of sodium bromide, and 2 pounds of potassium cyanide. If the chloride is omitted and the bromide alone is used in conjunction with the cyanide, the quantity of the bromide must be increased in order to make the electrolyte sufficiently conductive. The presence of the bromide has, however, been found very desirable in order to effect a high extraction of over 90 per cent., and when it is omitted, the results are negative. The chloride is added mainly in order to make the electrolyte better conductive. The pulp is charged into the vat and is kept agitated, while an electric current is simultaneously passed through it. A high-current density, exceeding 10 amperes, and preferably about 40 amperes per square foot of anode surface is used, about 100 amperes per ton of ore. It is stated that the current density may be varied according to the nature of the ore. No density, however, is to be employed which would cause a substantial deposit of gold upon the cathode, as would happen if the current density was too low. An ore of about the following composition is stated to be well adapted for treatment by the process: silica 64 per cent, alumina 27.5 per cent., lime 1.0 per cent, oxide of iron 3.5 per cent, sulphide of iron 3 per cent, oxide of manganese 0.5 per cent magnesia, gold, tellurium 0.5 per cent. It is stated that the extraction is more rapid and more complete, and that more metal is dissolved than would be the case were the current not employed; the increased extraction amounting in some instances to over 40 per cent. The cyanide alone is decomposed, practically all the haloid salts being found at the end of the operation. After electrolysis the pulp is filter-pressed, or the solution separated otherwise from the solids, after which the gold is precipitated by zinc, or by any other method. The recovered solution may be used on a fresh batch of ore, after sufficient cyanide has been added to replace the amount decomposed. The extraction is stated to be generally complete in about twelve hours.

Electroplating Isolated Designs on Vitreous Surfaces.—L. Blower, New York. Patent 774,976, November 15, 1904. Application filed March 23, 1904.

The process consists in fusing a mixture of pulverized metal and flux in the desired design upon the vitreous surface and then increasing the thickness of the metallic deposit thus obtained by electrodeposition. The metallic powder is mixed with a suitable flux, the resulting mixture being mingled with a suitable medium, such as silicate of potash. The desired design is then laid out with this mixture on the vitreous surface, and the article then fired so as to fuse the mixture into the design. During this operation of firing the metal and the flux are melted, and the flux unites with the vitreous surface and is separated from the metal in such a degree that a metallic surface is exposed suitable to form an electric conductor in the subsequent plating process. The article is then introduced into a plating bath and the surface of the design connected as cathode, the anode, of course, being made of the metal which it is desired to plate on the design. Suitable provision is made to connect the various parts of the design to the electrode. After the electrodeposition has attained the desired thickness, the articles are removed from the bath and the surface of the electroplated metal may then be polished, burnished, frosted, or finished in any desired manner. The process is chiefly intended for decorating or applying labels to articles of glass, porcelain, china, etc.

Storage Batteries.

Protective Coating or Covering for Storage-Battery Plates.—A. Meygret, Paris, France. Patent 776,192, Nov. 29, 1904. Application filed June 29, 1903. Renewed April 6, 1904.

Protective Covering for Storage-Battery Plates.—A Meygret, Paris. Patent 776,480, Nov. 29, 1904. Application filed June 29, 1903. Divided and re-filed July 27, 1903. Renewed April 6, 1904.

The two patents relate to a protective covering for storage batteries, in a similar manner to that described in the inventor's patent 756,176, abstracted in Electrochemical Industry, Vol. II., page 206. The protective covering described in the first one of the above patents is composed of a solution of tetraacetate of cellulose or tetraacetate and tetrabutyrate of cellulose. The plate after being filled with the active material is dipped into either one of these solutions, which cover it with an elastic protective film. The solution described in the second patent, which is to be used for the same purpose, is composed of tetrabutyrate of cellulose, without admixture of any other body. The protective covering thus formed is pierced with a number of fine needle points or fine gashes, in order to allow the electrolyte to reach the active material beneath it. This elastic coating prevents the active material from becoming loose and detaching itself from the plate.

Battery Charging Apparatus.—H. G. Pape, New York. Patent 775,732, Nov. 22, 1904. Application filed May 17, 1904.

The apparatus consists of a flexible cord which is to be attached to a lamp or other socket connected with a source of current. At the other end of this cord there is a connector provided with two female terminals, into which a similar connector with male ends connected to the battery terminals is to be inserted. A current indicator is used in order to ascertain the proper pole-connection. The device is particularly intended for charging small pocket-batteries for audiphone sets.

Galvanic Elements.

Method of Converting the Energy of Fuel into Electrical Energy.—Hugo Jone, Chicago. Patent 775,472, Nov. 22, 1904. Application filed Nov. 18, 1901.

The above process is a simplification of the inventor's first patent which was described in Electrochemical Industry, Vol. II., page 326. (See also Vol. I., page 586; Vol. II., page 18.) It consists essentially in the production of an electric current in a galvanic cell by using tin as the positive and carbon as the negative pole, the electrolyte being molten caustic potash. Mercuric oxide is used as the depolarizer, which becomes reduced to mercurous oxide or metallic mercury during the operation, and is regenerated in a separate vessel. The tin is oxidized to oxide of tin, and the latter is constantly reduced in the apparatus by means of fuel gas passed into the cell. The fuel gas is generated in a gas producer in about the same manner that water gas is generally made, omitting, however, preferably the impregnation of the gas by hydrocarbons. The battery is illustrated in vertical cross-section in Fig. 3.

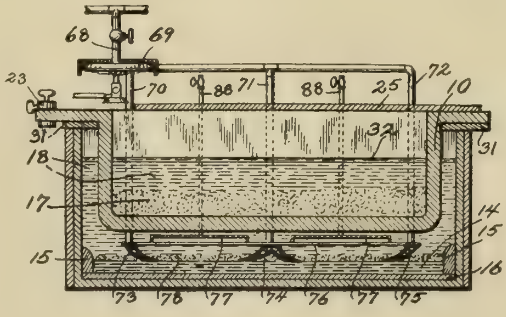


FIG. 3.—CELL FOR CONVERTING ENERGY OF FUEL INTO ELECTRICAL ENERGY.

It consists of a vessel made of some material unaffected by caustic potash, upon which rests by means of a flange the square, porous carbon vessel, 10. This vessel serves as the negative plate and contains the mercuric oxide, 17. A flat pan, 16, with a ledge, 15, rests on the bottom of the outer vessel and contains the metallic tin which constitutes the positive pole. The fluid, 18, in the inner and the outer vessel is a watery solution of caustic potash, kept at an elevated temperature. The solution is covered with a layer of paraffine, 32, to prevent absorption of carbon dioxide and oxygen from the air. The reducing gas is introduced through pipe, 68 [illegible] in pipe 89, and supplied to the cell by pipes 71, 73 and 75, which are provided with nozzle jets 72, 74 and 76. The diameter of the jets is preferably such that the gas is liberated [illegible] with [illegible] 78 as possible. The several pipes and jets are made of enameled copper. The [illegible] gas [illegible] pressure, passes through the coils and is heated to nearly the temperature of the coils, and is then delivered into the liquid through the distributing pipes and the jets, and reduces the stannous oxide to metallic tin. The metallic tin is thus recovered and the generation of the electric current continues uninterruptedly. The electromotive force of a cell of this kind is stated to be 1.03 volts, the current strength of a cell of the capacity of one gallon of liquid being 12 amperes, when the external resistance is .04 ohm. The gas which rises in the liquid before being oxidized accumulates in the inverted pans, 77, whence it escapes through pipes, 88. It is claimed as an advantege that the cell can be stopped easily by simply lowering the temperature. This stoppage may also be brought about by discontinuing the supply of gas, in which case the tin is gradually covered by a film of oxide which retards and finally almost entirely stops the battery reaction. The energetic action of the battery is largely accounted for by the acid properties of the oxide of tin, since even the more electropositive metal, which has no acid properties, would dissolve in alkali less rapidly, or not at all. Cadmium may be substituted for the tin, the latter being, however, preferred.

Galvanic Element or Battery.—Paul Brandt, Schöneberg, near Berlin, Germany. Patent 775,802, Nov. 22, 1904. Application filed April 20, 1904.

A small battery, as employed for pocket lanterns, etc., and consisting of a receptacle with several partitions. Each of the compartments contains a tubular cylinder of zinc, in the interior of which cylinders there are arranged carbon rods surrounded by a substance adapted to suck up a comparatively large quantity of the electrolyte. The electrodes in the different compartments are connected in series by strips of metal embedded in a layer of pitch, asphalt or the like at the bottom of each compartment.

Battery Case.—R. H. Wappler and F. H. Wappler, New York. Patent 777,457, Dec. 13, 1904. Application filed April 20, 1904.

The invention relates to the construction of a compact battery case for the use of physicians, etc., the cover and base of which case are connected by hinges and hold the various cells securely in place.

MISCELLANY.

Thermopile.—J. A. Lyons and E. C. Broadwell, Chicago. Patent 775,187, Nov. 15, 1904. Application filed Dec. 9, 1903.

Thermopile Elements.—Same inventors. Patent 775,188, Nov. 15, 1904. Application filed July 6, 1903. Renewed July 6, 1904.

The thermopile described in the first patent is heated by a Bunsen burner arranged in the center, and consists of a number of layers of concentric rings, the spaces between which are filled with alternating layers of negative and positive electrically-conductive and thermally-resistive substances. The positive elements are preferably composed of finely divided metallic copper, each particle of which is electroplated with bismuth. The negative elements are composed of cuprous sulphide alloyed with about 18 per cent. of metallic antimony for the purpose of increasing the electrical conductivity. The inwardly directed surfaces of the rings are convex and highly polished, while the outer convex surfaces are painted with lampblack. The object of the second patent is to obviate the use of metals or alloys in thermopiles, as an instance of which a pile is described consisting of bars or plates of a mixture [illegible] of [illegible] sulphide and lead sulphide, which are used alternately in conjunction with bars or [illegible]. To the latter an excess of phosphorus has been added so as to insure complete conversion of the copper into phosphide during the process of its formation and varying proportions of copper sulphide or tin telluride may be added to it.

Process of Manufacturing Ammonium Formate.—H. Pauling, Brandau, Austria. Patent 776,543, Dec. 6, 1904. Application filed April 2, 1902.

The process consists in subjecting a gas mixture containing hydrogen, nitrogen and carbon monoxide, together with steam, to electric non-luminous or brush discharges in the presence of porous contact substances, such as spongy platinum. Ammonium formate is then obtained according to the equation: $2N + 3H_2 + 2CO + 2HO_2O = 2HCO_2(NH_4)$. The presence of porous contact substances is essential, for no ammonium formate is formed if the mixture is exposed to non-luminous or brush discharges alone. Preferably Dowson gas or a mixture of water, gas and nitrogen is used, but instead of this a mixture of air and the gas mixture mentioned above may be used, in which case, however, the air has been previously exposed to electric discharges in order to transform the nitrogen present in it into nitric acid.

Process of Heating Air.—H. Pauling, Brandau, Austria. Patent 777,485, Dec. 13, 1904. Application filed August 12, 1902.

The process relates particularly to the production of oxides of nitrogen from atmospheric air by heating it to a very high temperature with a view of dissociating it. For this purpose the air or gas to be heated is conducted alternately from both sides into a chamber provided with a heat-generator, for which purpose electric arcs, struck between carbon electrodes, are utilized. The heating chamber is situated between two, regenerator chambers, patterned after those employed in open hearth furnaces. The cold air in the beginning of the operation is passed into the heating chamber and then travels through the checker work of one of the regenerators, where it parts with its heat. When this regenerator has been heated enough, the current of air is reversed so that the air to be treated passes through the regenerator before entering the heating chamber and thus arrives in the latter already in a highly heated condition. The other regenerator is heated in the same manner, the process being continued until the temperature of the regenerators nearly or quite equals that of the heating chamber. A diagrammatic plan of the apparatus is given in Fig. 4. It is made of the best fireproof material, adapted to withstand a temperature of 2,500° and comprises a heating chamber, 1, and two regenerators, 2 and 3, separated by the bridge, 4. The heat is furnished by a number of carbon electrodes, 5, which extend transversely into the furnace. Conduits, 6 and 7, lead into the lower parts of the respective regenerators, through which conduits the air under treatment alternately enters and passes out. Each regenerator is preferably divided into a number of compartments, 8, 9, 10 and 11. The course traveled by the air current from its entrance through the conduits, 6 and 7, to its exit through these conduits is indicated by the arrows I. and II. When the temperature desired has been obtained, the current is interrupted and the electrodes withdrawn, after which air can be driven in suitably reversed directions through the furnace for reaction purposes until the requisite high temperature does not exist

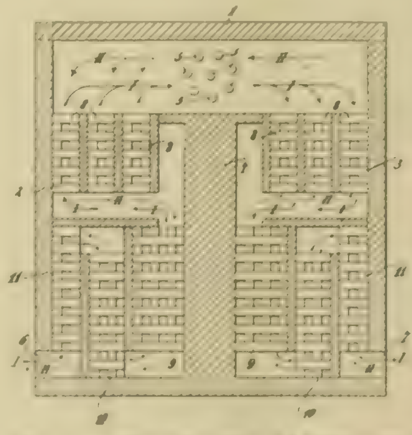


FIG. 4.—APPARATUS FOR HEATING AIR.

any more, after which the electrodes are reintroduced and the process repeated. It is stated that by this method the nitrogen and the oxygen of the air may be made to combine, forming nitrogen dioxide, which latter may be converted into nitric acid in any suitable manner.

Process of Manufacturing Nitric Acid.—H. Pauling, Brandau, Austria. Patent 777,486, Dec. 13, 1904. Application filed August 12, 1902.

The first step of the process consists in treating a certain quantity of atmospheric air by either ozonizing it by dark electric discharges, or by heating it to a temperature of 1,000° or 1,200° C., at which temperature nitric oxide is decomposed into nitric dioxide and nitrogen. The air thus treated is then subjected to electric-spark discharges, the amount of water necessary to convert the nitrous compounds formed into nitric acid being added by introducing steam and fresh air or hydrogen or gas mixtures containing hydrogen, such as water gas, after the electric-spark discharges have been interrupted. When a body of air at normal temperature is subjected to the effect of a discharge of electric sparks, nitric oxide, nitric dioxide and ozone are produced. Nitric oxide and ozone require considerably more energy for their formation than nitric dioxide. As, however, ozone cannot exist at temperature above 350° C., and nitric oxide splits up into nitrogen dioxide and nitrogen at approximately 1,000° C., neither one of the two can be formed if the spark discharge takes place in air heated above that temperature. The current is supplied by an induction coil connected with a dynamo; the ozonizing of the air is effected by dark electric discharges, the air being conducted through a receptacle enclosing a condenser, which consists of two parallel glass plates covered with metal on the sides forming the adjacent faces or those facing each other and connected with the induction coil. The air is then conducted into another receptacle and there subjected to electric-spark discharges, the apparatus being fitted, if desired, with the regenerative features described in the preceding specification.

Apparatus for Electrically Treating Gases.—K. Birkeland, Christiania, Norway. Patent 775,123, Nov. 15, 1904. Application filed June 15, 1903.

The apparatus is intended for carrying out the inventor's process for the production of nitrogen compounds from atmospheric nitrogen, described in Electrochemical Industry, Vol. II., page 399 and 507. In order to obtain the arc in the shape of a disc at right angles to the lines of force of the magnetic field, the inventor has found it advantageous in the case of currents of high voltage, *f. i.*, 10,000 volts, to have the poles of the electrodes at a distance of a few millimeters. If direct currents are used, there will be a great number of such arcs formed in succession, without any mechanical interruption of the current, the number being up to several hundreds per second. If alternating current is used, there will be two systems of arcs, one-half of them being formed on the under side, and the other on the top side of the electrodes. The electrodes are preferably provided with strips at their ends, these strips being preferably in the shape of arrow-heads. If a current of six hundred volts or less is used, either direct or alternating, it has been found desirable to arrange the electrodes in such a way that their poles approach and recede from each other in rapid succession, sufficiently to come into contact with each other. Various constructions are outlined for effecting this vibratory movement.

RECENT METALLURGICAL PATENTS.

Aluminium.

An attempt of reducing aluminium from alumina without the use of the electric current is made by H. S. Blackmore (United States patent 778,100, Dec. 20). He uses as reducing agent acetylene mixed with magnesium vapor. Porous aluminium oxide is heated in a retort to bright redness. A current of acetylene is then passed through molten magnesium, "regulated so that it will carry in suspension the magnesium vapor in about equivalent proportions." The acetylized magnesium vapor is then passed through the hot porous aluminium oxide. For the resulting reaction the author gives three equations. The first is

$$Al_2O_3 + C_2H_2 + Mg = 2Al + 2CO + MgO + H_2$$

This reaction is stated to develop sufficient heat to allow the following reaction to go on

$$2Al_2O_3 + 3C_2H_2 = 4Al + 6CO + 6H$$

The third equation given by the author is the sum of the first two

$$3Al_2O_3 + 4C_2H_2 + Mg = 6Al + 8CO + MgO + 8H.$$

It is difficult to say how well this process, if practicable at all, would work on a large scale in competition with the extremely simple electrolysis of alumina dissolved in a fluoride bath. The proposal is certainly interesting—even from a very restricted electrochemical standpoint, since the reducing agents used—acetylene and magnesium—are electrochemical products.

R. Fortun and E. Semprun (778,025, December 20) patent a solder for aluminium or aluminium alloys, including the following ingredients, silver, aluminium, zinc and tin, "at least a portion of the silver, aluminium and tin being phosphorized, and at least a portion of the zinc being sulfurated."

Copper.

A patent of P. Weiller and A. Weiller (775,548, November 22) refers to the separation from their ores of copper, silver, lead, mercury, and all other metals adapted to be precipitated from an acid solution by means of sulphureted hydrogen. The special features are the use of neither coal nor air blast in the process. "The reducing agent and fuel for generating the high temperature required is metallic iron, the oxygen being supplied by means of a powerful oxidizing agent which is added." The crushed ore is mixed with iron filings and saltpeter, the proportions of the latter two being preferably one to one; but if the ore contains combustible substances, such as iron sulphide or disulphide of iron, the quantity of iron-filings must be proportionately more than that of saltpeter. The mixture is placed in a small furnace and ignited, whereupon the metal is reduced and fused and descends to the sole of the hearth. If the ore requires a higher temperature or stronger oxidation for washing, then a chlorate is added to the mixture.

Gold and Silver.

There exist certain alluvial deposits, consisting for the most part of sand containing a greater or smaller percentage of magnetic iron oxide in a granular or pulverulent state, with which fine particles of free gold are associated, or to which they adhere in a manner which renders mechanical or hydraulic separation impracticable. T. J. Lovett (775,043, November 15) proposes to treat these deposits by wet magnetic separation. The magnetic iron oxide thus separated and carrying gold is subjected to lixiviation with cyanide of potassium. The gold is then precipitated in any suitable way, while the tailings from the leaching vats are "substantially pure magnetic iron oxide which may be dried if desired, and shipped in a granular state to the furnace, or it may be briquetted for smelting. The concentrated iron oxide in a granular or pulverulent state is in the best condition for electric smelting."

C. H. Rider (776,424, November 29) extract gold and silver from ores by crushing and treatment in a series of tanks, one-half of which are the leaching tanks, the other half the precipi-

[illegible] [illegible]. The tanks are connected by pipes running from the top of one tank into the bottom of the next one; the object is to pass the gases which are developed in one tank from its top to the bottom of the next tank for the purpose of agitation. The ground ore is first placed in the leaching tanks with a solution containing 3 to 5 gallons of sulphuric acid and 15 to 20 gallons of nitric acid per 100 gallons of water. This dissolves the content of silver in the ore. The solution is then removed from the leaching tanks to the precipitation tanks, and a fresh solution containing 5 gallons of nitric acid and 10 gallons of hydrochloric acid per 40 gallons of water, is brought into the leaching tanks to act on the undissolved ore. This dissolves the gold, and by adding this solution to that now in the precipitation tanks, an immediate precipitation of the silver is caused as silver chloride. The gold is afterwards precipitated by a saturated solution of ferrous sulphate. This brings the gold down in the metallic state.

A greater number of patents refer to details of construction of leaching or filtering apparatus. W. S. Jones (777,379, December 13) patents an ore-leaching vat with perforated pipes in the bottom for passing air into the solution, both for agitation and for supplying the oxygen required for the solution of the gold in the cyanide solution; the main feature is that the bottom of the tank is divided into sections which are separate and air-tight from each other laterally, and the air is introduced into each section independently of the others.

F. H. Long (775,405, November 22) effects a repeated circulation of the entire charge of mixed ore and cyanide solution in successive portions through a contracted channel by injecting into the channel an air-blast of sufficient force and volume to maintain the circulation of the charge, and to revivify the cyanide and clear the ore while the circulation persists.

J. J. Berrigan (775,414 and 775,509, November 22) passes the comminuted ore and the solution through a series of agitators, making use of centrifugal forces. The ore is first conducted by a conveyor into a descending stream of solution so that both enter the first agitator well combined and with a large amount of air intermingled with them. A revolving liquid ring of solution and ore is established in all agitators, and finally in the separator in which the enriched solution is separated from the ore by centrifugal forces. The thorough agitation, with its continuous introduction of oxygen into the solution, are intended to accelerate the solution of the precious metals.

L. E. Porter (773,221 and 773,222, October 25) patents two forms of slime-washer. By means of a revolving tubular shaft with lateral arms, he introduces compressed air into the tank for agitation; the same device is also used for introducing fresh wash-water. The enriched solution filters through gunny-sacking at the top of the tank and flows over the edge into a launder. The sands and slimes are discharged through a tube in the conical bottom of the tank.

D. C. Boley (774,736, November 15) employs a tank with a perforated bottom to which is secured a lining of textile fabric "by means of folds of the fabric itself, the edges of the fabric being secured to the sides of the tank, whereby overlying battens above the fabric are dispensed with and perforation of the fabric obviated." The tank is filled with the sands, slimes and cyanide solution, and air is introduced into it upwards through the bottom. When the gold is dissolved, the supply of air is stopped; the contents are allowed to settle, and finally fresh solution, containing no gold, is introduced through the bottom to replace the enriched solution.

H. R. Cassel (773,473, October 25, and 774,340, November 8) describes details of construction of filtering apparatus, by means of which the liquid is filtered off from the pulp by suction in a tank containing filter cells, while simultaneously the pulp is agitated between the cells.

Evidence of the increasing interest of Australian cyanide engineers in the use of filter presses, is a patent of W. A. Pritchard (776,084, November 29) for a pressure-filter for slimes. The mixture, as a whole, is fed into the lowest end of the main chamber of the apparatus and the liquids forced out through the filtering mediums, while the mixture during its process of exhaustion is concurrently conveyed upward by revolving propeller plates, and the undissolved residue is finally discharged at and from the top of the chamber.

ZINC.

C. S. Brant (775,359 and 775,360, November 22) patents two devices for preventing lead from coming into the spelter during the process of distillation. Fig. 1 shows one device in which the lead is separated from the zinc vapors by means of a filter bed or the like, composed of broken clay crucibles or other chemically inert non-combustible or refractory material.

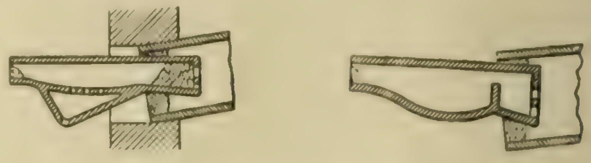


FIG. 1. FIG. 2.

DISTILLATION OF ZINC.

The zinc vapors produced pass through the upper aperture in the extremity of the nozzle. The filter bed arrests and throws down the lead, while it allows the zinc vapors to pass through to the condensing-chamber. The lead flows back into the retort by way of the lower aperature in the extremity of the nozzle. Fig. 2 shows a second device of the same inventor. In this case he provides a lead—interception chamber.

MISCELLANEOUS METALLURGICAL APPARATUS.

W. B. Simons (778,149, December 20) patents an apparatus for the treatment of pyrites, the object being to facilitate the driving off of the sulphur fumes for the manufacture of sulphuric acid. The apparatus consists of a series of superposed grates. The pyrites are introduced at the top and while burning are passed from one grate to the next. In this way, they are sufficiently agitated to effect complete combustion and cause the sulphur fumes to escape.

Frequently in the operation of blast furnaces there occurs a violent expulsion of gases from the top, caused either by an explosion of gases or by the slip of the charge. As a result, the fine ore dust is carried out in large quantities. J. Coyne (777,498, December 13) describes details of construction by which the dust thus driven from the furnace may be separated from the gases collected so as to be returned in a suitable form to the furnace.

U. Wedge (777,577, December 13) describes details of construction of a roasting furnace, the principal objects being the ready application or removal of the arms carrying the stirring blades and the provision of means for cooling these arms.

C. H. Repath and F. E. Marcy (776,085, November 29) patent details of construction of rabble arms and rakes for roasting furnaces so that any rake when broken can be removed and another one substituted without disturbing the other rakes or without first cooling the furnace.

S. V. Huber (777,509, December 13, and S. M. Guss, 775,563, November 22) patent valve mechanisms for furnaces.

C. Raapke (777,746, December 20) patents a reversible hearth converter "comprising an oblong converter, means to turn the converter about its transverse axis, a passage communicating with the interior of the converter and substantially at right angles to its longitudinal axis of rotation, chambers, communicating with each other and surrounding said passage, a blast-pipe connected with said chambers and with the bottom of the converter, the converter being provided with a suitable charging and with a suitable delivery opening, one of the said chambers being provided with a side-opening and suitable covers to close the said openings."

E. Kratochvil (777,112, December 13) patents a water-dispersing rotary device through which blast furnace gases are passed for the purpose of cleaning them for use in gas engines.

J. C. Fox (777,725, December 20) patents details of construction of cupels of bone-ash, as used by assayers. The base of the cupel instead of being made flat, as heretofore, is hollowed out or recessed slightly on the under side, so as to form a space between the bottom of the cupel and the floor of the muffle, the effect of this space being that the litharge, when it reaches the bottom of the cupel, cannot pass into the floor of the muffle, but will be caused to spread laterally until the full absorption of the cupel is reached.

E. Keller, A. Ferrell, and K. W. McComas (777,421, December 13) patent details of construction of assay-button droppers; E. H. Fosdick (773,203, October 25) patents details of construction of a blow-pipe.

SYNOPSIS OF PERIODICAL LITERATURE.

A Summary of Articles on Electrochemistry and Allied Subjects Appearing in American and Foreign Periodicals.

Industrial Applications.

Titanium.—A very long and interesting account of experiments on the production of titanium and titanium alloys from rutile and titanates in the electric furnace is given by W. Huppertz in Nos. 17, 18, 19, 21 and 22, 1904, of *Metallurgie.* The great difficulties involved in the problem are due to the very high affinity of titanium to nitrogen and carbon. The rutile used by the author was a product from mines in Virginia, U. S. A., and was of great purity, consisting of 96.26 per cent TiO_2, 2.26 per cent Fe_2O_3 and 1.42 per cent SiO_2. The author first tried to reduce TiO_2 by means of carbon in the electric furnace, but could obtain only a mixture of TiO, titanium, titanium nitride and titanium carbide. He then varied the conditions of the experiment in so far as TiO_2 was reduced simultaneously with the oxide of another metal, carbon being again the reducing agent. Though better results were thus obtained, yet it seems from his experiments that the only possibility to get in this way a metal free from nitrogen and carbon would be to produce a raw metal by Moissan's method and then to reduce it by means of TiO_2. Moissan obtained in this way a metal containing not more than 2 per cent C. The author suggests, however, to modify Moissan's method by carrying out the reduction by carbon in an atmosphere of hydrogen. The author then experimented with calcium carbide as a reducing agent. When he tried to reduce TiO_2 by means of calcium carbide, he obtained only titanium carbide and calcium titanate, but no metallic titanium. The result can be improved, however, if another metal, especially iron, is present. In this case he obtains an alloy which contains really metallic titanium, besides titanium carbide. The author then makes some remarks on the reduction of TiO_2 in the blast furnace, and then passes over to the use of aluminium as a reducing agent. The aluminothermic reaction takes place after the mixture has been ignited according to the equation

$$3TiO_2 + 2Al_2 = 3Ti + 2Al_2O_3.$$

However, the speed of reaction is small; moreover, the excess of heat is not sufficient for completely melting the titanium on account of the high temperature of reduction. The author, therefore, employs external heat by combining the aluminothermic reaction with electric heating. He concludes that by this method, if certain precautions are taken, any quantities of pure titanium may be obtained. Finally, the author employed an electrolytic process with calcium chloride as electrolyte, the titanium oxide being reduced cathodically by the calcium ions. The cell used for this purposse is the strontium metallic furnace illustrated in our Vol. I., page 104. By this method, he succeeded in preparing titanium in form of grains melted together, with a perfect metallic lustre and a brilliant white color. The metal thus prepared is absolutely free from nitrogen and carbon. In a flame, it gives a beautiful light effect, and could therefore be used in pyrotechnics. By the same method titanium alloys can be made, if, instead of TiO_2, titanates, for instance, of iron, copper, etc., are used.

Electric Furnace for Making Glass.—A continuation of the article by J. Bronn on the experiments made by him in continuation of those by Becker and Voelker with electric glass furnaces is given in the November issue of *Electrochem. Zeit.* In the experiments described in the present installment the heat for the smelting of the mass was obtained by radiation from arcs between carbon electrodes. The main trouble experienced was that carbon impurities came into the glass and colored it. He describes various experiments which he made to prevent this, but which were unsuccessful. He obtained the best results with an arc between two cored carbons. When the melting process was going on smoothly, the ammeter and voltmeter needles showed, nevertheless, strong oscillations. The reason was that he used really a combined arc and resistance furnace, since from each electrode, a discharge passed over to the glass, so that part of the current passed through the glass (similarly, as is the case with the Héroult steel furnace). The regulation of the electrodes was rather difficult. The article is to be continued.

Gold.—The *Engineering and Mining Journal,* of October 27 contained an illustrated article by W. E. Greenawalt on chlorination in Colorado. He states that 60,000 tons of ore are mined per month in the Cripple-Creek District, yielding bullion worth $1,901,800. Of these, 40,000 tons are treated by chlorination, 9,000 tons by cyanidation and 11,000 tons by smelting. He claims that with the application of chlorine, as generated by electrolysis, cyaniding will not be able to compete with it, even on the basis of economy. He discusses the various stages of the treatment, namely, sampling, bedding, fine crushing, roasting, chlorination in barrels, precipitation and chlorination of the barrel tailings. The usual barrel charge is said to be:

	Pounds.
Ore	20,000
Water	10,000
Sulphuric Acid (66° B.)	200 to 400
Bleaching powder (33½ per cent Cl)	100 to 200

The acid costs from 1 to 1.25 cents per pound, and the bleach from 2.5 to 3 cents per pound, delivered. Hydrogen of sulphide is used as precipitant.

P. Argall in the *Engineering and Mining Journal,* of November 24, criticises severely the estimate of cost given in the above article. He thinks chlorination cannot compete with cyanidation, and that the day of barrel chlorination is past. To this W. E. Greenawalt replies in the issue of December 15, maintaining his original position. The issue of December 1, of the same journal, contains a second illustrated article by W. E. Greenawalt under the title, "The New Chlorination." He gives a comparison of chlorination versus cyanidation in favor of the former. He points out that in the chlorination process there is no particular difficulty in precipitating the gold to a trace by means of hydrogen sulphide, charcoal or ferrous sulphate. He advocates strongly the introduction of the electrolytic process of making chlorine from common salt into chlorination plants. In connection with this it may be interesting to read the description of the J. E. Greenawalt elec-chlorination process in our January issue, 1904, page 24.

Gold and Silver.—An account of some electrolytic experiments with gold and silver sulphide ore is given by M. Vaygouny in the *Electrical Review* of November 5. He experimented with Tonopah ore, which was a grayish quartz ore, containing some galena, pyrite, calcite, silver sulphide and gold, with a gangue consisting mainly of quartz. It also contained a notable proportion of metallic iron introduced in the ore through improper grinding. The electrolyte used contains one to two per cent ferric chloride, 15 per cent to 20 per cent common salt, and one per cent hydrochloric or sulphuric acid. The percentage of extraction of the silver values is of about the same degree, namely, 95 to 97 per cent, whether the solution and ore mass is kept boiling for two or three hours, with constant stirring, or is left in the cold for two to three days, with only occasional shaking. It also does not materially affect the results whether the solution contains only one per cent ferric chloride, and is constantly chlorinated, or oxidized in some way and thus regenerated, or is made up of two or three per cent ferric chloride and is left without chlorination. This is so, however, only so far as the silver extraction is concerned. The thoroughness of extraction of the gold values depends entirely upon the thoroughness of chlorination. By prolonging the time of treatment in cold to three to four days the almost totality of the silver values can be got, provided they occur as sulphide, as chloride or as metal, though this latter is rarely the case in nature. By constant removal by electrolysis of the silver dissolved and consequent regeneration of the solution, and proper stirring of the ore mass, the time of treatment is very much shortened. In most cases, if the ore carries also much gold, a sufficient amount of this metal is also extracted, besides the silver, to pay a large share, if not the whole, of the expenses of the treatment, and this with no more care than is given to the extraction of silver alone. If it be desired to extract the gold values to anything like the thoroughness of the silver, then care must be taken that the solution never lacks chlorine. If this were the case, that is, if any chance were given for the ferrous chlorine, resulting from the action of ferric chloride upon the sulphides in general, to accumulate in the solution to any extent, then the gold, if at all dissolved, would be reprecipitated and would thus remain in the ore mass by the well-known secondary influence of ferrous salts upon dissolved gold chloride. The precious metals dissolved in the solution are precipitated by electrolysis. He finds that when a very small percentage of glue is added to the solution slimy deposit is mostly avoided, the deposit being generally perfectly solid. (This is quite analogous to the case of the Betts lead refining process). Carbon or graphite electrodes are stated to be most suitable.

Zinc.—A method of K. Kaiser for an improved leaching process of zinc from ores is mentioned in *Metallurgie*, No. 7, 1904. After oxidizing, roasting and suitably crushing, the ore is mixed with a sufficient quantity of a solution of zinc chloride so as to change the zinc oxide contained in the ore into oxychloride. The zinc chloride solution may be employed in any strength upwards of 25 per cent. $ZnCl_2$. Very soon after the addition, the mass dries and hardens. It is then again crushed and leached out hot with muriatic acid. This is claimed to give a very complete solution of the zinc contained in the ore, while the quantity of iron dissolved is small. The solution is then purified and electrolytically treated to obtain the zinc. The same issue contains an editorial note in which some doubt is expressed upon the full success of the method.

Separation of Tin and Lead from Tin-Lead Alloys.—The increased consumption of tin during recent years has caused a greater use of tin ores containing lead in tin smelting works, while in electrolytic detinning works, the tin of the tinned iron sheets is impure on account of the solder. Both these conditions result in the production of greater quantities of tin containing lead. L. Peetz discusses in *Metallurgie*, 1904, No. 14 and 16, at length, various processes for separating the lead from the tin. He concludes that the separation of tin and lead is practicable in a galvanic couple or short-circuited cell, in which solid lead oxide serves as cathode and the lead-tin alloy as anode with an alkaline electrolyte at a temperature of over 35° C. The tin from the lead-tin alloy passes into the solution, while the lead oxide is reduced to lead. If the plates of the tin-lead alloy are thick, it is necessary to remove at intervals the layer of spongy lead from the surface from which the tin has been leached out. The author recommends to use alloy plates of not more than 1 mm. thickness, since with thicker plates losses of tin are unavoidable.

Increasing the Speed of Copper Deposition.—It has long been known that the deposition of a metal from its salt by electrolysis depends essentially on the condition that a sufficient number of metal ions are always in contact with the surface of the cathode. For this reason stirring or heating are advantageous, since both means tend to bring new ions to the cathode. In *Electrochem. Zeit.* of November, S. von Maximowitsch describes another method which has the same effect. It consists simply in a horizontal arrangement of the electrodes, the anode being placed above the cathode. Since the solution of the copper salt when giving off copper to the cathode becomes lighter, the light solution rises upwards, and new fresh solution of the copper salt is carried to the cathode.

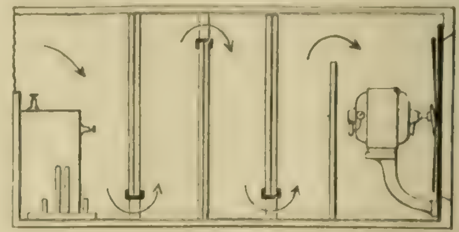


FIG. 1.—OZONIZER.

Ozonizer.—A new ozonizer devised by A. Rosenberg and shown in Fig. 1, is described in the Lond. *Elec. Rev.*, of November 11, as follows: Each element of the ozonizer consists of a thin sheet of highly insulating material, such as micanite, against which are applied, on either side, sheets of copper gauze having 40 meshes to the inch. These sheets are connected alternately to the two poles of the step-up transformer, which gives a potential difference of 4500 volts, this having been found by experiment to be the best pressure for the purpose. It will be seen that the current of air drawn through the ozonizer by the fan is constrained by the arrangement of the elements, acting as baffle-plates to pass over the whole of the ozonizing surfaces, which have an area of nearly 4 square feet in the apparatus in question. As at each corner of each mesh of the gauze the wire is necessarily bent so as to form an elevation, which may be regarded as a rounded point, there are 230,400 such points to every foot of the ozonising surface, or a total of over 900,000. From each of these a discharge takes place upon the surface of the dielectric; but, owing to the extreme sub-division of the discharge, there is no sparking whatever, and hence no nitrous compounds are formed. The apparatus is capable of ozonising 30,000 cubic feet of air per hour, with the expenditure of 60 watts, including the losses in the rotary converter. The actual production of ozone is said to amount to 250 grams "for an expenditure of 1 unit" (measuring 1 kw-hour).

THEORETICAL AND EXPERIMENTAL.

Aluminium Anode.—The valve action of the aluminium anode has been the subject of an investigation by F. Fischer, who studied especially the question whether this effect is due simply to large (ohmic) transition resistance or whether it is a polarization effect. The results of his investigation are published in *Zeit. f. Elektrochemie*, November 11. The author concludes from his experiments that the valve effect is primarily dependent on the temperature. Whenever the temperature rises so much that the liquid in the pores of the film begins to boil, the film is broken and the valve action is destroyed. With dilute sulphuric acid and electrodes of ordinary construction, the film breaks down under ordinary conditions at 30 volts. When, however, the electrode was constructed in such a way that the heat produced at it, may be easily carried away, the film did not break down until a voltage of about 70° was reached, the other conditions of the experiment being unchanged. Finally, if with the latter construction of the electrode, artificial water cooling of the electrode was employed, the film did not yet break down at 200 volts. The counter e. m. f., due to the anode film, is, according to the author, mainly due to the ohmic transition resistance of the film. It is a polarization effect only so far as in the case of ordinary oxygen polarization. The thickness of the films with 12, 24, 35, 72 volts was about 0.05, 0.10, 0.15 and 0.30 mm. respectively, so that the thickness is proportional to the voltage. The film obtained with 12 volts is transparent, that obtained at 24 volts still shows transparency, while those obtained with 36 and 72 volts are no longer transparent and show a shade of grayish green. For the great rigidity and compactness of the film, the author offers the following explanation based on electric endosmosis. Aluminium hydroxide, when suspended in water, migrates toward the anode on account of endosmosis. Consequently a paste of water and aluminium hydroxide gets drier at the anode and gets more wet at the cathode.

Electrolysis With Alternating Currents.—An account of a rather full investigation on this subject by A. Brochet and J. Petit is published in *Zeit. f. Elektrochemie,* December 2. One of the chief results of their experiments is the proof that the effect of a variation of the frequency is not always such as has been supposed. In some cases the efficiency decreases the higher the frequency. In other cases the efficiency increases with increasing frequency, passes through a maximum and then decreases. The same is true for the effect of the current density. The frequency increases quickly with the current density, passes through a maximum and then decreases more or less quickly. When alternating-current electrolysis is considered, the general results cannot be predicted, but must be determined experimentally under special consideration of frequency, current density and the nature of the electrodes. The authors then discuss the question why some metals (Cu, Zn, Ni, Co) are dissolved in KCN as well by direct current as by alternating current; other metals (Ag, Cd, Hg) are dissolved by direct current, but not by alternating current; still other metals (Fe, Pt) are dissolved in KCN by alternating currents only. The authors point out that Cu, Zn, Ni, Co, are-dissolved under the influence of alternating current on account of the ease with which they are anodically dissolved by direct current and the difficulty with which they are deposited cathodically. Ag, Cd, Hg are easily deposited cathodically, and are, therefore, not found to dissolve under the influence of alternating currents. Pt, when mechanically or chemically powdered disolves spontaneously in cyanide solution. Fe is dissolved by the influence of alternating current, but shows quite a distinct and special behavior.

Platinum Salts Made by Alternating-Current Electrolysis.—The fact mentioned above, that platinum dissolves as a result of electrolysis with alternating current, and not with direct current, is made use of in a paper in *Zeit. f. Elektrochemie,* December 2, by A. Brochet and J. Petit, for producing the following reaction:

$$Pt + 2Ba(CN)_2 + 2H_2O = Pt(CN)_4Ba + Ba(OH)_2 + H_2.$$

$Pt(CN)_4Ba4H_2O$ is the salt which has become of special interest during recent years on account of its use for luminescent screens such as used in the study of X-rays and radioactive phenomena.

The Effect of Stirring on Metallic Deposition.—A note by R. Amberg on this subject is published in *Zeit f. Elektrochemie,* of Nov. 4. He claims that all facts observed concerning the accelerated deposition of metals has an effect of revolution of the electrodes which can be explained by Nernst's "theory of the reaction velocity in heterogeneous systems, and by the experimental investigations of Brunner on this theory. According to Brunner, three stages of the phenomenon are to be distinguished in an electrolytic reaction: first, diffusion of the metal towards the boundary surface; second, chemical reaction, namely, discharge of metal irons; third, passage from the liquid into the solid phase which must take place very quickly. If the second process is very slow compared with the diffusion, artificial stirring cannot have any influence on the speed of the deposition, since even without stirring, a sufficient number of ions diffuses to the cathode. For a medium speed of the second process it will be possible to find a limit of the speed of rotation of the electrodes up to which a beneficial effect is observed; this case appears to exist in the electrolytic reduction of certain organic compounds. Finally, it is possible that the second reaction occurs with practically infinite velocity; in this case stirring will always improve the velocity of deposition. The latter case appears to exist for the deposition of metals.

Molecular Attraction.—A former paper by J. E. Mills on this subject was abstracted in our Vol. 2, page 370. The author endeavored to show that the attraction between molecules varies inversely as their square of their distance apart, and does not vary with the temperature, so that the law of gravitation is assumed to hold between the molecules of a substance. On this assumption, he derived an equation according to which the ratio of the difference of the latent heat of vaporization minus the energy spent in overcoming external pressure to the difference of the third roots of the density of the liquid and of the vapor should be constant. The author contributes a further paper on this subject to the *Journal of Phys. Chemistry,* December. He gives some further facts confirming the correctness of the latter equation for normally constituted substances. He also shows that the equation is applicable with equal exactness in the immediate neighborhood of the critical temperature. He deals at some length with an equation of Crompton, discusses the conditions at a critical temperature and deals with the variation of the latent heat of vaporization with the temperature.

Conductivity of Very Dilute Solutions.—Measurements of the electrical conductivity at 18° C. of very dilute (0.0001 to 0.002 normal) hydrochloric acid and nitric acid solutions are described in a paper by H. M. Goodwin and R. Haskell, published in the December issue of *Physical Review.* These very dilute solutions are made by adding to a known weight of water successive portions of 0.01 normal acid. It was, of course, most important to eliminate the effect of the impurities of the water, and two methods of computation of the equivalent conductivity were used by the authors. In both of these, the assumption is involved that the total effect of the acid and impurities on the conductance of each other is produced when a relatively small quantity of acid has been added to the water. The experiments themselves furnish evidence of the correctness of this assumption. In the first method, the increase of the specific conductance over this initial value, divided by increase of concentration, is regarded as the equivalent conductivity at the higher concentration. In the second method, such a quantity, constant at every concentration, is added to the observed values of the specific conductance as will cause the maximum in the equivalent conductivity values calculated therefrom to occur at the lowest concentration at which the total influence of the acid and impurities on each other has been produced. In spite of considerable variations in the observed conductivities due to the use of different samples of water, the corrected values of the equivalent conductivity derived from different experiments, and those computed by the two methods, agree well with each other. New determinations were also made at higher concentrations with hydrochloric acid. The equivalent conductivity of hydrochloric acid and nitric acid solutions for indefinite dilution are given as 380.1 and 377.0, respectively.

Conductivity of Non-Aqueous Solutions.—The December number of the *American Chemical Journal* contains a paper by H. C. Jones and C. G. Carroll on the conductivities of certain electrolytes in mixtures of two solvents. The authors discussed especially one striking phenomenon, that is, that the curve, giving the conductivity as function of the proportion in which the two solvents are mixed together, shows a minimum, so that the conductivity at this point is decidedly less than the corresponding conductivities in the pure solvents. The authors extended the investigations of Zelinsky and Krapiwin, and of Jones and Lindsay, and show the occurrence of the minimum in conductivity in three substances—cadmium iodide, sodium iodide and hydrochloric acid in mixtures of methyl alcohol and water. The dissociation—as determined from conductivity—of sodium and potassium iodide and potassium bromide in 50 per cent methyl alcohol was found to be greater than that in water at the corresponding dilution. The phenomenon of the occurrence of the minimum is shown to depend primarily upon the decrease in fluidity which results when the liquids are mixed. The hypothesis of Dutoir and Aston is proven quantitatively for certain salts in three solvents—water, methyl alcohol, and ethyl alcohol. The hypothesis of Kohlrausch (formation of an atmosphere of the solvent around the ions in solution) is shown to hold for binary electrolytes in methyl and ethyl alcohols. A hypothesis correlating conductivity, association, and viscosity (or fluidity) is proposed, and is shown to hold for all the cases available for discussion.

Electrolysis of Potassium Acetate.—A reply to the recent paper of H. Hofer and M. Moest (mentioned in our December issue, page 501) is published by F. Foerster and Piguet in *Zeit. f. Elektrochemie*, December 2.

Batteries.

Formation of Storage Battery Plates.—Among the quick formation methods of storage battery plates, that of Lucas with perchlorate is well known. In the *Centralblatt f. Accum.* Schmidt-Altwegg describes some experiments which seem to indicate that chlorate is equally suitable. He used an electrolyte of 1.06 specific gravity, consisting of chemically pure sulphuric acid, dissolved in water, with an addition of one per cent potassium chlorate. Positive plates formed by this method are stated to be specially resistive against the bad effects of discharges down to low voltages.

Storage Battery.—*Elektrochemische Zeit.*, of November, contains an article by G. Hommel, giving the results of tests of the "climax" accumulator, made by a German Company. The negative plate is pasted, while the positive is a Planté plate, the large active surface being obtained by pressing out of a solid lead plate, mechanically, a great number of vertical ribs. The remaining central core is 2 mm. thick, while the thickness of the whole plate, including the ribs, is 13 mm. The results of tests are given in forms of diagrams and tables.

Carbon Cell.—In our January issue, 1904 (Vol. II., page 30), we described a carbon cell by J. H. Reed, and in the same issue, page 18, C. J. Reed gave a critical discussion of this cell. *Iron Age*, of December 8, contains now a fully illustrated article from the pen of S. D. V. Burr, under the title "The Dynelectron."

Metallurgy

Iron and Steel

Cementation.—Georges Charpy describes in the *Iron and Steel Magazine* for October, some very illuminating experiments on the carbonization of iron by cementation. Filings of soft steel were kept in melted potassium cyanide, and samples taken out at intervals tested for carbon. The amounts found were, at the start, 0.00 per cent, after 48 hours 4.50, after 85 hours 6.72, after 110 hours 6.72. Since the formula Fe_3C calls for 6.67 per cent of carbon, it is quite evident that 85 hours' treatment converts the iron entirely into this carbide—cementite—and chemical tests further confirmed this, the material being dissolved completely in acids, and containing no trace of graphitic carbon. Mr. Charpy also carburized steel by heating it in a current of pure carbonic oxide. Below 750° C., there is free carbon deposited on the metal, while up to 0.28 per cent was found in the metal itself; above 750° no free carbon was deposited, but corban was absorbed up to 0.72 per cent in two hours' treatment. These experiments throw considerable light on the *rationale* of the cementation process, showing how carbonic oxide, by becoming carbon di-oxide, deposits carbon in the interior of the soft iron which it permeates.

Shrinkage.—The cause and the effect of this phenomenon are of vital importance to the scientific foundryman, and are discussed in the October number of the *Iron and Steel Magazine* by that veteran foundryman, Mr. Thomas D. West. As explained in the article, the practical man calls *shrinkage* the reduction in bulk of the metal from the time it is poured until it assumes the solid shape, while he uses the term *contraction* for the reduction in size after solidification takes place, while cooling down to ordinary temperatures. The particular phase of shrinkage discused by Mr. West is the strange fact that when castings are made from pig iron, remelted in a cupola or air furnace, the higher the silicon present, the less the shrinkage, while when making castings from liquid iron, taken direct from the blast furnace, directly the reverse occurs. The proposed solution of this riddle is that the direct metal carries carbon in the graphitic state, while melted, and is expanded in volume thereby, while remelted iron contains only combined carbon when molten, and since both contain the same amount of graphitic carbon when set, the former shrinks the most. It appears to the reviewer that the *explanation* is more mysterious than the facts themselves.

J. E. Johnson, Jr., in the November number of the same journal, discusses Mr. West's proposition and advances the much more reasonable explanation, that the difference is due to the facts that remelted iron contains at least 0.05 per cent more sulphur, and that it is not so hot as direct iron, and has not been so hot.

Nickel.

An interesting occurrence of nickel in an unusual form is described by Herr Stören in *Metallurgie* for September 22. The ore occurs in large quantities at Evje, in Norway, and may be described in general as a nickeliferous pyrite. It consists really of a mixture of pyrite with 20 to 30 per cent of pyrrhotite (magnetic pyrites), and smaller amounts of pentlandite. The pyrite contains 10 to 12 per cent of nickel, the pyrrhotite only 2.5 per cent, the pentlandite 27 to 33 per cent, the mixture 4.5 to 9 per cent. The low percentage in the magnetic pyrites is the particular difference from our Sudbury ores, together with the unusual amount in the pyrites itself.

Zinc.

The Industry in America.—Walter R. Ingalls, the author of the best treatise on zinc ever written, contributes to Dr. Borchers' *Metallurgie* of August 22 a very timely review of the present prospects of this industry in America. No one is better fitted than Mr. Ingalls to discus this topic, and the article, though short, is full of information and suggestion. One of the principal items discussed is the state of the industry in the Missouri-Kansas fields, where undoubtedly the cheapest zinc-ore smelting in the world is being done. The preëminent advantage of thes localities is the proximity of rich zinc ore, and an abundant supply of natural gas. The amount of gas used for smelting a ton of ore has never been accurately measured, but is estimated as 1100 to 1200 cubic meters (38,500 to 42,000 cubic feet), which costs, on the average, 84 cents. However, there is already beginning to be felt a shortage of gas in this district, and some works, as at Pittsburg, Kansas, are already being changed to burn coal or producer gas. Only four zinc works in the United States manufacture sulphuric acid from their waste roasting-furnace gases, and the necessity of economizing will undoubtedly force many more into this manufacture in the near future. It appears likely, also, that the centers of zinc production may follow the gas fields, rather than the sources of ore supply.

In the same journal for September 22, Mr. Ingalls continues his observations. He calls attention to the carelessness of chemists in overlooking cadmium in the analysis of zinc, since the Joplin ores contain an average of 0.3 to 0.4 per cent of that metal, which may be concentrated to almost 1 per cent in the first zinc drawn from a retort. The mixed zinc concentrates from the Colorado mines are next spoken of. They are treated to a large extent in Kansas—not mixed, however, with the Kansas ores—and by keeping the temperature of the retorts lower than usual, at the cost of not extracting so large a proportion of zinc, the injurious slagging of the retorts by the lead and iron gangue is largely avoided. At one works, these retort residues are smelted down in a shaft furnace to work lead, to recover their lead, gold and silver values. Mr. Ingalls predicts a considerable extension, in the near future, of the electromagnetic and electrostatic concentrating treatment of these and other zinc blende ores.

In the same journal for October 22, there is described what may prove the long-looked-for solution of the difficulties besetting the making of durable zinc distillation retorts. The German patent, 154,536, of March 27, 1901, describes a retort made of a mixture of carborundum, with fire clay as a binding material. Tests have shown that the silicon carbide is not attacked by the zinc vapors, and does not recrystalize at the temperature of the retort furnace, so that the retorts remain dense and strong after 100 days' continuous use. The inventor states that the retorts have better heat conductivity than the ordinary fire clay ones, last much longer and furnish a better output of zinc from the ores treated.

Sodium.

Amalgams.—A Schüller, in *Metallurgie* for October 22, furnishes us with the first complete and satisfactory knowledge of the possible amalgams of sodium ever published. His investigation was a systematic study of the cooling curves of alloys of these two metals, a diagram being constructed with abscissæ representing the change of composition from Hg_{100} to Na_{100}, and all compositions between of the general formula Na_4Hg_{100-4}. The actual abscissæ represent, therefore, what are called the "atomic percentages" of sodium present. This method of presenting the results has no theoretical advantages over that of recording simply weight percentages, and has the disadvantage of rendering the curve not directly comparable with the usual curves plotted on the more natural and usual system. The author is to be commended for having determined and plotted on his diagram the points of arrest due to solidification of eutectics, as well as the real melting points, and, further, for having added to these horizontal eutectic lines, vertical strokes representing by their length the relative times during which the eutectic temperature remained constant.

Starting with pure mercury, the first eutectic occurs at 0.33 per cent of sodium, with a melting point of — 48° C. (These quantities are estimated from the diagram, the actual experimental figures are unfortunately not given numerically.) With increasing sodium content the melting point rises regularly, until another eutectic appears at 2.45 per cent sodium, melting point 160°, indicated by a knick in the curve of melting points. Thence, the melting points rise nearly uniformly to the very clearly marked maximum of 360°, corresponding exactly to the compound $NaHg_2$, with 5.44 per cent of sodium. Above this, the melting points fall, with a fair regularity to a second well-defined eutectic melting at 20°, and containing 39.46 per cent of sodium. From this point the curve rises nearly linearly to pure sodium, melting at 95°. The above are the only certain conclusions to be drawn from the investigation. A study of small knicks in the melting-point curves leads the author to suspect the presence of $NaHg_4$, $Na_{12}Hg_{13}$ NaHg, Na_3Hg_2, Na_5Hg_2 and Na_3Hg, but the evidence is not satisfactory.

One theoretical conclusion to be drawn from this study is the observation that true chemical compounds are invariably maximum on the melting point curves, and the indeterminate chemical compounds in indefinite proportions, which constitute an ordinary solution, according to Prof. Kahlenberg's views, are characterized by descending or minimum melting points.

Precious Metals.

Gold.—Two extended papers by M. Merz are published in *Oesterr. Zeit. f. Berg-u. Hütten-Wesen*, 1904, Vol. 52, pages 59, 70, 86 and 99. And in *Metallurgie*, 1904, No. 8, 9 and 10, on the treatment of the gold selenium silver-ores of Lebong-Douck on Sumatra. These ores contain the precious metals in extremely fine distribution. While the latter dissolve well in cyanides if sufficiently crushed, they have a tendency of forming slimes which can be separated only with difficulty from the sands. The author concludes that the ore is a very special one, and that its treatment will have to be different from that employed in South Africa or anywhere. From his experiments, it seems that the decantation process would probably yield the best results.

Silver Coins.—J. W. A. Haagen-Smit describes in *Metallurgie*, 1904, No. 10, a new method for accelerating the process by which in mints the surface of silver alloys used for coins is whitened.

Miscellaneous.

Gas Engines.—An anonymous contributor to the December number of the *Iron and Steel Magazine* gives some interesting up-to-date facts concerning these latest assistants in blast furnace economics. Instances are cited of single engines driving blowing cylinders 80 inches diameter by 60 inches stroke, and delivering 30,000 cubic feet of air per minute at 25 pounds' pressure per square inch. It is only five years since the first small gas-blowing engine was put into operation. The efficiency of such engines, under the varying loads found in practice, may be safely counted on as 25 per cent, which is 150 per cent advance on the best steam boiler and engine plant. The first cost of the plant is somewhere about 50 per cent more than the steam engine plant, but this reduces only slightly the great saving due to the greatly increased efficiency.

If the gas is made in producers, the statement is made that an entirely satisfactory producer, using bituminous coal, has not yet been devised. The Loomis, Mond and Duff, and Schmidt producers are reported as doing fairly well. The gas is always cleaned, prefereably by a water spray in a centrifugal fan. The Cockerill and the Nuremberg are the preferred types of gas engine for blast furnace work.

Pyrometry.—The exact control and measurement of the temperature in a metallurgical process is often of most decisive importance for the economy of its operation. The December issue of *Physical Review* contains a full discussion of radiation pyrometry by C. W. Waidner and G. K. Burgess. After a general exposition of the fundamental principles and methods of radiation pyrometry, the authors discuss in detail the construction of the following instruments: LeChatelier, Wanner, Holborn and Kurlbaum, Morse, Féry. The Holborn-Kurlbaum pyrometer is shown in Fig. 2, where the tip of the filament L of the four-volt incandescent lamp is set to the same brightness as the object whose temperature is sought, by means of a rheostat. When the filament disappears against the bright background, the current through it is a measure of the temperature sought, when the lamp has been calibrated. One or more red glasses are placed before the eyepiece as the temperature rises above 900° C., and for extreme temperatures, above 1500° C., absorbing glasses or mirrors are inserted in front of the objective. In the Féry thermoelectric telescope shown in Fig. 3, radiation from the source whose temperature is sought is focussed by fluorite lens *F* on one junction of a minute iron-constantan couple in series with a potential galvanometer, whose deflections are proportional to the energy received by the couple, and hence to the fourth power of the aboslute temperature, according to the Stefan-Boltzmann law, if the radiating source is a black body. This pyrometer, since its indications are given by a pointer over a scale, is readily made recording, a desideratum in many pyrometric investigations. The authors conclude that no one instrument of those discussed can claim a decided superiority as to precision, certainty and ease of calibration, permanence of indications and range. All the instruments are capable of one per cent in temperature measurements in the ranges for which the yare adapted.

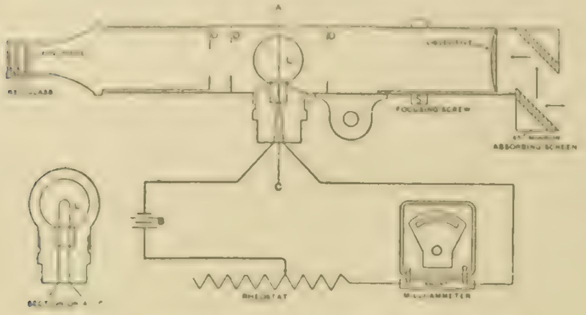


FIG. 2.—HOLBORN-KURLBAUM PYROMETER.

The authors conclude their paper by a few remarks on the measurement of very high temperatures, such as produced in the electric furnace, in the Goldschmidt thermit process, etc. Attempts are being made by Nernst and others to estimate these high temperatures by means of chemical phenomena taking place at high temperatures, but this work is still in a preliminary state. For this purpose, therefore, recourse must be had alone to the extrapolation of the laws of radiation which have been verified throughout the range of measurable temperatures. Lummer and Pringsheim have recently taken a single set of observations on an electrically heated carbon tube in an atmosphere of nitrogen, using three radiation methods: photometric (Wien's law); spectrophotometric, and total radiation (Stefan-Boltzman law), the results agreeing to 20° at 2300° C. absolute. From the work of the present authors, it would seem that the radiation laws are still in agreement at the temperature of the arc. Their mesurements have given as the black body temperature of the hottest part of the positive crater 3690°, 3680 and 3720° absolute, as determined with the Holborn-Kurlbaum, Wanner and LeChatelier pyrometers, based on the extrapolation of Wien's law. Féry gets for this temperature 3760° by a method based on Stefan's law. On the basis of these experiments it would seem that the several laws of radiation are in quite satisfactory agreement at the highest attainable temperatures, and thus serve to define the same scale of temperatures.

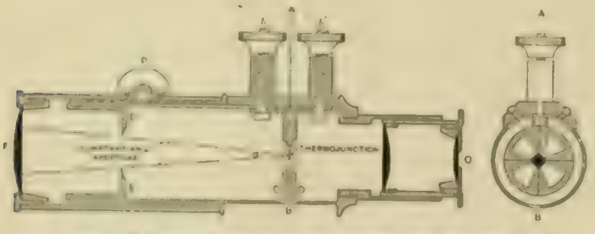


FIG. 3.—FERY THERMO-ELECTRIC TELESCOPE.

Book Reviews.

Modern Electro-Plating. By J. H. Van Horne, Chicago: George K. Haslitt & Co. Price $1.00.

This little work of 189 pages and 27 figures, will be a valuable asset in the hands of beginners and platers in country towns. It describes how the work is done and deals with the principles. The book contains a large number of formulæ and other useful information helping the plater out of possible difficulties.

THE OCCURRENCE OF ALUMINIUM in Vegetable Products, Animal Products, and Natural Waters. By C. F. Langworthy, Ph. D., and Peter T. Austen, Ph. D. New York: John Wiley & Sons, 1904. 168 pages. Price $2.00.

This is an extremely laborious and commendably complete compilation of the literature of this subject, giving in each case the reference whence derived. It is, however, more than a mere index of titles and catalogue of analyses; it gives, in many cases very satisfactory abstracts of the articles, enough to inform the reader of the important items of collateral information. The information in this book will be of first importance to anyone making a thorough study of the presence of alumina in soils, mineral waters, articles of food springing from the soil, fertilizers, or, in fact, of agricultural and physiological chemistry in general.

A TEXT-BOOK OF STATIC ELECTRICITY. By Hobart Mason. New York: McGraw Publishing Co., 1904. 155 pages. Price $2.00.

This work sets forth in easily understood terms the basic principles of static electricity. The subject is treated under the headings of I., General Phenomena; II., The Electrostatic ield; III., Capacity; IV., Experimental Measurement of Capacity; V., Instruments Used in Electrostatics; VI., High Potential Static Generators. A very instructive chapter might have been added on "The Application of Static Electricity," which would have clinched many of the principles in the student's mind by showing their usefulness when intelligently handled. However, the technical electrochemist, if he masters thoroughly the principles here so clearly explained, will be able to recognize the ideas and principles utilized in the technical applications of static electricity, and may even have the satisfaction, through this exercise, of foreseeing some applications not as yet discovered.

THE STUDY OF THE ATOM; or, the Foundations of Chemistry. By F. P. Venable, Ph. D., D. Sc., LL. D., Easton, Pa.: Chemical Publishing Co., 1904. 290 pages. Price $2.00.

The treatment is strictly chronological, and therefore strictly logical, because it thus necessarily traces the various stages of the natural development and evolution of the atomic theory. The respective chapters treat of the views of the ancients as to the nature of matter (34 pages), from the Greek philosophers to Dalton (41 pages), the atomic theory (38 pages), the relative weights of the atoms (45 pages), the periodic or natural system (33 pages), studies in affinity (28 pages), valence (27 pages), and recent theories of molecular and atomic constitution (34 pages). It thus appears that the whole field is very thoroughly covered, and the treatment well-balanced; the author is to be particularly commended for avoiding the pitfall which a less experienced teacher might easily have fallen into, viz., that of giving undue prominence and weight to the most recent hypotheses and speculations. Professor Venable has treated these as impartially and as judicially as the rest, and in so doing has increased very much the value of the work. There is no educated person who could not derive both pleasure and profit from a perusal of this book.

A Systematic Handbook of Volumetric Analysis, or the quantitative estimation of chemical substances by measure applied to liquids, solids and gases. By Francis Sutton, F. C. S., Ninth Edition, revised and enlarged. Philadelphia: P. Blakiston's & Co., 1904. Price, $5.00.

This standard work is so well and favorably known to chemists that no introduction to the name and character of the book is needed. We will, therefore, briefly note the specially valuable features of this new ninth edition.

Part I., devoted to general principles, discusses the methods of calibrating and comparing the burettes, flasks and measuring instruments, of normal solutions and the calculations needed in their preparation and is very clear and complete.

Part II., upon alkalinity and acidimetry, begins with an account of indicators, and these are classified according to the conditions of sensitiveness, and the limitations of individual indicators fully set forth. The titration methods for alkaline and alkaline earth salts are followed by a series of examples of technical examination of some alkaline compounds found in commerce or occurring in course of manufacture, such as soda ash, salt cake, soap, sulphate of ammonia, and ammoniacal gas liquor. This latter subject is especially treated in a very complete way and includes all the important constituents in this complex mixture. The Kjeldahl method so largely used for nitrogen in a wide range of organic compounds, is well illustrated and described, with mention of the most recent modifications. Under acidimetric methods we notice the determination of boric acid in milk, butter, and other foods, the determination of carbonic acid gas in waters, in aerated beverages, and in air, the analysis of commercial tartrates and argols, and of citric acid in lime and lemon juices.

Part III., devoted to analysis by oxidation or reduction, covers the use of permanganate of potassium, of chromic acid, of iodine and thiosulphate, of arsenic acid and iodine, while Part IV. covers the use of silver nitrate solutions and precipitation methods. Part V., covering over 250 pages, or nearly half the space in the book, gives in detail the application of the foregoing principles of analysis to special substances, and includes the whole range of inorganic and organic compounds which can be analyzed by the application of volumetric methods. A few examples from this part of the book will show the wide range of usefulness of the work. Thus, on pages 173-175, we have a variety of methods for the estimation of chlorine in mixtures of chlorides, hypochlorites, and chlorates, a very important problem for the success of electrolytic work at times; on pages 182-183 the methods for the estimation of chromium in ferro chrome and chrome steel; on pages 254-274 the estimation of nitrates and nitrites under a great variety of conditions and in various mixtures, such as soils, air, waters, etc.; on pages 308-322, the methods for the determination of sugars by the use of Fehling's solution, by mercuric cyanide and other reagents; on pages 351-362, a variety of methods for the determination of zinc in mixed solutions; on pages 368-370, the methods for the determination of formaldehyde; and on pages 378-393, the volumetric methods used in the analyses of oils, fats and waxes, including specially the analysis of butter, genuine and adulterated.

In Part VI. we have the special applications of the volumetric system to the analysis of urine, potable waters, sewage, etc., presented in something over 100 pages. The portion devoted to water and sewage analysis is especially valuable and includes numerous illustrative analyses, showing the varying conditions of the problem of the purity and contamination of water supply.

Part VII. is devoted to the volumetric analysis of gases and covers a full account of the older methods of Bunsen and others, using mercury and the straight Endiometer tube and the new methods of Hempel, Bunte, and Lunge, in which absorption pipettes of various forms are used, as well as endiometer tubes or gas burettes and greater rapidity of action is attained.

The book is indeed a systematic hand-book, as its name states, of all methods which allow of the more rapid analysis which can be carried out by measure as distinguished from the slower gravimetric methods, and is of the greatest importance to the technical chemist or analyst. The book is well printed, and contains a number of good illustrations, but for a laboratory reference book in frequent use ought to be put in a stronger binding.

The Pebble Tube Mill in Metallurgy and Especially in the Cyanide Process.

If a chemical reaction between different substances is desired, it is self-evident that the two substances should be in intimate contact. If one of the two substances is a solid, the logical conclusion would be to reduce the solid to as fine a powder as possible. It is most interesting to note that in gold metallurgy cyanide practice has now come to the same conclusion.

It is well known that former practice in stamping and grinding endeavored to avoid the production of much slime. Quite recently, however, the tendency has become to reduce the ore as fine as possible, with the resulting better extraction of the gold from the ores. A paper by two eminent gold metallurgists, Messrs. Charles Butters and E. M. Hamilton—both well known to the readers of this journal—is most interesting in this respect. This paper was recently delivered before the Institution of Mining and Metallurgy in London, and dealt with the cyaniding of ore at El Oro, Mexico, and especially with the regrinding of sands. The authors have made a very thorough study of the influence of the degree of fineness upon the extraction and upon the best method of pulverizing the ore to a very fine condition.

One chief result is that the extraction becomes better and better with increasingly fine grinding; this "points to the theory that if the whole could be reduced to an absolutely amorphous and impalpable powder, there would be nothing to hinder a total extraction of the value." One other chief result is that the machine which affords the best conditions for the grinding is the tubular flint mill, and that it is likely that much will be heard from the machine in the near future. Since the results obtained by Messrs. Butters and Hamilton were similarly obtained by others, some notes on recent developments and improvements in the construction of pebble mills should be of special interest.

Chemical manufacturers, makers of colors and drugs, etc., have recognized for a great many years that the pebble mill is the most economical and effective machine, when hard materials are to be reduced to a fine powder. In fact it has been found that the product of these machines contains at least 50 per cent more extremely fine particles than a similar quantity reduced in any other existing form of grinding machine.

For industries where large capacities on one class of material are desired it has become necessary to modify somewhat the design of the old pebble mill. Its successor is the modern pebble tube mill. This differs from the pebble mill, in that the material is fed into it at one end and discharged at the other, thus making it a continuous operation, the fineness of the finished product being regulated by the feed. The slower the material is fed, the longer it is subjected to the action of the pebbles and the finer the discharged product will be, whereas the faster it is fed, the coarser it will be discharged, so that a product of 100 mesh, 150 mesh or 200 mesh can be produced with the simple adjustment of the feed.

Up to the present time, the Portland Cement industry has been the largest user of the tube mills in this country, but most of this material is not reduced finer than so that 97 per cent will pass a 100-mesh sieve, but the Abbe tube mill, which will be described more in detail further below, has been used for reducing such materials as raw flint, pumice stone, feldspar, plumbago, talc, etc., to finenesses of 150, 160, 170 and [illegible] mesh, showing that it is well adapted to that class of work.

The tubular pebble mill has undergone considerable perfection during recent years, but had still one drawback. This was the trouble caused by the feeding and discharging mechanism of the machine. There are generally required for feeding a separate driving arrangement, stuffing boxes to stop it from leaking, the use of a conveyor, which is liable to wear out, clog up or break, while similar troubles exist at the discharge end of the tube. These troubles are avoided in the very simple and interesting "Ideal" spiral feed and discharge covered by patents granted to Mr. Max F. Abbe within the past year.

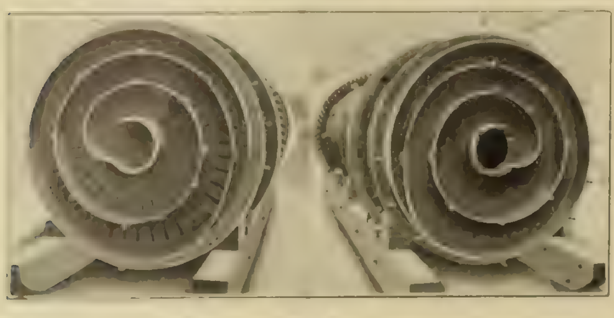


FIG. 1.—DISCHARGE END AND FEED END OF PEBBLE TUBE MILL.

The feed end is shown on the right of Fig. 1, while the discharge end is shown on the left hand. This arrangement does away with the special drive, as it is attached directly to the mill with which it revolves, requires no stuffing boxes, does away with manholes in the shell (which are always an obstruction to making a perfect lining inside of the machine), can be used to feed the pebbles into the mills, and can also be used to discharge them by simply removing the handhole plate on the outside head of the discharge end of the tube mill, taking out the one shown in the left-hand picture of Fig. 1, replacing the one on the outside head and starting the machine. The pebbles will come out through the handhole left open, be conveyed by the spiral to the center, at which point they will be discharged just like material. In this way all labor of shoveling pebbles is avoided, when, for any reason, the user desires to empty the mill completely.

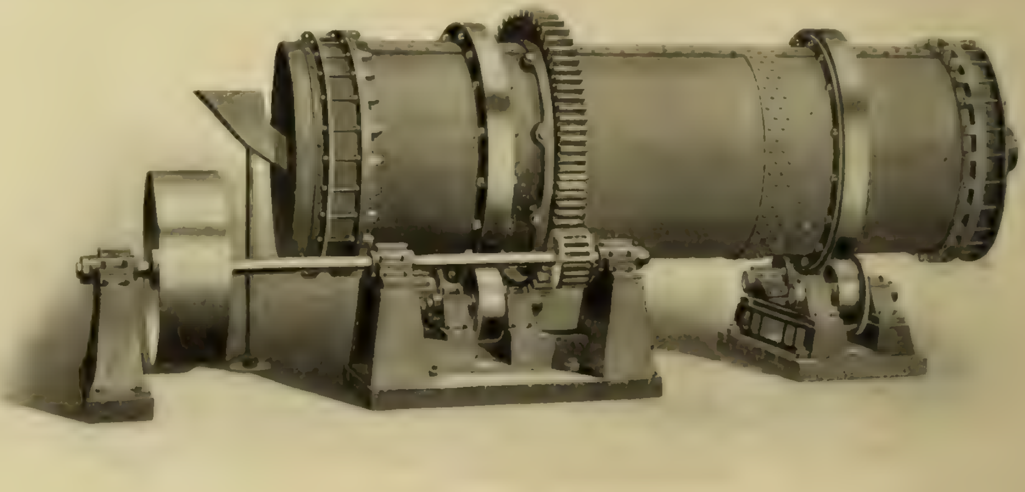


FIG. 2.—PEBBLE TUBE MILL.

The right-hand of Fig. 1 shows the feed end of the latest style machine open; over the outside surface of the spiral there is a one-inch thick plate, which is held in position by the bolts shown without nuts. This plate is perfectly flat, having an opening near its circumference, the opening ending where the spiral starts. Through this opening the material passes into the spiral and is fed to the mill. In front of this plate is fastened a wrought-iron receiving chamber, into which the material can be fed by a chute from a bin or other means convenient to the user. Fig. 2 shows this wrought-iron construction on the feed end of the machines.

The machine shown in Fig. 2 represents the most modern of the pebble tube mills built by the Abbe Engineering Company, St. Paul Building, New York City. It is stated that two of these mills, 5 feet diameter x 22 feet long, will be installed in a Colorado gold mine in the near future, and the results which will be obtained there will be awaited with great interest. The Abbe Co. has made a specialty of fine pulverization for a number of years, and are building to-day pebble mills in eight styles and thirty different sizes, thus having the most complete line of this class of machinery in the world. From their new eighty-page catalogue we note that this company has recently added a new feature to their business, which is a complete testing laboratory, where they have upwards of a dozen of their different mills constantly set up, ready to crush, cut, grind or mix, either dry or wet, any product, when interested parties desire to see the machines do their work.

An Improved Short Beam Analytical Balance.

The accompanying illustration shows a new analytical balance just placed upon the market by Wm. Ainsworth & Sons, the well-known balance manufacturers of Denver, Col.

Among the more important improvements embodied in this balance may be mentioned the beam, which is made of hard-rolled nickel-aluminium instead of cast pure aluminium, as is usually supplied; its design such as will enable it to maintain its adjustment throughout a wide range of temperature and still have ample strength to carry 50 per cent. more than its rated load.

The rider apparatus is of improved design, and so constructed that the rider arm or carrier cannot touch the beam, and, if need be, the rider can be taken off and replaced in the dark without dropping it off the carrier or knocking it off the beam.

The hangers are of skeleton construction, similar to those used on their button balances and have a maximum carrying capacity with a minimum amount of weight.

BALANCE.

Two sensitive levels are set in the metal base in front of the index and in plain view of the operator.

The case is of French polished mahogany, with counterpoised sliding door in front and a removable sliding door in back. A plate glass sub-base covers the entire top of base. All metal work is gold plated. In the illustration the counterpoised sliding door has been removed to better illustrate the balance. A gold medal has been awarded Messrs. Wm. Ainsworth & Sons for fine balances and weights at the World's Fair, St. Louis.

Bauxite Brick.

In connection with the problem of refractory furnace lining to which we have given considerable attention in the past (especially in our November issue, 1904), a note published in *Iron Age* of December 1, is interesting. It refers to a process recently patented by W. F. Berger, for making refractory brick out of bauxite. The value of bauxite as a refractory material has been known for several years, but its lack of cohesive qualities has heretofore prevented its being molded into brick. The patent of Berger covers the use of a binder with calcined bauxite.

The new process uses a natural bauxite from Arkansas which is particularly low in silica, bonded by a small percentage of plastic fire clay. The amount of silica in the finished brick is considerably less than 10 per cent., and is not appreciably detrimental to its elasticity. The brick contains from 88 to 90 per cent. of alumina, and from 10 to 12 per cent. ferric oxide, titanic acid and silica. This brick is especially adapted for linings of basic open hearth furnaces. The highest grade bauxite brick are used in the floor and walls up to the slag line, protected by a bed of calcined bauxite. Above the slag line cheaper brick, with a lower percentage of bauxite, may be used.

The brick is now made on a commercial scale by the American Bauxite Company. The price at which it will be sold is stated to be considerably lower than that of magnesite brick and there are one-third more bricks to the ton. Severe tests have been made in several of the principal steel plants of the country which are said to have proved very successful.

A Novel Filter-Press Design.

We herewith illustrate a new type of filter-press plate, patented by the Niles-Bement-Pond Co., of 136-138 Liberty Street, New York city, which constitutes a rather radical departure from the types at present in use.

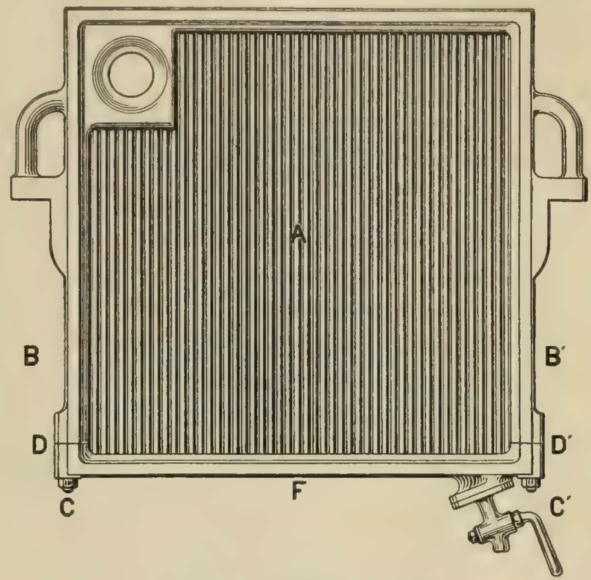
Instead of being made of cast-iron in one piece, the plate consists of a panel *A* of corrugated steel, held loosely in a groove in a cast-iron frame *B*. This frame is of sufficient depth to give the required thickness of cake. In making the frames, special precautions are taken to insure uniform thickness and true joint faces. This is accomplished by casting the frame in one piece, drilling the holes for the studs, *C*, *C'*, and then only parting the frame at *D*, *D'*. Only after this joint has been permanently made, the two joint faces of the frame are planed.

To carry away the clear liquor, a drainage groove *E* is formed in the bottom member *F* of the frame, which opens into an outlet port *G*, provided, as usual, with a nozzle or cock.

The advantages claimed for this method are the following: A much deeper groove can be employed with the steel panel than is possible with a cast-iron plate, without making the latter excessively thick. Consequently, the area of free filter cloth is much larger, as is also the cross-section of groove behind it, the result being more rapid filtration and better drainage of the clear liquor. The latter is still further improved by the method of carrying off the filtrate. It will be noticed in the plan and cross-section of the plate, that the grooves of the steel panel open directly and to their full cross-section into the drainage groove *E*. This outlet passage cannot be obstructed by the sagging of the filter cloth, as in a cast-iron plate, where it is formed on the surface of the plate.

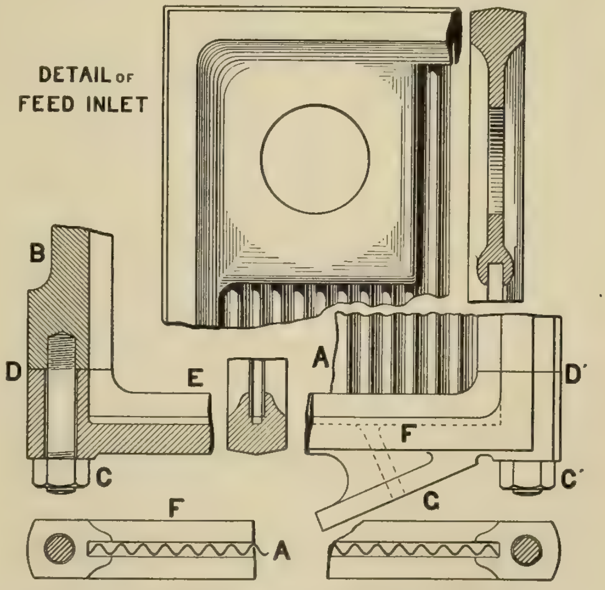
One of the greatest advantages is the very considerable reduction in the weight of the plates, the saving amounting to about 50 pounds on the 24-inch square, and about 100 pounds on the 32-inch square plate. The price of the plates is correspondingly lower, while freight charges, labor in handling, and risk of breakage are all much reduced.

Panel plates are much more easily cleaned than cast-iron plates, and any scale formed on them may be removed by tapping. For the filtration of hot liquors, the steel panel plate is also much superior to cast plates. The latter often break in consequence of the central part of the plate expanding more rapidly than the tightly held rim. The steel panel is only loosely held in its frame, and is thus free to expand without causing any trouble. In case of damage to a plate by unequal pressure, such as is sometimes caused by the blocking of the feed channel to a chamber, only the steel panel is rendered useless. With cast-iron plates, which, just like the panel plates, cannot be made strong enough to resist pressure, applied to one side only, a broken plate is entirely lost, the damage amounting from five to ten times the cost of a new corrugated panel



STEEL PANEL PLATE

FIG. 1.—FILTER PRESS PLATE.



DETAILS OF FRAMING SHOWING DRAINAGE GROOVE ETC.

FIG. 2.—DETAILS OF PLATE CONSTRUCTION

The [illegible] in the illustration is shown in one corner, in a lug of the cast-iron framing, but it may also be placed in the center, as in cast-iron plates. These steel panel plates are made in all sizes from 18 inches to 36 inches square, and in all usual types, non-washing and washing, with open or closed deliveries.

Pyrometers Suitable for Metallurgical Work.

(*Concluded from Vol. II., page* 510.)

5. *The Siemens Electric Pyrometer* (made by Messrs. Siemens Brothers & Co.).—The Siemens electrical pyrometer is a platinum resistance thermometer which has a coil of platinum wire wound upon a cylinder of refractory material, and fixed usually at the end of a long closed iron tube where it is protected by a platinum shield. To measure in actual practice the resistance of the coil and thus its temperature, two types of apparatus are employed. One of these, comprising a differential galvanometer and a set of resistance coils, gives readings in ohms, from which the temperatures are ascertained by means of special tables. The other is a combination of a small D'Arsonval galvanometer and a Wheatstone bridge of circular form with sliding contact, and gives readings directly in thermometric degrees. The first of these types is an instrument of great accuracy, and its determinations can be relied upon to agree within a fraction of 1 per cent with those of an air thermometer. It is very simple in its arrangement and working, and may be entrusted to the care of any intelligent workman. It finds special application in works where it is necessary to ascertain from time to time the heat of furnaces, flues, etc., particularly those in which the processes demand the maintenance of a definite temperature during certain operations. It is employed in most of the principal steel works in the United Kingdom and on the Continent for determining temperatures during the manufacture of steel plates.

Fig. 1 is a diagram showing the pyrometer and galvanometer with resistance coils and battery connected up for taking a reading. The galvanometer is placed with its needle pointing to zero and the end of the pyrometer tube is inserted in the furnace the temperature of which is to be measured, the upper part of the tube being protected from the heat by fireclay or a sheet of iron. When the tube is thought to have acquired the temperature of the furnace, the key marked K is pressed down, causing the battery current to divide at the back of the key into two circuits, one of which includes one coil of the galvanometer and the pyrometer coil, while the other includes the other galvanometer coil and the resistance box. If the resistance of the pyrometer coil does not equal that of the circuit of the resistance box, the needle is deflected, and in order to establish a balance the holes in the resistance boxes must be plugged or unplugged, until the galvanometer needle returns to zero. The sum of the unplugged whole numbers of the resistance box, plus the decimal marked opposite the hole filled by the traveling peg, will then give the resistance of the pyrometer coil in ohms. In order to take a reading, the traveling peg connected with the terminal R' is inserted in the hole marked *o*, and the resistance between the fixed terminals T and *o* is unplugged until the galvanometer is balanced as nearly as possible, while the key K is continuously depressed. The traveling peg is then moved along the decimal coils of the resistance box until the point is reached when the galvanometer needle remains exactly at zero and the unplugged resistance between terminals T and *o* is then read off plus the decimal indicated by the traveling peg. The resistances between the terminals T and *o* are additive, while the decimal parts progress by 0.05 ohm from 0 to 1 ohm. Thus, if the holes 1, 2, 4 and 20 are unplugged, and the traveling peg is in the 0.65 hole, the resistance of the coil will be 27.65 ohms. The length of the [illegible] A and B connecting the galvanometer to the pyrometer tube does not affect the reading so long as the wires are of equal resistance, as their resistances vary equally with variations of temperature and balance one another.

The second type, or direct-reading apparatus, comprises a Wheatstone bridge, one limb of which is an adjustable resistance formed out of a helical coil of wire arranged round the edge of a circular dial. The dial is divided into either Fahrenheit or Centigrade degrees as desired, and in the center of it is a small D'Arsonval galvanometer, a type of instrument unaffected by external magnetic fields. A sliding contact with a pointer serves to adjust the resistance of the variable limb of the bridge, and at the same time to indicate on the dial the temperature of the coil when the balance has been obtained. The keys are combined and fitted to the arm of the sliding contact in such a way that by depressing the knob of the contact the battery is put into circuit with the galvanometer and pyrometer. The whole is mounted upon a tripod stand. To take a temperature, the pyrometer is connected by means of a three-cored cable to three of the terminals and the battery to two terminals of the instrument. The sliding arm is moved to some point on the scale and the lever depressed until the point is found at which the depression of the key produces no deflection of the needle of the galvanometer. The pointer attached to the sliding contact will then indicate the temperature of the pyrometer coil. This type of instrument is quite as sensitive as the first type described, and it possesses a great advantage in avoiding the inconvenience and liability to error arising from the use of conversion tables. There is, moreover, only one piece of apparatus to be handled instead of two, and, unlike the other, it is not affected by external magnetic fields due to the proximity of dynamos or powerful electric currents, while its manipulation is even more simple.

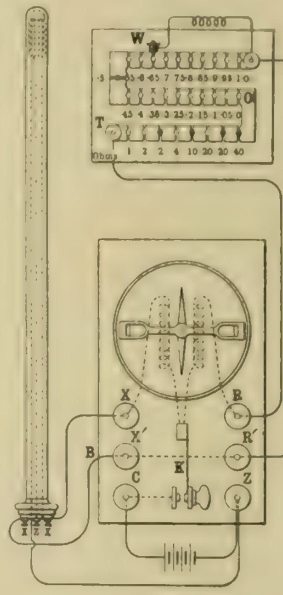


FIG. 1.—SIEMENS PYROMETER.

The **Westinghouse Electric & Manufacturing Co.** has closed a contract with the Ontario Power Company for an alternating current generator with a rated output of 10,000 hp., at 85 per cent power factor. This is in addition to three other machines of similar type, which the Westinghouse Co. is furnishing for this plant. The generators are of the revolving-field, two-bearing type, designed for direct connection to water-wheels; they generate three-phase current at 12,000 volts and 25 cycles, and run at a speed of 187½ r. p. m. Among other apparatus included in the contracts are twelve 3000-kw. oil-insulated, water-cooled transformers, wound for 12,000 and 60,000 volts; two 375-kw. exciters, and complete switchboards. The plant is being built by the Niagara Construction Company.

Largest Electrical Generators in the World Driven by Gas Engines.—The California Gas & Electric Corporation, San Francisco, Cal., has just placed an order with the **Crocker-Wheeler** Co., Ampere, N. J., for three 4000-kw., three-phase, 13,200-volt, 25-cycle, 88-r. p. m., revolving-field, alternating-current generators, to be driven by 5400-hp. gas engines built by The Snow Pump Co. These generators are the largest in capacity in the world driven by gas engines, and will furnish power for operating all the street railways in San Francisco and vicinity. The alternating-current machinery of the Crocker-Wheeler Company is of the design of Brown, Boveri & Cie., the celebrated Swiss electrical engineers.

St. Louis World's Fair.

In former issues, and especially in our last September number, we have given illustrated descriptions of those exhibits which should be specially interesting to our readers. The following notes and illustrations are supplementary to our former articles. The exhibits of the Goldschmidt Thermit Co., and of the Roessler & Hasslacher Co., illustrations of which are given below, are described on page 376 of our volume II.

SOCIETE ELECTRO-METALLURGIQUE FRANCAISE.

This company was formed in 1888 for the manufacture of aluminium by the Héroult process. The company has works at Froges and LaPraz. Besides aluminium, the chief products of the company are steel and ferro-alloys, made in the Héroult electric furnace, and it is with respect to these products that the exhibit of this company was specially interesting.

Dr. Héroult's work in this line has been noticed in our columns repeatedly at length, with descriptions of his various types of furnaces, so that we may record here only the principal features of the exhibit. The wide range of steel of different qualities which can be produced by the Héroult process, was well illustrated by two samples of extreme grades exhibited—namely: one of extra mild quality, containing traces only of carbon; and another with a high percentage (4.07) of carbon, this high yield in carbon being obtained by liquid carbonization in the electric furnace. These two extreme grades, together with the various other qualities shown, were manufactured in the Héroult oscillating electric furnace, a model of which, on a reduced scale, was shown in the center of the exhibit.

By starting a heat with low quality material, steel is produced commercially in the electric furnace containing 0.005 to 0.015 phosphorus; 0.005 to 0.020 sulphur; any desired percentage of carbon from 0.01 to 4 per cent, any desired percentage of silicon and manganese, and any desired percentage of chrome, tungsten, etc.

The specimens of castings exhibited were not given as illustrating perfect foundry practice; some of these specimens, however, were interesting in that they showed that the manufacture of castings of extra mild steel (pure iron), perfectly sound notwithstanding the small thickness, ⅛ inch in the specimens exhibited, is a very easy matter. The ordinary class of cast-iron articles were used as patterns in the moulding of those shown in the exhibit. A cast plate, exhibited 3-16 inch in thickness, was also of extra mild steel. A bell was a specimen of a steel casting, the steel containing about 1 per cent of carbon. The carbon steel obtained by the Héroult process is said to be easy to work either cold or hot; it welds readily, tempers well, and can be annealed and tempered repeatedly without deteriorating. Numerous specimens of ingots obtained by the Héroult process, containing various percentages of carbon, were displayed. They were broken to show the homogeneity of the metal and its soundness. These ingots were chosen to illustrate the variety obtained in the brands, from iron containing 0.02 per cent of carbon to the specimen of highly carburized steel containing over 4 per cent. There were exhibited also, grouped together to form two pillars, sample bars of various brands of carbon tool-steel, chrome steel bars, a bar of special steel for chisels, etc. The show-cases contained besides, numerous broken and bent test pieces.

Samples of ferro-chrome, ferro-silicon, ferro-nickel and ferro-tungsten, and of white and gray pig iron, made in the electric furnace, were also exhibited; also some samples of copper matte, produced at LaPraz in the electric furnace from a silicious attle ore containing about 6 per cent of copper. The matte yields approximately 40 per cent copper.

SOCIETE ANONYME ELECTROMETALLURGIQUE PROCEDES PAUL GIROD.

The exhibit of this company consisted of samples of electric furnace products, especially ferro-alloys, such as ferro-silicon, ferro-titanium, ferro-chrome, ferro-nickel-silicon, also tungsten steel and vanadium steel.

CHEMICAL EXHIBITION OF GERMAN GOVERNMENT.

This very interesting exhibition, arranged under the auspices of the German Government, was intended to give an historical outline of the development of chemical science and industries in Germany. The following societies had contributed exhibits: The German Chemical Society, the Society for Protecting the Interests of the Chemical Industry of Germany, the Society of German Chemists, the German Bunsen Society, besides manufacturers of chemical instruments and apparatus. Besides thirty-nine chemical manufacturers and twenty-five makers of chemical apparatus, 117 professors and instructors of universities were represented by exhibits.

The exhibition was divided into eleven departments, namely, a reading room, an alchemistic laboratory, Liebig's laboratory, exhibits of general and inorganic chemistry, pyrochemistry, balances, electrochemistry, organic chemistry, dyeing laboratory, and physiological chemistry.

The electrochemical department was specially interesting for a great many instruments exhibited, among them the historical apparatus by which Kohlrausch made his classical researches on conductivities of solutions; various apparatus of Ostwald; a complete arrangement of working desks with instruments, etc., for electrochemical researches, the arrangement being the same as in the electrochemical laboratory of the University of Goettingen; apparatus of Classen for quantitative analysis for electrolysis; and apparatus of the Vereinigte Fabriken für Laboratoriumsbedarf (New York branch, Laboratory and School Supply Co.) for demonstrating electrochemical processes and laws.

EXHIBIT OF ELECTRIC STEEL, HEROULT PROCESS.

There was also exhibited a Siemens & Halske ozone apparatus, while the Hofman Haus in Berlin, Prof. Ost and Prof. Elbs had contributed a collection of electrochemical products.

THE WESTINGHOUSE STEAM TURBINE.

A memorable incident of the morning following the close of the St. Louis Exposition was the formal shut-down and inspection of the 600-hp. Westinghouse steam turbine generating unit in the Palace of Machinery, after a continuous run of over 3962 hours—a performance which has had no parallel in steam turbine history. During the five and a half months that the unit was in operation it supplied current for light and power throughout the Westinghouse exhibits in the Palaces of Machinery, Electricity and Transportation. When the machine was stopped, it was found to be in perfect condition, and there were no signs of wear, the bearings still retaining the tool marks as they had come from the shops. There have been at least two instances on record in America in which piston engines have been run continuously for about the same length of time as that of the record run of the Westinghouse turbine. The remarkable feature of the turbine run, of course, was the maintenance under load of a speed of 3600 revolutions a minute for such a long period. From 8.30 o'clock in the morning to 10.30 o'clock in the evening, the load carried throughout the Exposition varied from 25 per cent underload to 25 per cent overload. The total number of revolutions almost touched the billion mark—855,702,000.

ALCHEMISTIC LABORATORY IN GERMAN CHEMICAL EXHIBITION.

Storage Battery Locomotive.

In the court of the Palace of Electricity there was an exhibit and demonstration of an "industrial" railway and locomotive, the design of the C. W. Hunt Co., of West New Brighton, S. I., N. Y. The locomotive was of the narrow-gauge electric storage battery type, and was especially designed for use with "industrial" railways in machine shops, factories, storage yards, smelting and refining plants, steel works, shipbuilding yards, wharves, plantations, lumber mills and wherever it is designed to handle economically, material from one part of a plant to another. The gauge of the road at the exposition was 21½ inches, which is the standard of the "industrial" railroads built and laid by the C. W. Hunt Co., and with which it is possible to operate around curves of 12 feet radius, thus enabling any part of the works of a factory to be reached, a matter of importance for manufactories with irregularly placed buildings. The locomotive carries a storage battery which furnishes power to two independent motors. There are eight driving wheels to the locomotive, and the design is such that all of the wheels are used as drivers, thus securing a maximum tractive effort and a duplication of mechanism. The motors are made by the Westinghouse Electric Co., their electrical efficiency being high.

In the Electricity Building the company was exhibiting a straight-line electric hoist, together with photographs of [illegible] which have been equipped with their coal-handling machinery, a line of manufacture in which the company was the [illegible]

Awards.

Among the prizes awarded to exhibitors were the following:

Machinery Building. Grand Prizes.—Allis Chalmers Co., Chicago; General Electric Co., Schenectady, N. Y.; Goldschmidt Thermit Co., New York; A. Leschen & Sons Rope Co., St. Louis; Lidgerwood Manufacturing Co., New York, Niles-Bement-Pond Co., New York; Robbins Conveying Belt Co., New York; Westinghouse Machine Co., Pittsburg, Pa.; R. D. Wood & Co., Philadelphia, Pa.; Henry R. Worthington, New York.

LIEBIG'S LABORATORY IN GERMAN CHEMICAL EXHIBITION.

Gold Medals.—Buffalo Forge Co., Buffalo, N. Y.; Builders' Iron Foundry, Providence, R. I.; A. S. Cameron Steam Pump Co., New York; Carborundum Co., Niagara Falls, N. Y.; Dearborn Drug & Chemical Company, St. Louis; De Laval Steam Turbine Co., Lombard Governor Co., Boston, Mass.; Lunkenheimer Co., Cincinnati; Aug. Mietz, New York; Mietz & Weiss, New York; Norton Emery Wheel Co., Worcester, Mass.; Otto Gas Engine Co., Philadelphia; Weber Gas & Gasoline Engine Co., Kansas City, Mo.; Société De Laval, Paris, France.

EXHIBIT OF GOLDSCHMIDT THERMIT CO.

Mines and Metallurgy. Grand Prizes.—E. G. Acheson, Niagara Falls, N. Y.; Allis-Chalmers Co., Chicago, Ill.; American Coal Products Co., New York; Austin Manufacturing Co., Chicago, Ill.; Baker & Co., Newark, N. J.; Bethlehem Steel Co., South Bethlehem, Pa.; Canadian Copper Co., Sudbury, Ontario; Dubois, Pinard et Cie., Sougland, par Saint-Michel, Arsne.; Goldschmidt Thermit Co., New York; International Acheson Graphite Co., Niagara Falls; International Nickel Co., New York; Julian Kennedy, Pittsburg, Pa.; Koenigliche Porzellan Manufaktur, Berlin, Germany; Lanyon Zinc Co., St. Louis, Mo.; A. Leschen & Sons Rope Co., St. Louis, Mo.; Manganese Steel Safe Co., New York; Morgan Construction Co., Worcester, Mass.; Northwestern Terra Cotta Co., Chicago, Ill.; Norton Emery Wheel Co., Niagara Falls, N. Y.; Pittsburg Reduction Co., Pittsburg, Pa.; Société Electro-Metallurgique Francaise de Froges, Froges, Isere; Solvay Process Co., Syracuse, N. Y.; Taylor Iron and Steel Co., High Bridge, N. J.; Tiffany & Co., New York; Worth Brothers' Co., Coatesville, Pa.

Gold Medals.—William Ainsworth & Sons, Denver, Colo.; American Concentrator Co., Joplin, Mo.; Baker & Co., Inc., Newark, N. J.; Baldwin Locomotive Works, Philadelphia, Pa.; C. O. Bartlett & Sons Co., Cleveland, Ohio; Cyrus Borgner, Philadelphia, Pa.; F. W. Braun & Co., Los Angeles, Cal.; Carborundum Co., Niagara Falls, N. Y.; Colorado Iron Works, Denver, Colo.; Sherard Cowper-Cowles & Co., London, England; Crawford & McCrimmon Co., Brazil, Ind.; De Laval Steam Turbine Co., Newark, N. J.; Denver Fire Clay Co., Denver, Colo.; Eimer & Amend, New York, N. Y.; Charles Engelhardt, New York; Arthur Fritch Foundry & Milling Co., St. Louis, Mo.; General Electric Co., Schenectady, N. Y.; Granbury Smelting & Refining Co., St. Louis, Mo.; Aug. Gundlach, Joshua Hendy Machine Co., San Francisco, Cal.; Hinque, Marret et Bonnain, Paris, France; Hohman & Maurer Manufacturing Co., Rochester, N. Y.; Keuffel & Esser, New York; Laclede Fire Brick Co., St. Louis, Mo.; Lidgerwood Manufacturing Co., New York; Louisville Fire Brick Co., Louisville, Ky.; Missouri Fire Brick, St. Louis; National Lead Co., St. Louis, Mo.; Primos Chemical Co., Primos, Pa.; Robins Belt Conveying Co., New York; Salt Lake Hardware Co., Salt Lake, Utah; Schoellborn-Albrecht Machine Co., St. Louis; Siemens-Halske Co., Berlin; Société Anonyme Electro-Metallurgique D'Albertville, Savoie, France; Robert M. Thompson, New York City; Trenton Iron Co., Trenton, N. J.; Western Carbonic Gas Co., Sacramento, Cal.; Western Gas Construction Co., Fort Wayne, Ind.; Wetherill Separator Co., New York City; Jos. Wharton, Philadelphia, Pa.; Wilson Aluminum Co., Holcomb, Va.; Williams Patent Crusher & Pulverizer Co., St. Louis, Mo.; R. D. Wood & Co., Philadelphia, Pa.

EXHIBIT OF THE ROESSLER & HASSLACHER CO.

Liberal Arts Building. Grand Prize.—The Roessler & Hasslacher Co., New York.

Manufacturers and Varied Industries Building. Grand Prize.—Carborundum Co., Niagara Falls, N. Y.

The official list of the awards in the Electricity Building has not yet been published.

Industrial Notes.

The plant of the J. T. Baker Chemical Co., of Easton, Pa., is nearing completion. This plant comprises several buildings located on the outskirts of Phillipsburg, N. J. The intention of the company is to manufacture strictly c. p. acids and chemicals for analytical and laboratory use. J. T. Baker, formerly of the Baker & Adamson Chemical Co., is president of the concern. It is expected to have the plant, or at least part of it, in operation some time during this month.

The Power & Mining Machinery Co., of Cudahy, Wis., has just issued bulletin C1 on the Holthoff revolving hearth roasting furnace. This is built in four different types. Type A is designed for roasting low sulphur ores, preparatory to chemical treatment, and is built with a centrally located gas producer. Type B is designed for the same purpose, and is similar to type A, except that a plain fire-box is substituted for the gas producer. Type C is designed for roasting sulphide ores preparatory to smelting. Type D is designed especially for roasting pyritic ores without the use of fuel other than sufficient to ignite the sulphur. The pamphlet describes the special features of the four various types and their general construction, and is neatly illustrated.

Mr. E. R. Taylor, president of the Taylor Chemical Co., of Penn Yan, N. Y., reports for the month of December, 1904, the largest monthly output of bisulphide of carbon he ever had. Mr. Taylor's electric furnace was described in our Vol. I., pages 60, 63, 76 and 568.

Personal.

Messrs. FitzGerald and Bennie, of Niagara Falls, have become the American associates of the Société Anonyme d'Etudes Electrochimiques, of Geneva, Switzerland.

The many friends of Mr. Alois von Isakovics, the well-known secretary of the New York section of the American Electrochemical Society, will be glad to hear that he is now almost completely restored to health, and has started in business at Monticello, N. Y., under the firm name of Synfleur Scientific Laboratories.

Mr. C. J. Pretzfeld, formerly of the Niagara Research Laboratories, is now associated with the People's Gas Light & Coke Company, of Chicago.

Mr. Woolsey McA. Johnson has resigned his position as metallurgist of the No. 2 works at Iola, Kansas, of the Lanyon Zinc Company, and head of the laboratory for metallurgical research at La Harpe. Mr. Johnson expects to be in New York for some time.

Mr. Jacob Langeloth, president of the American Metal Company, New York, has been elected president of the Granby Consolidated Mining, Smelting & Power Company of British Columbia.

Almanacs.

The *Westinghouse Electric and Manufacturing Co.* has issued a very useful diary for 1905, neatly bound in black leather. Besides the pages ordinarily contained in a diary, and several small maps of the United States and its foreign possessions, the little book contains carefully selected information on various subjects of timely interest and usefulness for electrical and mechanical engineers, the data given being based on the latest and best practice. The subjects dealt with are electric railways (train resistance, electric car heating, drop in line and rails, single-phase traction); electric motors for machine tool drive (lathes, boring mills, drill presses, milling machines, slotters, shapers, planers, power pressers); electric illumination (Nernst lamps, Cooper-Hewitt lamps, arc lamps); power transmission (the new alternating current formulae, wire tables, convenient formulae, lightning arresters and choke coil (low equivalent arresters, multi-path arresters, alternating or direct-current circuits); "sparks;" operating hints on electrical machinery; first aid to the injured; steam boilers and heating value of coal; commercial power gases; gas engines, performance curves; steam turbines, performance curves. The handsome diary should prove very useful on the desk, as well as in the pocket of engineers.

The *Pope Bicycle Daily Memorandum Calendar* for 1905 is a desk calendar containing a memorandum leaf for every day in the year and 365 original sayings in favor of good roads, good, healthy out-door exercise and the modern bicycle, by a great many prominent men. Dr. Ira Remsen, of Johns Hopkins, writes: "If anything I can say or do will help spread the proper use of the bicycle, I shall be glad to say or do it, for I feel that the bicycle is one of the best friends mankind has ever had." Prof. Elihu Thomson says: "Good roads are an important asset in national wealth. The bicycle has been, and it is hoped may continue to be a potent cause of this great sentiment leading finally to the nation's enrichment." Dr. Charles S. Palmer, now one of the editors of the *Engineering and Mining Journal*, writes: "A good thing may be measured by the variety and quality of its benefits—to the one and to the many. Time, health, happiness—that is the measure of a good wheel on a good road." The calendar is free at the Pope Manufacturing Company's stores, or any of our readers can obtain it by sending five two-cent stamps of the Pope Manufacturing Co., Hartford, Conn., or 143 Sigel Street, Chicago, Ill.

Digest of U. S. Patents.

PRIOR TO JULY, 1902.

Compiled by Byrnes & Townsend, Patent Lawyers, National Union Building, Washington, D. C.

MOLTEN ELECTROLYTES, MISCELLANEOUS.

(*Continued from Vol. II., page* 518.)

610,014. August 30, 1889. Heinrich C. Aschermann, Cassel, Germany.

Reduces refractory oxides as the oxides of chromium and aluminium, and produces the metals free from carbon by employing as a flux and reducing agent sulphide of antimony. The reaction occurs according to the following equation:

$$10Cr_2O_3 + 23Sb_2S_3 = 20Cr + 46Sb + 15SO_2 + 54S.$$

The mixture to be reduced is placed in an iron vessel connected as the cathode, an anode of carbon being centrally inserted. The E. M. F. is in all cases kept at the practicable minimum. The antimony is separated from the chronium by volatilazation.

618,575. January 31, 1899. Farnham Maxwell Lyte, London, England.

Treats complex sulphides of zinc, lead and silver by grinding, calcining at low redness to convert zinc sulphide to sulphate, which is leached out. The equivalent quantity of sodium or calcium chloride is added to the previously neutralized lixiviate forming zinc chloride from which the alkaline sulphate is separated by refrigeration. The zinc chloride solution is evaporated and dehydrated with production of some zinc oxide by heating it in contact with molten zinc. The oxidation of the zinc is accelerated by alloying it with antimony to form a galvanic couple, or by associating it with carbon. The anhydrous fused zinc chloride containing zinc oxide in solution and suspension is electrolyzed above a cathode of molten zinc. The vessel is of iron, the sides being lined with firebrick set in china clay, powdered pumice or silica sand and water glass, and the bottom being covered by a layer of silicated asbestos cardboard or fire clay. Carbon conductor to the cathode is enameled. Carbon anodes with enlarged ends perforated for escape of chlorine. The molten zinc is re- [illegible]

641,438. January 16, 1900. James D. Darling, Philadelphia, Pa.

Apparatus for the electrolysis of hydroxides or nitrates of the alkali metals, employing a diaphragm of concentric perforated metallic walls, the interspace being filled with particles of vitrified magnesia as described in patent 590,826. The inner wall of the diaphragm is preferably of double thickness. The cathode is within the inner cup and the anode of cast iron surrounds the diaphragm. In order to protect the metallic walls of the diaphragm against local action, a portion of the current, say 5 per cent., is shunted to these walls, the shunt being connected through a suitable resistance with the anode lead.

642,390. January 30, 1900. Frank P. Van Denbergh, Buffalo, New York.

Sulphuric acid is produced by the electrolysis of melted gypsum in an oxidizing atmosphere and in presence of a suitable silicious flux, such as sand, quartz, clay, etc. The oxidizing atmosphere may be obtained by mixing hematite iron ore with the charge, or by injecting steam, air, or oxygen. Preferably steam is injected in sufficient proportion to oxidize the sulphur dioxide and to hydrate the resulting trioxide. A continuous furnace is described having lateral electrodes of carbon, and a siphon tap for the calcium silicate slag. Continuous charging devices are described, and the gaseous products are removed by a steam jet. It is claimed that the acid is free from impurities, particularly from arsenic.

642,933. February 6, 1900. Oscar J. Steinhart, Julius L. F. Vogel, and Henry E. Fry, London, England.

Produces zinc by electrolyzing a molten solution of zinc oxide in zinc chloride, to which may be added a small amount of sodium chloride. The cell is a rectangular pan of refractory material placed over furnace flues. The cover of the pan consists of a series of transverse carbon blocks having depending portions constituting the anode. The cathode is a layer of zinc about 1 inch deep on the bottom of the pan and maintained at a temperature just above its melting point. The bath floating on the zinc is about two inches deep. The zinc chloride is preferably obtained by evaporating and dehydrating zinc chloride in a partial vacuum. The voltage is 3. The zinc is withdrawn through a siphon tap and fresh oxide is added as required.

669,271. March 5, 1901. Frank P. Van Denbergh, Buffalo, New York.

Produces phosphoric acid by electrolyzing a molten mixture of apatite and a silicious flux such as sand, clay, etc., an oxidizing atmosphere being maintained above the bath to convert the phosphorous which is freed and volatilized into the pentoxine. This oxide is then hydrated as by the introduction of steam. The oxidizing atmosphere contains an excess of free oxygen which may be supplied by hematite ore, steam, oxygen, ozone, etc. The fused slag runs out through a siphon tap. The apparatus is that of patent 642,390.

677,906. July 9, 1901. Frank P. Van Denbergh, Buffalo, New York.

Produces alkali silicates by electrolyzing a molten mixture of an alkali chloride and a silicious flux such as silica feldspar, etc., in the presence of an atmosphere containing an excess of oxygen which may be supplied by iron oxide, manganese oxide, an alkali sulphate or nitrate, water, steam, oxygen or ozone. The preferred bath is sodium chloride containing a small amount of potassium chloride to lower the melting point. The sodium chloride should be in amount from two to four times that of the silicious flux. The preferred apparatus is that of patent 626,373. Steam or air is projected into the furnace to oxidize and hydrate the molten mass. The molten silicate or mixed cilicates flow out into a tank containing water and may be treated with carbon dioxide to produce sodium carbonate and hydrated silica; or milk of lime may be added to produce calcium silicate and caustic soda or [illegible]

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Card Index.

Commencing with this issue, we print on one of the advertising pages the table of contents of our preceding issue in such a form that it may be cut out and pasted on a card. The contents of our different issues may thus be kept in the convenient form of a card index, until, at the end of the year, the volume index appears.

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Manufacture of Brass in the Electric Furnace.

We have pointed out on several occasions the present commercial developments in the manufacture of special steels in the electric furnace. We noted that there are several marked points of superiority in the new method. These were enumerated briefly as follows: First, larger units in place of small units, and consequently smaller labor charge per ton of output; second, durability of the electric furnace, as against the small and costly "graphite-clay" crucible; third, ease of control over purity of product, due to the fact that the heat is generated inside the furnace, and can thus be easily regulated. Just these same factors enter into the manufacture of brass in the present crucible process. There is is an extremely high labor charge, for the brass founder's skill in producing a first-class billet can immensely increase the efficiency of the rolling mill. Naturally, such labor deserves and receives high wages. The crucibles, costing two to three dollars each, only melt from 150 to 250 pounds of metal at a time, and their life is short, not over thirty-five "melts," as the maximum average. Naturally, with the small weights of metal, it is hard to control the quality of brass produced. Any accidental large amount of impurity in the particular slab of spelter or in the particular ingot of copper will manifest itself immediately in the particular billet of brass produced. If, on the other hand, the "melt" had been of two tons size, this accidental impurity of a single piece of the raw material would have been obliterated by the general average purity of the metals. Add to this freedom from dirt, and ability to control heat to 10° C., we see there stand out in bold relief great innate points of advantage on these scores. Moreover, there is also a certain loss in metal in melting due to the volatility of the zinc, and to the breaking of crucibles in the furnace. This would be nearly eliminated in the new process. Other distinct improvements might be adduced, but the above are enough to show what has happened in the case of crucible steel will happen in the case of brass-melting. What is needed is the combination of good practical electrometallurgical experience with good practical brass experience.

The replacement of the old crucible process is sure to come; perhaps it will come in two years, but it will, necessarily, in ten years. Just as the more positive and stronger metal, as iron, throws down the weaker metal, as copper, from soluthen, as well the process, stronger in its fundamentals, replace the old and well tried process that was developed under old conditions. In both cases, however, the question of the time for the completion of the reaction depends on conditions; and in the industrial replacement, this interval cannot be estimated with any accuracy. Nevertheless, it must and will proceed, unless something, not known at the present time, occurs to disturb the conditions. The inevitable advance along this industrial line will have great influence in the metal world, and will also provide in large cities a new kind of load for the central station.

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Tantalum and the Tantalum Lamp.

In this issue we publish an article giving the first information on an investigation of the preparation and the properties of pure tantalum, carried out by Dr. von Bolton in the chemical research laboratory of the Siemens & Halske Company in Berlin. These researches have led to the construction of the tantalum lamp, which, if judged from the results obtained by Siemens & Halske, promises to become, in future, of great importance in the electric lighting industry. In our article, on another page of this issue, we cover specially the chemical side of the subject. Electric lighting is generally considered to be far removed from chemistry and electrochemistry; and in fact if a type of lamp is given and the problem is to install lamps within a given district and to supply electrical energy from a central station, no chemical considerations are involved in the solution of the problems. It is very different, however, with the manufacture of electric lamps, and it does not seem exaggerated to state broadly that no important progress in the invention of new electric lamps can be made without extended chemical research. The fundamental idea of the incandescent lamp was simple enough, but it is generally not appreciated, what an enormous amount of chemical research had to be done by Edison in the invention of a suitable filament; yet Edison's successful work in this field places him in the front rank of the practical chemists of his time. All later developments in electric lighting were intimately connected with chemical research; we may simply refer to the use of impregnated carbons in arc lighting, such as in the Bremer lamp, and to the development of the magnetite lamp and of the Cooper Hewitt lamp; all these developments would have been impossible without chemical research; and if investigations along the lines indicated by the last mentioned lamps should lead to an absolutely new and highly efficient method of lighting, the results must be based on chemical experimental work on one side, and on the principles of thermodynamics and physical chemistry in general on the other side.

To come back to the incandescent lamp, it is surely significant that the two most important recent developments in electric incandescent lighting—the Nernst lamp (which its inventor wanted to call "Elektrolytlicht") and the Welsbach osmium lamp—were the direct outcome of chemical research. One principle on which inventors have been working in their research work in this field, was to produce a filament of a material with a very high melting point. The same principle was the leading idea in Dr. von Bolton's research on tantalum. The results of his work are also interesting in various general respects. First, after Auer von Welsbach's renowned work, we have here another instance showing how important the results of researches on rare elements can become; tantalum was so far only a chemical curiosity and of no practical use. The main reason for this appears to be—and this emphasizes another general point—that up to the present, pure tantalum and its properties were unknown. What Moissan prepared in the electric furnace seems to have been a carbide of tantalum. The fact that pure tantalum has new, unexpected and highly useful properties, should encourage similar work with other elements. There is already much being done along such lines. The universal use, in the old metallurgical methods, of carbon as a reducing agent has resulted in metals and alloys containing carbon. Since we have learned to produce metals and alloys free from carbon either by the aluminothermic method or in the electric furnace, their advantages have been recognized. In the same respect Burgess's and Hambuechen's pure electrolytic iron, absolutely free from carbon, is interesting and should prove a most valuable starting material for researches on the properties of steels, for it is a fact that up to the present in all researches on the influence of additions of various metals to steel, the influence of carbon in the iron has never been eliminated. While to the general public the importance of Dr. Von Bolton's work will appear to rest in the new lamp, we may hope that his success will emphasize the importance of chemical research, and will encourage the establishment of research laboratories by our manufacturers.

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Recent Developments in Cyanide Practice.

In this issue we publish an interesting paper by Mr. Torrente proposing a new continuous process of settlement and separation of the slimes from the cyanide solutions in the decantation process. The method has much to recommend it and has proven successful on a large experimental scale; it will be interesting to await its behavior on a real commercial scale. In the meanwhile, the paper of Mr. Torrente is being discussed by his fellow-members of the Chemical, Metallurgical and Mining Society of South Africa. Thus, Mr. Dowling points out two points of difference between Mr. Torrente's scheme and present practice on the Rand, namely, first, the use of the spjtzkasten principle against its disuse in most modern plants, and, second, the introduction of pulp quietly without agitation; against the tangential delivery now gaining favor. Mr. Dowling concludes that since present practice is the result of years of study and experience, and is giving satisfactory results, it would seem a doubtful policy to return to discarded methods. Against this, one might say that in many cases, methods which were already discarded have later on proven successful under new conditions. Mr. Laschinger points out that a continuous treatment not only requires the establishment of the proper conditions of working, but also the uniform maintenance of these same conditions, and it is the latter problem that is most difficult. The continuous treatment involves the danger that a small amount of foreign matter or slimes with peculiar properties may throw the whole plant out of working at any time. It is the old problem—so important in many metallurgical and chemical processes—of continuous versus intermittent operation. But this general problem really deserves a more detailed discussion than can be given in this short note.

Another problem of most timely interest in cyanide practice is the proposition—which, we believe, was first made by Mr. Denny—to slime everything. It is, of course, clear that in order to get the best chemical effect, *i. e.*, the best extraction, the material to be treated should be as finely subdivided as possible. As a matter of fact, this has really proven to be correct in practice. Thus, Messrs. Butter and Hamilton, from their experience at El Oro, Mexico, conclude that the gradual increase of extraction with increasingly fine grinding, points to the theory that if the whole could be reduced to an absolutely amorphous and impalpable powder, there would be nothing to hinder the total extraction of the value. Mr. Denny's scheme has, however, other advantages; thus, since there is no longer an independent treatment of concentrate, sand and slime, we have a consolidation of plant, resulting in saving in the first cost of installation. On the other hand, it remains to be seen whether on large scale working these advantages are not counter-balanced by an increased working cost, though the experience of those who used the new method speaks against this and distinctly favors the new method. In view of this industrial development it is certainly amusing to remember that not so very long ago the endeavor was to avoid as much as possible the production of slimes—for the simple reason that one did not know how to handle them.

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Consolidation of Metallurgical Experience.

In a recent paper presented by Messrs. Charles Butters and E. M. Hamilton, before the British Institution of Mining and Metallurgy, on regrinding of sands in cyanide practice at El Oro, Mexico, the authors remark that in the earlier days of gold mining, such figures as they give in their paper would never have been given to the public. "But the old policy of secrecy in such matters is gradually passing away, and mining companies all over the world are beginning to see the advantage of thus comparing notes with one another—a course which has resulted in an enormous increase of knowledge in general." It is sincerely to be hoped that this new policy, based on a sense of enlightened self interest, will extend further and further, especially in these days of new enterprises. For it is not at all an exaggeration to say that at the present time there is a more active development in the mining and metallurgical world than there has been at any former period. This new work cannot fail to produce results in the course of time. But as a necessary concomitant to the success, there may be many times as many failures. Industrial progress is like progress in any line—a process of evolution with the destruction of the unfit. Some critic of Darwin once said in a witty criticism on the latter's doctrine, that he did not believe that nature was as foolish as a man who, on a rabbit hunt, should discharge a million random shots in a field and perhaps bag one at this great expense. Nevertheless, nature does make progress in a most expensive and destructive manner.

It is somewhat similar with the development of a new process or a new mine. A certain lot of hard work must be done and a certain amount of good money must be spent to prove with absolute definitiveness that this reaction will work on a large scale, that this construction contains weak points, or that this vein extends beyond the vein which interrupts it. Fortunately, the action of man's intelligence differentiates industrial progress from organic progress. The mind steering the course of the new craft can exercise discrimination. It can reason from analogy and from previous experience that such a reaction will not work on a large scale, because it is impossible to hold conditions uniform and right in the large scale apparatus. Nevertheless, new work, however well planned, has been always expensive and always will be expensive, for the reason that in certain cases human prognostications must be proven before any forward step can be conservatively attempted. Then, too, the expense is heaviest in proving that certain things cannot be done. For it is foolish for any one to say absolutely that anything is impossible. In every case, the probabilities must be balanced against each other. The estimation of these chances is the basis of action. The results should on the average be worth far more than the expenditure.

Such deliberations should act as an incentive to every investigator and developer of new things to give whatever he can afford to the common fund of knowledge. We know, of course, that there are certain points which even the most generous man should rightly keep to himself. Every chemical and metallurgical failure injures indirectly each metallurgist and each chemist, for it justly makes capital more suspicious and cautious about entering on a new enterprise. The optimistic boastings in the columns of the Sunday newspaper about certain alleged triumphs of engineering do incalculable harm. If there are enough well trained engineers, the charlatan will be less prominent. There is work enough to keep all good men at work busy and active. Some narrow spirits do not realize this. We hope that the selfish engineer will not only not receive the confidences of his associates, but that he will also lose directly from the general distrust, caused by the short-sighted policy which he and his ilk engender. The broadest and grandest successes are not envied, but admired, when a broad man is generous in giving information about his trials. In the narrower field of electrochemistry and electrometallurgy the American Electrochemical Society—and, we modestly hope, this journal—have had in the past an influence in systematizing the experience of all and thus making experimental work more efficient. This has reacted on all, and, we trust, has made the obscurantists see the error of their ways. Many of the failures of the past might have been avoided if experimenters had pooled their experience. This fact seems now to become clearly understood, and in the future we can hope that the evolution of the metallurgical industry will not be fraught with too many failures; that the engineers of the future works will not shoot too blindly in the field of practical endeavor; that when a place is to be aimed at they will bring out a gatling gun of experience and thoroughly cover that one spot. To effect this we will try to do our share, so far as our ability and the good will of our contributors extends, as a clearing-house of metallurgical ideas.

American Electrochemical Society.

The date for the spring meeting to be held in Boston, Mass., has now been definitely fixed; it will be held on the 25th, 26th and 27th of April. The local committee, with Prof. W. H. Walker as chairman, is making arrangements so as to make this meeting a most memorable one. There will be especially fine facilities for making exhibits and experiments in the convention hall.

New York Section of the American Electrochemical Society.

The first meeting of this section in the present season was held on January 24, in the hall of the Chemists' Club, Dr. C. A. Doremus being in the chair. Dr. E. F. Roeber presented a paper giving a summary of the various uses to which electricity may be put in iron and steel metallurgy. After some introductory remarks on the use of blast furnace gases for driving gas engines—which has led to a new commercial source of electric power—and on the special reasons which have caused European metallurgists to become fascinated with applications of the electric furnace in iron and steel industries, he touched briefly on the use of electricity for all the various power purposes in modern steel plants, and on the use of electricity for electromagnetic and electrostatic separation—two kinds of concentration of ores—and then passed over to the use of electrolysis for refining iron. The author described the method of Burgess and Hambuechen for producing almost absolutely pure iron by electrolysis, and then referred more briefly to Edison's manufacture of the steel containers of his storage battery by electrolytic deposition upon a mould. The main part of the paper dealt with the discussion of applications of the electric furnace, the following four problems being separately discussed: Production of ferro-alloys; manufacture of high-grade special steels to compete with our present crucible steels; the production of structural steels to compete with Bessemer and open-hearth steel; the reduction of iron ore to produce pig iron. The author emphasized the importance of the experimental evidence contained in the recent report of the Canadian Commission and summarized their conclusions and the evidence on which the conclusions are based. Dr. Eugene Haanel, the head of the Commission, had courteously placed a number of copies of the report at the disposal of the Section.

In the discussion which followed, Mr. J. T. Morehead gave very interesting details on the work now being done in the production of ferro-alloys by the Wilson Aluminium Company. He stated especially that the method of von Kuegelgen and Seward (our Vol. II., page 468) is now used by the company on a commercial scale; the electric furnace process is so conducted that the consumption of the carbon electrodes is a minimum. Under such conditions the alloy will contain much carbon, which is afterwards removed by means of metallic calcium. He also mentioned the use of silicon as a [illegible]ng agent. The company makes the calcium metal by an electric method. Dr. George F. Kunz gave some interesting notes on natural and artificial diamonds and carborundum with reference to experiments of Moissan; Mr. C. O. Mailloux gave some humorous reminiscences of the time when aluminium was first made commercially, while the difficulty was to find uses for it. Mr. A. J. Rossi, the distinguished titanium expert, had sent a communicated discussion, which was essentially the same as his letter on page 53 of this issue.

Dr E. Durant then read a highly theoretical paper, in which he referred to an electrical gyroscope which was exhibited and shown in operation.

Tantalum and the Tantalum Lamp.

The attempts of getting a more economical incandescent lamp have mainly gone towards finding a material which has a very high melting point. The Nernst lamp and the Osmium lamp represent examples of work done in this direction. In the chemical research laboratory of the Siemens & Halske Co., of Berlin, Dr. W. von Bolton has been making extended experiments with various materials, and has specially studied vanadium, niobium, and tantalum. The outcome of this research is the tantalum lamp, which appears very promising. Dr. von Bolton reduced potassium tantalo-fluoride in the manner described by Berzelius and Rose, and found that the finely divided tantalum so produced became fairly coherent on rolling, so that by this treatment metallic strips could be made of it. On the other hand, it was tried to bring tantalum oxide into the shape of a filament by mixing it with paraffine, and to reduce it directly into the form of a metallic thread. In these experiments there was for the first time observed a minute globule of molten tantalum, which was of sufficient toughness to permit hammering and drawing into wire. Following up this observation, tantalum powder was melted in a vacuum and then it was found that the highly heated metal parted with the gases it contained. In this manner the first filaments of pure tantalum metal were produced, which, however, were quite small. When they had been used in lamps with promise of good results, an attempt was made to devise a definite process of purification.

Potassium tantal-fluoride reduced to metallic powder, contains a small proportion of oxide and of hydrogen absorbed during the reduction. When the powder was melted in a vacuum the oxide and absorbed gas disappeared and a reguline metal remained, and on carefully remelting, no appreciable impurities could be detected in it.

The chemical properties of this pure tantalum are very remarkable. When cold, the material resists chemical agents strongly; it is not attacked by boiling hydrochloric acid, aqua regia, nitric acid or sulphuric acid, and it is also indifferent to alkaline solutions; it is attacked solely by hydrofluoric acid. Heated in the air, it assumes a yellow tint at about 400° C., like steel, and also like steel the tint changes to dark blue when the tantalum is exposed for some time to 500° C., or for a shorter time to 600° C. Thin wires of it burn, when ignited, with low intensity and without any noticeable flame. It greedily absorbs hydrogen as well as nitrogen, even at a low red heat, forming with them combinations of a metallic appearance, but rather brittle. It combines with carbon very easily, forming several carbides, which, as far as they are at present known, are all of metallic appearance, but very hard and brittle. The product which Moissan thought was tantalum was, according to von Bolton, clearly a carbide of this kind, or an alloy of a carbide with pure tantalum. While Moissan ascribes to the metal the specific gravity 12.8, great hardness and brittleness; these are, according to Dr. von Bolton, not properties of pure tantalum. The specific gravity of Bolton's tantalum, when purified by fusion and drawn into wire, is 16.8. It is somewhat darker than platinum and has a hardness about that of mild steel, but shows greater tensile strength. It is malleable, although the effect of hammering is relatively small, so that the operation must be rather long and severe to extend the metal into a sheet. It can be rolled as well as drawn into very fine wire. Its tensile strength as a wire is remarkably high, and amounts to 95 kilos per square millimeter, while the corresponding figure for good steel is 70 to 80 kilos., according to Kohlrausch.

The electrical resistance of the material at indoor temperature is 0.165 ohms for a length of one meter and a section of 1 square millimeter (specific conductivity as compared with mercury 6.06); the temperature coefficient is positive and has a value of 0.30 between 0° and 100° C.; at the temperature assumed by the incandescent filament in the lamp under a load of 1.5 watts per c. p., the resistance rises to 0.830 ohms for a length of 1 meter and a section of 1 square millimeter. The coefficient of linear thermal expansion between 0° and 60° C. is 0.0000079, according to experiments made by the Imperial Normal-Aichungs Commission. Fusion is produced by a gradual softening which appears to extend over a range of temperature of several hundred degrees. The specific heat is 0.0365, so that the atomic heat is 6.64, which is in accord with the law established by Dulong and Petit.

The lamp department of the Siemens & Halske Co. appears to have successfully overcome the difficulty of making a tantalum filament for use of the lamp in any position. The energy consumption is between 1.5 and 2.5 watts per c. p.

Current Notes.

Franklin Institute.—At a recent meeting of the Franklin Institute, Prof. Wilder D. Bancroft, of Cornell University, presented a paper on the chemistry of electroplating.

Electrolytic Iron.—We have received from Prof. Charles F. Burgess, a small but magnificent sample of his pure electrolytic iron, the production of which was described in great detail in our Vol. II., page 183. We understand that the Carnegie Institute has granted funds to Prof. Burgess to enable him to continue his splendid researches in this field.

Aluminothermics.—The Goldschmidt Thermit Company has presented to the National Museum at Washington their entire exhibit at the St. Louis World's Fair, at the suggestion of the museum authorities. The collection gives a complete picture of various applications of aluminothermics, and consists of large and bulky pieces of pure, rare metals, free from carbon, such as chromium, manganese, ferro-titanium, etc.; diagrams and models showing various applications of thermit; numerous pieces—in particular, welded girder rails—showing the successful operations performed by the process in joining and fusing iron and steel, and metals and specimens of the methods used.

Electric Furnace Curiosities.—We have received from Messrs. FitzGerald & Bennie, of Niagara Falls, two interesting samples which came out of recent experimental electric furnace runs, and which show clearly the effects of high temperatures from small radiating surfaces. One of them is a section of a long tube of pure quartz, and is pretty well fused on the inside; the heat was applied from a bare carbon rod which had been heated to a very high temperature. (This is precisely the method of producing fused carbon articles, quite recently patented by Prof. Elihu Thomson). The other specimen is from an experiment with a very small carbon rod, surrounded by a porcelain tube; enough heat was generated in this small rod to fuse the porcelain so that it ran down into the lower portion of the tube and finally filled the tube up to the carbon. The specimen shows some light greenish material which is probably some silico-carbide, while some small globules sticking to the small carbon rod are probably metallic silicon, reduced from the silica in the tube in contact with carbon, or they may be aluminium silicide.

Calendar.—The Laclede Fire Brick Manufacturing Company, of St. Louis, Mo., issued a diabolically striking and amusing calendar, on which "His Satanic Majesty" expresses his satisfaction with fire-bricks, which are "the only material to stand my heats."

CORRESPONDENCE.

The Electric Furnace in the Iron and Steel Industry.

To the Editor of Electrochemical and Metallurgical Industry:

Sir.—The subject of the use of the electric furnace in the iron and steel industry is of such a vast importance, and one which has given rise to so many conflicting opinions, pro and con, that it is not possible to do justice to it in a short note. The able report of the Canadian Commission, covering, as it does, the most improved methods used on the Continent, speaks for itself and its conclusions, as a whole, cannot be contested. We will confine ourselves in our remarks strictly to the metallurgical part of the question, leaving to others to discuss the disposition of the furnace, and other details. We may say that, in a general way, the whole question is one of economy of the electric treatment in comparison with other metallurgical processes employed for the same purposes. As to the possibilities, they have been demonstrated beyond question, both as to results obtained and qualities of the products.

Economy is, to a certain extent, a relative word. As regards steel, for instance, crucible steel commands a much higher price than open-hearth steel, which itself is far superior to converter steel; so that the economy of an electric treatment may well be demonstrated for a special quality of steel, while it may be different for another. In what is called the crucible steel method, the metal is melted in a vessel of small dimensions, and is protected against the obnoxious influences of certain gases, such as those of the open-hearth. Electricity can certainly compete favorably with the present method for crucible steel.

Even as regards open-hearth steel the operation involved in the method of making steel can be secured electrically, as surely and as advantageously and under conditions which secure a metal of superior quality, hence commanding a higher price, and thus upsetting the lack of economy were it proved to exist; we mean for special steels. Many people, when electric steel is mentioned, understand it to mean steel obtained directly from iron ores electrically, while all the processes which have been successfully carried on in Europe are what is called the pig and scrap process, or the pig and ore scrap or mixed process, as they are carried out now in the open hearth. In the best apparatus examined and discussed, the electricity is used as a source of heat for raising the temperature of the charge. This electric heating can be done very successfully, very cleanly and even economically when the cost of fusion is much less important than the soundness, toughness and, in general, the superior qualities of the steel, as is the case in the manufacture of the finer grades or special steels.

There is quite a difference between the amount of heat necessary to bring pig iron and scrap to a certain temperature, and to melt them, on one hand, and reduce iron from its oxide to the metallic state, on the other hand; 1796 calories are required to reduce 1 kg. of iron from ferric oxide, and some 200 calories more for impregnating the metal with carbon, phosphorus and sulphur—for pig iron, and also to a less but still important extent for steel—this, with the heating of the materials to the proper tempertature and melting the metal, represents some 2500 calories or thereabouts per kilogram obtained. On the other hand, some 500 or 600 calories at most (say ¼ to 1/5 of the former figure) are sufficient, if no reduction of oxide of iron is resorted to (pig iron and scrap process), or if but a limited amount of it is to be reduced (mixed process).

We do not take into consideration in giving these figures, such heat or part of it as may be supplied by the combustion to CO_2 of the CO resulting from the reduction of ferric oxide by carbon. If we take in consideration the equivalence of heat and mechanical energy, a certain amount of heat corresponds to a certain amount of horse power in a given time, and it is immaterial if the heat appears as fuel burnt (like in the blast furnace), as gases (like in the open-hearth), or as electrical energy developed. It can be easily shown that starting from pure oxide of iron reduced by carbon, with all other contingencies for making pig iron, it is necessary to supply the heat equivalent to some 145 hp. per gross ton of pig iron made in twenty-four hours, and if we adopt an 80 per cent duty for electric smelting, we come to 175 hp. per gross ton-day. It is true that some of the heat of CO forming CO_2 can be made available. But if, on the other hand, we consider that we will have practically some slag to meet; that there will be a loss of heat by radiation, and a loss of electricity from the supply to the furnace; that the composition of the ores will vary; that even their physical or chemical state will be different, the above figure ought to be considered in regular practice as a very favorable one, due weight being given to causes of occasional imperfect working inherent in this as in any other metallurgical method; in certain cases 200 hp. per gross ton-day would not be exaggerated.

From the "Canada Report" we find that an average of the horse-power-year per ton of steel was for Kjellin's and Héroult's process (mixed process) 0.16 hp., which corresponds to 58 hp. per ton of ingot-day, and for the pig and scrap process (Keller) 46 hp. per ton of ingot-day. For pig iron the figures are: 0.525, 0.53 per hp.-year (Héroult process and first run of Keller); this gives say, 0.53 hp.-year per ton of pig iron or 189 hp. per gross ton of pig iron in twenty-four hours, which agrees well with the figures we have given above.

At this rate, counting even only on the best possible condition, theoretical ones, so to speak, in one case we need between 150 to 200 hp. for 1 ton (gross ton) pig iron per day; that is, for a production of 500 tons a day, such as is frequent and current practice with the Pittsburg blast furnace, 75,000 to 100,000 electric horse-power would be required. We have never claimed that electric smelting could compete for such daily productions with the blast furnace, but we believe that for limited or moderate local production an electric plant of moderate size involving as expenses of installation only turbines and dynamos, and the collecting of smaller powers to a central plant, could be established so as to supply the current at a cost of $8 to $10 per hp.-year or even less.

Under such conditions, with ores near the furnace and which could be had at practically the cost of moving, or say at 0.75 to $1.00 a ton or so, a local industry to supply local wants could be successfully carried on. In electric smelting the carbon being used for reduction only, not more than ¼ to 1/3 of the amount required in a blast furnace would be required; furthermore, its physical state is not like in the blast furnace, a condition *sine qua non* for use. Charcoal, made from twigs and small branches can be used in an electric furnace. The advantage of the latter is the cheapness of its construction, compared with that of a blast furnace. No machinery for blowing nor for the heating of the blast are necessary, no expensive stack is required. It adapts itself to an increase or decrease of production by the addition of a furnace put up at a very moderate cost, or the shutting off of a switch. Finally the metal thus obtained is more homogeneous, close-grained, in short, of a superior quality, and may command and has commanded a higher price.

In short, so far as pig iron is concerned, electric smelting is a question of cheapness of establishment or supply of horse-power; cheapness of ores used in loco, favorable location, limited production for local consumption, or under conditions where the lack of proper fuel excludes the possibilities of a blast furnace plant.

As far as ordinary steel is concerned, the same may be said. Electrical plants capable of making 1800 tons of steel and turning them into rails, as it is done with two converters in twenty-four hours, cannot be thought of, and such converter plants are in daily operation in Pittsburg. As to open-hearth steel, there are now in operation open-hearth furnaces with an output of 200 tons of steel, while 50 to 100 ton open-hearth furnaces are of frequent use, and as far as quantity of steel made in a day is concerned, electric smelting would hardly prove economical. However, from the figures given above, since only some 50 to 60 hp. are required per ton of steel (gross ton) per day, a plant of 200 tons would only require 10,000 to 12,000 hp. Large as it is, this figure is admissible. If, on the other hand, special steels are made, then the daily production falls considerably below the large figures mentioned. The electric process then assumes an equality as to cost, and certainly a superiority as to quality of product.

For the production of crucible steels, the finer qualities of steel and special steels, electric smelting, we are confident, will at some near future assume its superiority, and will come more and more into use, and the same may be said of ordinary open-hearth steel if moderate productions are admissible.

We should add that in the above we have not intended to give the items of the balance of heat of each process, but merely approximate figures, which are at least sufficient for comparisons.

Auguste J. Rossi.

New York City.

Zinc Made in the Electric Furnace.

To The Editor of Electrochemical and Metallurgical Industry.

Sir.—With reference to the article of Dr. F. Meyer on the present status of the zinc industry, and an editorial discussion of the same subject, published in your January issue, I would like to make a few remarks on the properties and uses of zinc produced in the electric furnace of De Laval.

This zinc was placed on the market in Europe about a year ago. With respect to qualify, this pure zinc G. D. L. is able to compete with the best kind of zinc on the market, such as "Sterling" and "Vieille Montagne extra pur A," as will be seen from the following analyses made by me. Like Sterling zinc and Vieille Montagne extra pur A, the pure zinc G. D. L is sold under guarantee not to contain more than 0.1 per cent total impurities:

	Sterling.	Vieille Montagne Extra pur A.	Pure Zinc G. D. L.	
Number of Analyses.	9	3	2	
	per cent.	per cent.	per cent	per cent.
Pb	0.03 to 0.05	0.05 to 0.07	0.06	0.03
Cd	....	....	None	None
As	....	....	None	None
Fe	0.01 to 0.03	....	0.01	0.01
S	....	....	0.00 (3)	....

Such pure kinds of zinc are used in the brass industry. For ordinary kinds of brass, "Hohenlohe" zinc (containing in the average 1.1 per cent Pb) or similar kinds of zinc are sufficient, while for brass rods and for certain types of brass wires often some lead is added, in order to facilitate the working on the lathe. On the other hand, it is necessary to use the purer and more expensive brands of zinc for certain purposes, like the manufacture of brass sheet for cartridge shells.

According to the specifications of most European governments, this brass must not contain more than 0.2 per cent Pb and must satisfy certain mechanical tests. It is easy to calculate that for this purpose it is necesary to use the pure brands. Moreover, very small quantities of As and S have, in this case, an appreciable bad effect. If for the manufacture of cartridge shells pure G. D. L. is used, instead of "Sterling" or "Vieille Montagne extra pur A," which have proven suitable in many years' tests, no difficulties whatever are experienced in the method of manufacture, nor does the proportion of faulty products pass beyond the usual percentage.

At present, pure zinc G. D. L. is a little (though very little) cheaper in Europe than Sterling zinc and Vieille Montagne extra pur A.

C. Offerhaus.

Holcomb Rock, Va.

Resistance Furnace for Crucibles.

By Francis A. J. FitzGerald.

In making experiments with small electric furnaces, it is by no means easy to form an opinion as to what may be done with a very much larger furnace working on a commercial scale. In certain cases it is impossible to predict with certainty what the results will be when working with the larger furnace, and the only method of arriving at a conclusion of any value is to install the large furnace and try the experiments with it. The great difficulty we are met with in the design of electric furnaces is the lack of data on which to base our calculation, for though there can be no doubt that many of these data are in existence, they cannot be obtained. Most processes in which electric furnaces are used are patented, and the details connected with the subject are carefully preserved secrets. Nevertheless, a great deal of experience may be gained from the study of small furnaces that will prove valuable in the design of large ones, the more so in that certain causes of inefficiency due to poor design are made much more obvious when working on a small scale.

One of the most important considerations in the building of all electric furnaces is that of heat insulation, and in no way is this made more evident than in experimenting with small furnaces, for example, furnaces using from 20 to 50 kilowatts. If two blocks of iron of similar form, but of different dimensions, are heated to the same temperature, then, *ceteris paribus;* the rate of cooling will be proportional to the squares of the dimensions, the quantity of heat in the blocks will be proportional to the cubes of the dimensions, therefore the small block will cool faster than the large block.

If a given mass of iron in the form of a sphere is heated to a certain temperature, and a long rod of iron of the same mass is heated to the same temperature, then, under similar conditions the rod will cool faster than the sphere. If two spheres of iron of the same mass are heated to the same temperature, and one of them is surrounded with a good heat insulator, while the other is exposed to the air, the latter will cool more rapidly.

These observations are absurdly trite, yet they seem to be continually disregarded in the design of electric furnaces. Now, even under the most favorable circumstances, as to price of power, if we simply consider the cost of a unit of heat, that generated electrically is more costly than that generated by burning fuel. Therefore, in dealing with electric furnaces, one of the most important points to be kept in mind is efficient heat insulation. One of the reasons for using electrically generated heat is because with it very much higher temperatures can be attained than with burning fuel. Now, the rate at which heat is conducted or radiated from a body is a function of the difference of the temperatures of the body and its surroundings, and hence heat is conducted away more rapidly in the case of electric furnaces.

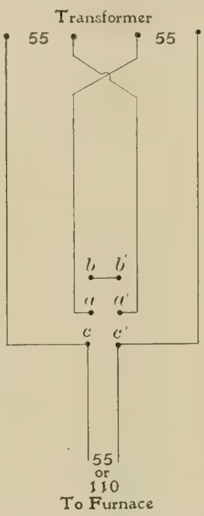

FIG. 1.—DOUBLE-THROW SWITCH.

This is merely one example of many points that can be studied in dealing with small furnaces, and is a very obvious example, too, as anyone experimenting with a small furnace will soon find out.

Many experiments and investigations can be conducted with furnaces using 20 kilowatts or less. If an alternating current, which is more satisfactory, is used, it will probably be brought to the furnace room at high voltage, and if so, one of those transformers should be used which can be connected up so as to give two secondary voltages. If possible, a transformer and regulating device should be used, so that a considerable range of current and electromotive force can be obtained. This, of course, adds appreciably to the expense of the apparatus, and if only an ordinary transformer is available, giving two voltages, then it will be convenient to have it connected so that 110 or 55 volts can be obtained. For a 20-kilowatt transformer the leads should be connected to a double-throw, two-pole, 500-ampere switch, as shown in Figs. 1 and 2. The upper terminals of the switch *b*, *b'* are connected by a copper bar, while the leads from the two 55-volt windings of the transformer are connected to *c* and *a'*, and to *a* and *c'*, respectively, so that when the switch is thrown up, connecting *a* and *a'* with *b* and *b'*, the windings are in series, and thrown down to *c* and *c'*, they are in parallel. In this way it is possible to have a maximum current of 180 amperes at 110 volts, or 360 amperes at 55 volts. This is about the cheapest arrangement that can be used, and is also satisfactory for experiments with small arc furnaces. It is, however, very far from being satisfactory for certain resistance furnace experiments where, as we shall see, a current of low voltage and very high amperage is required.

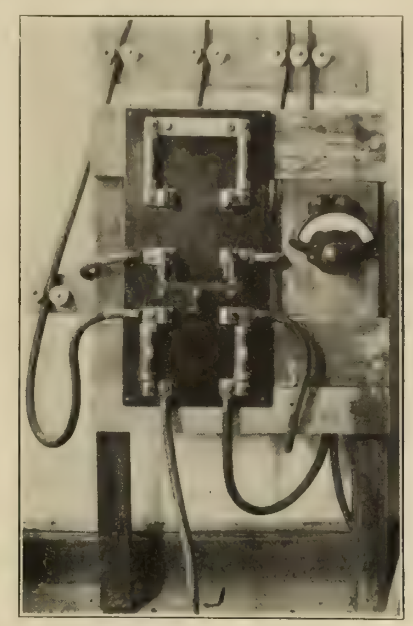

FIG. 2.—DOUBLE-THROW SWITCH.

In experimenting, even with so small a furnace as one using 20 kilowatts, it is well to have a good-sized bench for the work. This may be constructed of two-inch boards, supported on 4 x 4 inch legs placed at frequent intervals and properly braced to give rigidity to the structure, which will be called upon to stand heavy weights. The bench, which may be about 10 x 4 feet, having been set up, strips of wood are nailed along the sides and ends, so as to form a tray about 1 inch deep, and this is filled to a depth of about ½ inch with sand.

A floor of brick is now laid on the bench, and on this floor is built the brick work, cellular structure shown in Fig. 3 in isometric projection. The cells of the lower row are empty, but those of the upper row are filled with a good heat insulator, such as asbestos, or, better still, infusorial earth, which is one of the best heat insulators that can be obtained, and if pure, is much more refractory then asbestos.[1]

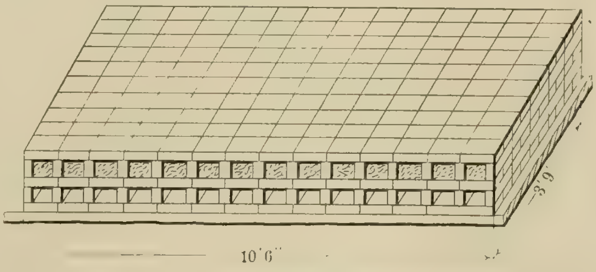

FIG. 3.—CELLULAR STRUCTURE OF BRICKWORK FOR ELECTRIC FURNACE.

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[1] I am told that a good grade of infusorial earth is hard to obtain. An excellent material is put on the market by the Lake Region Mining Company.

At first sight this bench may seem to be a needlessly elaborate structure, and doubtless it would be if furnace experiments only lasted half an hour or an hour, but if a furnace run lasts nine or ten hours, a less elaborate structure would not be suitable. Without the heat insulation in the upper row of cells the losses would be very serious, and by having the lower row of empty cells, it is possible to detect accidents, such as the fusion of the bricks forming the floor of the furnace. Such accidents as that are not infrequent in electric furnace experiments when long runs are made.

As an example, a method of constructing a small furnace for heating crucibles, may be described. In Fig. 4 a partial plan, and in Fig. 5 a partial vertical, longitudinal section of such a furnace is shown. In building the furnace the terminals *T* are first set up. These may consist of two carbons of square cross-section with a copper plate clamped (clamp not shown in figure) between them, to make contact with the cables carrying the current. It is preferable to use amorphous carbon electrodes in this furnace, since graphite electrodes will cause a serious loss of heat by conduction. The carbon terminals, as shown in the figure, are needlessly large.

Bricks are then built around the terminal, being cemented carefully with fire clay, so as to keep the carbons protected from the air. During a long run the inner ends of the carbons will become very hot, and, unless the brick work about the terminals is carefuly plastered with fire clay, the carbons are likely to be oxidized.

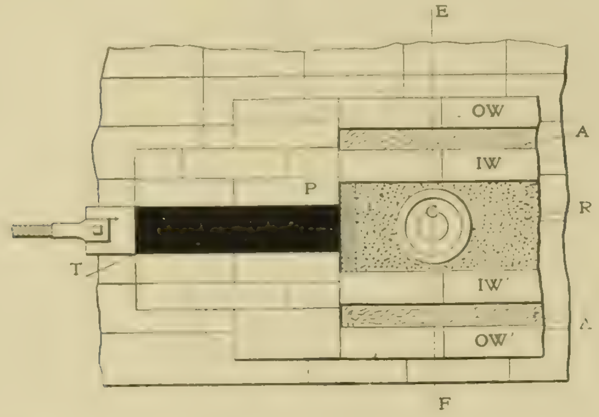

FIG. 4.—PLAN OF ELECTRIC FURNACE FOR HEATING CRUCIBLES.

The inner walls, *IW* and *IW'*, of the furnace are next built up, these being made slightly higher than the depth of the resistor, which consists of granular graphitized coke. In the resistor the crucibles *C* are embedded. Contact between the resistor *R* and the terminals is made by *P*, which consists of graphite powder. The making of this connection is very important, for if not done properly there will be a considerable generation of heat at the point and a consequent waste of energy. Not only does the current pass from a conductor of small cross-section and low resistivity, to a conductor of large cross-section and high resistivity, but even if the terminal had the same cross-section as the resistor, the connection would not be sufficiently good if the grains of the resistor simply rested in contact with the terminal. If an imaginary plane is pictured as passing vertically through the resistor, it will readily be seen that the parts of the resistor on either side of the plane will make much better electrical contact with one another than the resistor would make with a solid slab of carbon against which it rested.

In making this connection in a small furnace, such as is described here, a piece of stout cardboard of the right size should be inserted in the trough about one inch from the terminal, then, as the granular resistor is put into the trough, powdered graphite is put between the cardboard and the terminal and tamped into place till it forms a hard, compact mass. Then more graphite is added and again tamped down, the cardboard meanwhile being withdrawn a little way, so as to allow contact between the graphite and the resistor material. With proper care a most excellent joint is made in this way. It is the kind of joint used by Acheson in the construction of his carborundum furnace.

When the crucibles and resistor are in place the outer walls, *OW* and *OW'*, may be built up, as shown in Fig. 4, leaving a space between them and the inner walls. This space is filled with a good heat insulator, such as infusorial earth. If this heat insulation is omitted the losses from heat radiation are serious, as may readily be observed by running such a furnace with and without the outer wall and insulating space. The walls of the furnace, as was pointed out before, are made somewhat higher than the depth of the resistor, so that refractory tile or bricks may be laid over the resistor to diminish radiation losses. In dealing with a large furnace it would be worth while to build a cellular cover to go over the top of the furnace.

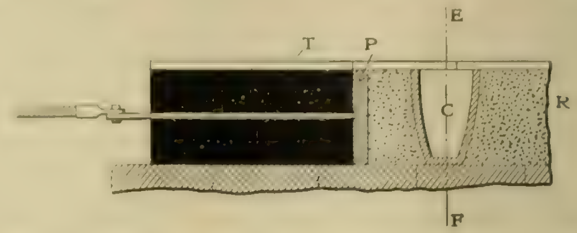

FIG. 5.—VERTICAL SECTION OF ELECTRIC FURNACE FOR HEATING CRUCIBLES.

If the furnace is to be of a permanent type, it is necessary to construct some form of receptacles for the crucibles. As described, when a crucible is removed from the resistor the granular carbon will fall in and the crucible cannot be returned. It is a simple matter to construct receptacles that fit the crucibles and embed the form in the resistor. These receptacles must, of course, be refractory, and should have as good a conductivity as possible, which indicates the use of silico-carbides for the material.

A furnace built in the way described above will work fairly well, but examination soon shows that its design is by no means perfect. Inspection of Fig. 4 shows that the sectional area of the resistor is least at a plane drawn through the line E F, hence the resistance at this part of the resistor is greatest, and consequently the rate of generation of heat will be greatest, and the heating of the crucible will not be uniform over its whole surface.

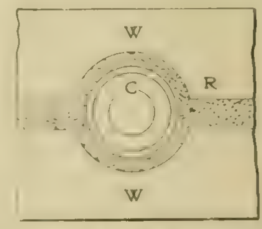

FIG. 6.—PLAN OF RESISTOR.

In order to get even heating, the plan of the resistor should be like that shown in Fig. 6, where the sum of sectional areas of the branches of the resistor passing round the crucible C is equal to the sectional area of the undivided resistor. As far as a horizontal plane drawn through the furnace is considered, this will give uniform heating, but the heating will not be uniform if we consider a transverse vertical plane drawn through E F, for two reasons: First, on account of the shape of the crucible, which gives the resistor a larger cross-section at the base than at the top; second, because the resistivity of the resistor is less at the bottom than at the top. The weight of the upper parts of the resistor compresses the lower parts, diminishing their resistivity. Thus, if we imagine the resistor divided into a series of horizontal layers, the resistances of unit length of the layers will continually decrease as we go lower down. From Ohm's law it may be shown that in a series of parallel circuits the energy developed in each will be inversely proportional to its resistance. In order, therefore to obtain uniform heating, the cross-section of the resistor should conform to that shown in Fig. 7.

In making a single experiment it is not necessary to go into these refinements of construction, but the conditions may be roughly approximated. If, however, a permanent furnace is to be designed all these details should be kept in mind. But before building the permanent furnace it is necessary to make rough experiments in order to obtain various necessary data for the design. Probably in the future certain general rules and quantities wil be available which will permit of the design of a furnace with the same accuracy that we now design a boiler for a given purpose; but at present this cannot be done, and the experimenter must be content to arrive at his final results by trial. As an illustration of this, let it be assumed that in a certain furnace there are a number of crucibles to be heated; that the amount of energy required for each crucible is known; that the e. m. f. and current available are fixed, then it becomes necessary to determine the sectional area of the resistor and the size of grain to be used. In another article[2] an experiment was described using two granular carbon resistors and data were obtained from which the resistivity of each may be calculated. The resistors were identical, except that one of them (G1) was composed of grains of graphitized coke that passed through a screen having 5 meshes to the linear inch, and were held on a 6-mesh screen, while the other (G2) was made of grains passing through a 3-mesh screen and held in a 4-mesh screen. The dimensions of each resistor, as tested, were as follows:

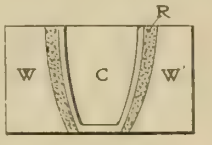

FIG. 7.—SECTION OF RESISTOR.

Length	42 cms.	16.5 inches
Width	5 cms.	2.0 inches
Depth	5 cms.	2.0 inches

Taking the readings at two parts of the experiment, we have the following:

	Volts G1.	Volts G2.	Amperes.
1.	44	30	138
2.	46	32	100

When the first set of the readings given above was taken, weights were resting on the resistors; when the second was taken, the weights had been removed. From these data the resistivities of the resistors may be calculated, and expressed either as the resistance of a cube measuring one centimeter or one inch on the side:

	G1 (cm.)	G2 (cm.)	G1 (inch)	G2 (inch)
1.	0.19	0.13	0.075	0.051
2.	0.27	0.19	0.11	0.075

These figures show clearly the great difference in resistivity found in resistors composed of grains of different size, and also the effect of pressure on the resistivity. The marked effect of the latter becomes still more evident when it is remembered that the area of the resistor to which pressure was applied during the experiment was less than one-third of the total surface area.

For purposes of comparison let us calculate the resistivity of the resistor of a commercially worked furnace with the laboratory experiment given above. Taking the published descriptions of the carborundum furnace[3] we obtain the following data, which are arranged in the form of a table:

Length of resistor (core)	426 cms.
Sectional area of resistor	2210 cms.^{2}
Resistor consists of ordinary coke,	
E. M. F. when furnace at load (746 kw.)	250 volts
Current	2960 amps.
Resistor consists of graphitized coke,	
E. M. F. when furnace at load (746 kw.)	150 volts
Current	4970 amps.
Resistor is now graphitized,	
E. M. F. at end of run	100 volts
Current	7460 amps.

From this table the resistor resistivities have been calculated, the resistivity being the resistance of a centimeter cube:

	Resistivity.
Ordinary coke, on reaching load	0.44
Graphitized coke, on reaching load	0.16
End of furnace run	0.070

The best figures to compare in the experiment and in the carborundum furnace are those of G2 in the former and the end of the run in the latter. It will be observed that they differ considerably, the resistivity of the carborundum furnace resistor being much lower. This is to be explained by the greater size of the coke particles comprising the furnace resistor, for these are from ½ to ¾ inch in diameter; the temperature of the furnace resistor was somewhat higher than that of G2; there is a certain amount of current leakage in the carborundum furnace, and finally the pressure on the furnace resistor is much greater than on the resistor taken for experiment.

The resistivity of the graphitized coke furnace resistor, when it first comes to load, is somewhat higher than that of the resistor G2 when the weight was put on the latter, but at that time it was very hot, while the furnace resistor would be nearly cold.

In the experiment with the resistors, an increase in resistance that was probably due to the formation of carbides was observed, and that enters as another disturbing factor in the comparison of resistivities under various conditions.

This somewhat lengthy discussion of the resistivity of granular carbon resistors seems to be worth while, in order to show the difficulty of the subject, and that our knowledge of it is still very limited. This being so it is not possible to give any definite figures for the calculation of resistors of this sort. It shows, moreover, that the best way of arriving at satisfactory results in the design of such a furnace as has been described is to make preliminary experiments that approximate as closely as possible the conditions that will exist in the furnace. It is of the utmost importance that we study these conditions with great care. Thus, the size of the grains of carbon composing the resistor must be determined by careful screening. A resistor of coarse-grained carbon has, as we have seen, a very different resistivity from one of fine-grained coke, hence the importance of knowing just what is the size of the grains used. Here, too, it may be noted that a resistor containing a quantity of powdered carbon, or fine grains mixed in with the grains that are of the proper size, will cause a lot of trouble, especially in dealing with large furnaces. The reason for this is that the powder will not remain distributed evenly throughout the resistor, and there will consequently be differences in resistivity in different parts of the resistor.

In making the experiments it is important to know the physical and chemical states of the carbon. The resistivities of the carborundum furnace resistors shows plainly the great difference in the resistivities of graphitized and ungraphitized coke. If the resistor used in a furnace is composed of ordinary coke, and its temperature is raised, while in use, to a higher temperature than it has ever been at before, there will be a permanent change in the resistivity of the carbon grains, and consequently in the resistance of the resistor. As may readily be seen, this causes a troublesome irregularity in the works. If, on the other hand, graphitized coke is used, this source of irregularity is eliminated, for in the process of graphitizing, the coke has been raised to the highest attainable temperature and has consequently reached a stable physical state.

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[2] Electrochemical Industry, Vol. II., No. 12 (December, 1904), page 492.

[3] Journal of the Franklin Institute, February, 1897.

Another advantage in the use of graphitized carbon is its purity. In the process of graphitization it has been raised to such a high temperature that a great part of the impurities present have been vaporized. This will naturally diminish irregularities in resistance due to the formation of carbides. It has been demonstrated that silica has a distinctly appreciable vapor tension at temperatures well below that of its fusion, and certain compounds of silicon and carbon are formed at relatively low temperatures. If, then, there is much silica present in the carbon used for the resistor, even if the temperature is not unusually high, there will be a slow formation of carbides which will appreciably effect the resistance.

Keeping all these details in mind, the data for the design of the permanent furnace may be obtained by experiment. The permanent furnace will then be similar to the plan shown in Fig. 8. Here *T1* is a furnace terminal, *T2* the terminal common to the regulator arc furnace resistors, *T3* the terminal of the regulator resistor, *P* is the graphite connection between the furnace resistor and the terminals, *A* the heat insulator between the outer and inner side walls *OW* and *IW*, *H* a refractory material containing the furnace resistor, *V* granular carbon forming the regulator resistor. In building the furnace resistor it is best to make a model of one section in wood or plaster, and having built the side walls of the furnace, put the model in place and pack the refractory material round it. It is understood that the refractory material is mixed with some binding substance, so that the model of the resistor can be withdrawn. Thus, the various sections are built in a perfectly uniform manner. The receptacles for the crucibles are put in next, then the granular carbon to form the resistor is introduced and the connections (*P*) to the terminals carefully made. In putting in the granular carbon care must be taken to avoid ramming it in tightly in one place and leaving it loose somewhere else. It ought to be packed down to a certain extent throughout, otherwise, when in use, it will become more tightly packed in one place than another, through mechanical jarring, and consequently the resistivity will not be uniform. When the resistor is in place the furnace is ready for final adjustment.

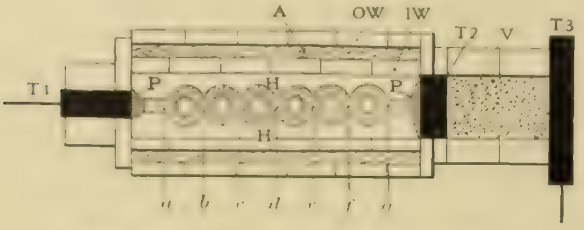

FIG. 8.—PLAN OF FURNACE FOR CRUCIBLES.

Small graphite rods are now thrust into the resistor at *a*, *b*, *c*, etc., and wires attached to the rods are brought to a suitable switching device, so that the rods can be quickly connected with a voltmeter. A small amount of granular carbon is put into the regulator, so that its resistance will be somewhat high and the current is then thrown on. After waiting for a short time the current, as indicated by the ammeter, becomes steady, and then the volts between *a* and *b*, *b* and *c*, etc., are determined. They will be found to differ, unless the resistor has been put in with absolute uniformity. If it is desired to heat all the crucibles uniformly, the volts across *a* and *b*, *b* and *c*, etc., must be the same, so now adjustment is made by taking away some carbon where the volts are too low and adding carbon where they are too high. Final adjustment is made by cautiously tamping down those places where the volts are too high. From the preliminary experiments the amount of energy required per crucible has been determined, and, consequently, the resistor must be adjusted to obtain that result.

In making an experimental furnace it is sometimes desirable to have it so arranged that at the same time different parts of the furnace are at different temperatures. A furnace of this kind can easily be built along the lines described above, the resistivity of the resistor being made greatest around those crucibles where the highest temperatures are desired. The best way of arriving at this result is to keep the sectional area of the resistor the same throughout the furnace, and to vary the nature of the resistor material so as to obtain the desired resistivity. Referring again to the experiment with the two resistors of coarse and fine grains, it is seen that where these have been treated in every way the same, G1 has a resistivity of 0.27, G2 a resistivity of 0.19. If, then, one crucible is surrounded with coke, like that used in G2, and the other with coke like that used in G1, the heat developed per second in the latter will be about 37 per cent greater than in the former.

In the description of the crucible furnace given above attention has simply been directed to the general methods of construction, various details being passed over. For example, the question of the material of which the crucibles are made has not been considered. This, of course, will modify details of construction. Thus, if the ordinary graphite or "plumbago" crucible is used, it will be found that it is a tolerably good conductor of electricity, and the furnace will have to be modified accordingly, proper allowance being made for the current carried by the crucible walls. Where the highest temperatures are required the only refractory material that will stand the work is pure graphite. Small crucibles of this kind may be made from graphite electrodes as manufactured by the International Acheson Graphite Co. Here we have, unfortunately, a material that is an excellent conductor for its resistivity is only 0.0008, so that it short-circuits the current at the part of the resistor where it lies. In order to get the most satisfactory heating of crucibles of this sort it is necessary to use currents of great amperage, so that a heating effect is obtained by the passage of the current through the walls of the crucible.

Electrolytic Preparation of Tin Paste.

In the electrolytic deposition of metals it is nearly always desired to get a good, *i. e.*, adhesive and coherent coating. This is always a desideratum in electrolytic refining and electroplating; to avoid a spongy deposit is then one of the chief points the operators have to look out for. A spongy deposit may be due to various causes, but may be obtained nearly always by using too high a current density at the cathode. This results in an impoverishment of the electrolyte near the cathode, with respect to the ions of the metal to be deposited, so that some other metal, and also generally hydrogen, is simultaneously deposited. It is under such conditions that the metal deposit becomes spongy. It is self-evident that under such conditions the ampere-hour efficiency of the deposition of the metal must be relatively low, since part of the current is consumed in setting free hydrogen.

In special cases, where it is desired to get a metal in spongy form, the above rule may be made use of. Such a case is discussed by F. **Gelstharp** in a recent Faraday Society paper, which we give below. An abstract of the discussion, which followed the paper, may be found on another page of this issue under Faraday Society Meeting.

Tin paste is used for the production of metallic or silvered paper, having the appearance of tin or lead foil, and it is also used for ornamental purposes. The process for manufacturing tin paste, described below, was specially designed for a firm of paper makers.

The oldest method of making tin paste was that carried out by the natives of India. They melted block tin at a high temperature, and poured it into a wooden box, which was then shaken vigorously; the tin was thus converted into very fine shot and dust. The finer particles which they used for the paste were separated from the coarser granules by gravitation by means of water. It was afterwards dried and mixed with a weak solution of glue, and in this state it was ready for use. This they used as a paint, the article to be ornamented was coated with it, and after being dried it was burnished.

The chemical process for making tin paste, as generally carried out in England, is as follows: The commercial ingot is reduced to what is known as feather tin by melting it and then allowing it to fall some distance. The tin in this form is dissolved in strong hydrochloric acid, with the aid of heat, in copper vessels, and a concentrated solution of stannous chloride thus obtained. The liquid is diluted and the tin precipitated in the form of paste by introducing metallic zinc. The process is troublesome on account of fumes, and, moreover, does not give the whole of the precipitated tin in the form of paste, but partly as bright metallic feathers and crystals, which have to be worked over again; whereas in working the electrolytic process about to be described there are no fumes, and the paste is perfect in every respect. Further, the process is not so costly and can be worked continuously, the deposited paste being removed at intervals without stopping the current.

The electrolytic process is similar to the electro-refining of metals. Anodes of tin are dissolved in an electrolyte of dilute hydrochloric acid and the tin is deposited in the form of a sponge or paste on cathodes of the same metal. The spongy deposit is lighter than the electrolyte on account of the hydrogen it contains, and consequently floats on the surface of the solution after being detached from the face of the cathodes, so that it can be conveniently removed without breaking the current.

An apparatus capable of producing 20 cwt. (2000 pounds or 900 kg.) of tin paste per week, consists of a generating plant giving 1500 amperes at eight to ten volts. There are five vats, which are worked in series and are made of slate or pitch-pine wood 106 × 106 × 91.5 cm. deep (3 feet 6 inches × 3 feet 6 inches × 3 feet deep), each containing four anodes and five cathodes coupled in parallel.

The anode plates are about 2.5 cm. (1 inch) thick, roughly cast from commercial ingots of tin; the surface exposed to the electrolyte being 91.5 × 76 cm. (3 feet × 2 feet 6 inches). The cathode plates are of block tin or tinned iron, the active surface being 91.5 × 76 cm. (3 feet × 2 feet 6 inches).

The anodes are supported at the two top corners, and in each vat are all connected to one conductor. The cathodes are supported by and soldered to copper conductors, which are all connected to a main conductor. These main conductors are of circular copper bars 3.2 cm. (1¼ inches in diameter).

The electrolyte is a dilute solution of hydrochloric acid in 5° Tw., which must be free from arsenic. The current density is 270 amperes per square meter, or 25 amperes per square foot of cathode surface.

As the anodes are dissolved away they are removed before they crumble and fall to pieces, and what remains of them is melted up with fresh ingots and cast, to form new anodes.

The electrolyte is kept in circulation by means of a small turbine or plunger pump of lead or block tin driven by a small motor, the liquid is drawn from the bottom of vats and discharged into the top; this is found necessary, as otherwise the electrolyte at the bottom of the vats would soon become very rich in tin, and instead of tin paste at the lower parts of the cathodes there would be crystal or leaf tin deposited.

When the deposited tin sponge has grown to about an inch in thickness it is detached from the cathodes by means of a T-shaped scraper, which is pushed down the face of the cathode plates; the floating, spongy tin is removed from the surface of the vat by means of a perforated scoop to prevent short-circuiting of the current through the floating tin; the cathode plates are varnished or painted with a non-conducting material from 5 to 7.5 cm. (2 or 3 inches) below the surface line. The tin paste is allowed to drain, and is then thoroughly washed with water and dried to make it ready for use.

This may not seem to be an efficient way of utilizing the electric current, because the current efficiency is only about 50 per cent; but any attempt that I have made to produce tin in the form of paste or sponge at a higher efficiency has been futile; it appears to be necessary to have hydrogen deposited along with the tin, for when no hydrogen is deposited the tin comes down in a crystalline or bright leaf-like form.

Some Present Problems in Technical Chemistry.*

By Prof. W. H. Walker, Ph. D.

(Concluded from page 28.)

In an address on Chemical Problems of To-day, delivered by Victor Meyer in 1889, the author pointed out that, although the synthesis of starch from carbon dioxide and water was a result not to be expected in the near future, yet, he says, "we may reasonably hope that chemistry will teach us to make the fibre of wood the source of human food." While we do not consider that this is a problem of technical chemistry for the present, the possible use of cellulose as a raw material from which to make food, renders more acute a problem which is to-day clamoring for solution, namely, the preservation of our forests. The influence which the forests of a country have upon its civilization is a topic which has been much discussed of late. That there is an intimate relation between the woodland of a district and the regularity of its rainfall, the absence of floods and freshets and the general climatic conditions, there seems now to be little doubt. But the consumption of forest products continues to increase far out of proportion to the growth of new timber. The substitution of other raw material in chemical industries which now use wood for this purpose becomes, therefore, an economic problem for the solution of which the chemist is held responsible.

The production of cellulose from raw materals other than wood is the first important factor in the chemical side of the question. The weight of wood consumed for the production of chemical fibre for the year 1902 was something over 2 million tons, while 1½ million tons were used for the manufacture of ground wood pulp. While from some points of view our American forests are sufficient to supply the demand for many years to come, it does not excuse us for the terrible waste of cellulose in forms other than wood, which we are constantly suffering.

On our flax fields of the West we are annually burning thousands of tons of flax straw which contains a large percentage of cellulose in a most valuable form. Considerable work has already been done on the utilization of this straw in the production of fibre, and some success has met the efforts of the By-Products Paper Co., now located at Niagara Falls. There is, however, still much room for improvements. In the straw of our wheat and oat crops, which is to-day largely destroyed on the fields, we have another source of cellulose of which we avail ourselves but little. In Europe the production of straw fibre is carried on to some extent, but is capable of great extension should sufficient economy in the process for treating it be introduced. The high content of silica has ever been a source of loss, owing to the fact that the formation of sodium silicate prevents the recovery of the soda now used in the digestion of the straw.

By far the greatest loss of valuable cellulose, however, is found in waste cornstalks and in bagasse or the sugar-cane after the soluble portions have been removed. There is a close analogy between these two products, in that there is associated with the woody portion carrying the cellulose a large amount of non-usable pith. Rapid progress has been made in the utilization of both of these raw materials within the last few years, and the indications are that before long they will prove a source of value rather than a nuisance, as is frequently the case at present. The market price of bleached cellulose fibre is to-day from 2½ to 3½ cents per pound. Starch may be bought for from 2½ to 4 cents, according to its source. It is seen, therefore, that there is little manufacturing margin in the conversion of cellulose to starch or sugar until the cost of the former has been considerably reduced. This can come about only through new processes designed to operate more economically than those at present in use, and to use as raw products the cellulose at present wasted on the fields.

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* An address delivered before the International Congress of Arts and Sciences at St. Louis

It would seem that a more economical step toward the production of food from wood might be through its ligneous or non-cellulose constituents. For every ton of cellulose produced there must be used two tons of wood; that is, an equal weight is wasted. In the soda process, as now conducted, these non-cellulose materials are burned to recover the soda which is held in combination with them. In the sulphite process this enormous amount of material, aggregating for America alone in a single year almost one million tons, finds its way into the water courses and ultimately to the ocean. This organic matter is most complex in its composition, but consists largely of one class of substances closely allied to the sugars, and another class having the general characteristics of tannins. That these sugar-like substances could be made to yield a food material is, from their nature, quite possible; so far as we know, however, but little has been accomplished in this direction. A number of uses have from time to time been proposed for this waste, but as yet none have been of practical value. Among the more promising may be mentioned a preparation to be used in tanning leather, a sizing material for paper and a substitute for dextrine in calico printing, and as an adhesive.

In addition to our annual supply of 4,000,000 tons of paper stock, we depend upon the forests for our supply of acetic acid, methyl alcohol and acetone. In countries where there is not the exorbitant tax upon fermented mash that exists in the United States, there would seem to be an opening for a process for the production of acetic acid from alcohol in a more concentrated form than can be produced through the aid of mycoderma aceti. It would, it is true, in the end depend upon the supply of fermentative material; but there are being wasted every year in the semi-tropical countries many thousand tons of crude molasses that could thus serve an economic end. For many uses acetic acid may be displaced by formic acid, a compound which admits of synthesis from carbon and water. The farther this substitution is carried the more acetic acid will be available for the manufacture of acetone and other compounds where the acetyl group is a necessity.

Concurrent with the disappearing forests is the increasing scarcity of vegetable tanning material. Hemlock and oak bark, sumac and chestnut wood are still the most important sources of tannins, although quebracho from South America and canaigre from Mexico and Texas are daily playing a more imporant part. The introduction of chrome tannage for upper leathers had a marked influence upon this industry, inasmuch as it furnished a cheap substitute for those finer tanning materials which are constantly increasing in price. A mineral tannage for heavy hides, along the lines so successfully followed for upper leather has, however, not been developed; the product lacks the rigidity and firmness combined with the flexibility which is characteristic of oak or hemlock tanned leather. There must exist methods for supplying to the hide materials having an action analogous to these vegetable tannins; it remains but to seek them out in order that a new and profitable industry may be established.

It is thus seen that technical chemistry can do much for the conservation of our forests; along many lines the time for action has already come.

When the consumption of a given article is in excess of its supply, the market price must rise. In accordance with this law we have seen the price of crude India rubber more than double in the last few years. The consumer of the finished article must pay this advance or accept an inferior grade of goods. Generally he does both.

The tropical forests of Africa and South America still contain untold quantities of India rubber; but so does sea water contain gold. For manufacturing purposes both might as well not exist. The only human beings that can live under the conditions obtaining in these tropical jungles are the natives; but the distance to which the natives can transport the rubber is comparatively limited. Although rubber-bearing trees are now being cultivated in the more easily inhabitable portion of the tropics, it will be a long time before the source of supply is an important factor in the market. And thus it comes that the synthesis of India Rubber presents to-day from at least the technical side, one of the most promising problems in chemistry.

The investigation of India rubber is greatly handicapped by the fact that it exists only in the colloidal state. The difficulties are, perhaps, more largely physical than chemical; that is, it is the molecular aggregation rather than the atomic structure of the individual molecule which presents such almost insurmountable difficulties. There are no clearly defined melting points, boiling points, tendencies to crystallize or any of those means of separating mixtures or characterizing individuals which aid in the investigation of most organic compounds. The researches of Weber and Harris, resulting in the establishment of the much-needed methods of analyses, have been of incalculable advantage to all those working with either the raw or the manufactured article. In many directions also, the paths along which important results are to be obtained have already been blazed by these investigators. Probably no other field presents such difficulties of manipulation, in addition to such profound problems of organic chemistry as does the investigation of India rubber; but on the other hand, few such unlimited opportunities for valuable work are offered in the field of chemical research.

Under the general head of utilization of trade wastes may be considered a large number of technical problems, the solution of which would not only add wonderfully to the economic resources of the country, but would aid in the solution of that much vexed question, river pollution. We have already mentioned the soda and sulphite liquor resulting from the manufacture of cellulose fibre from wood. Of almost equal importance is the waste yeast which is daily produced in the brewing of beer and ale. An extract of this yeast has a food value, as shown by analysis, equal to the best meat extracts. As the quantity of yeast allowed to go to waste is from one to two pounds for every barrel of beer brewed, we can form estimates of the great amount of this material at hand. Arsenic sulphide from the purification of crude acids, grease from the washing of wool, the utilization of city garbage and many other problems of this order are everywhere in evidence. It is not within the compass of this discussion to mention these almost innumerable sources of manufacturing waste which exist in the chemical industry; but keen competition on the one hand, and the State Boards of Health on the other, are constant stimuli to increased effort toward their utilization.

Although I have endeavored to select the above examples of unsolved problems with a view to touching upon as large a portion of the field of Technical Chemistry as possible, I could doubtless, with equal propriety have selected others. We can simply mention such important questions as the hygienic preservation of food, the flame-proofing and preservation of wood, prevention of the corrosion of structural iron and steel, the great problems of chemcal metallurgy, et cetera. We must, however, note some of the more recently developed forces and phenomena of nature, the application of which to technical chemistry forms problems for to-day. One of the most important of these is electricity. Thanks to the triumphs of modern electrical engineering we are now able to call to our aid unlimited amounts of this agent at a cost comparable to that of other forms of energy. Possibly the simplest, though not the earliest method of utilizing electrical energy in chemical processes is in supplying the heat necessary to carry on a reaction directly at the point where the reaction takes place. In a number of chemical industries, for example, the manufacture of phosphorus, it was previously necessary to produce within thick-walled retorts a very high temperature. The result was that a great deal of heat was wasted, the retorts deteriorated very rapidly and the reaction was carried on at a low efficiency. By using an electric furnace for the manufacture of phosphorus these expensive retorts are eliminated. In addition much cheaper raw materials may be used, the process is made continuous and a high efficiency obtained.

By the substitution of electrical heating for the closed retorts previously used in the preparation of carbon bisulphide the manufacture of this chemical has been placed upon an entirely new basis. The economy introduced by supplying the heat at the point where the union of carbon and sulphur takes place is clearly indicated by the low price at which this material can now be sold and its enormously increased consumption.

With the ability to obtain temperatures far above that which is possible by the ordinary combustion of fuel, there was opened up a new field in synthetic chemistry. Reactions which it was impossible to carry out on a technical scale, and others, the existence of which was not suspected, have now, through the application of electrical energy, become the bases of large manufacturing enterprises. Calcium carbide, carborundum, artificial graphite and many hitherto unknown alloys are the commercial products of the electric furnace where temperatures in the neighborhood of 3000° C. obtain.

The third and more strictly chemical application of electrical energy is in the use of the current for electrolysis. Faraday long ago determined the laws according to which chemical compounds break up when subjected to the passage of an electric current. It is only in recent years, however, that the cost of electrical energy has made it possible to apply the knowledge thus furnished by this great investigator. Among the many important advances due to this use of electricity may be mentioned the manufacture of caustic soda and bleaching powder by the electrolysis of brine. The percentage of the world's supply of these two standard articles, which is now made by this process is already a formidable figure, and constantly increasing. In the electrolytic production of aluminium we have seen an entirely new industry develop, until it is now one of magnificent proportions.

What the application of electricity will do for technical chemistry in the future can be predicted only by estimating the results of the past. In many fields it is practically virgin soil over which only the pioneers have trod, and which is still waiting to be tilled.

Under the name of catalysis or contact action is included the other force that we can mention this afternoon, the usefulness of which the technical chemist is only beginning to appreciate. These substances, which are capable of so wonderfully increasing or decreasing the speed of a reaction without themselves appearing in its final products vary in their nature from such simple ones as metallic platinum or ferric oxide to the most delicately constituted ferments or enzymes. The manufacture of concentrated sulphuric acid by such a process is, perhaps, the most striking example of the application of this idea, although, to be sure, the finely divided platinum used at present plays but the role which the oxides of nitrogen have done so successfully in the past. The reproduction of photographic negatives by substituting for the action of light on sensitized paper the contact action of certain chemical compounds, is a process worthy of its distinguished discoverer, Professor Ostwald. For this application of the catalysis idea even the most pessimistic must prophesy a great future. Still another phase of this question is found in the hydrolysis of fat by the enzyme found in the seeds of the castor-oil plant. Instead of the application of acid, heat and pressure the same result is obtained at room temperature by the quiet action of this catalytic body. The advantages to be reaped by the development of these phenomena can scarcely be foreseen. Even the wildest dreamer might easily do justice to the possibilities of this wonderful agent when intelligently used by the technical chemist.

We probably would not invite criticism were we to state that wherever we find a manufacturing establishment based upon chemical processes, there also exist problems in technical chemistry. That one factor which is so apparent that it scarcely needs mentioning, namely, the increase in the yield of processes now in operation, is enough to substantiate this assertion. The paramount question before us is, therefore, how can these problems best be solved. In any answer to this question there are two factors, both of which deeply affect the future growth of chemical industry. The first is the attitude of the manufacturer towards science and scientific work; the second is the training of the coming chemist. When, a few years ago, England awakened to the fact that many industries in which she was the pioneer and at one time the leader were in the main passing to other countries, there went up a great cry for "technical education." The nature of the industrial stimulus which has borne such magnificent fruit in Germany was not understood. In the minds of many, a panacea for all their difficulties was to be found in the technical education of the working classes. But this is unquestionably a mistake. Until there is a love of science for its own sake and an appreciation of the value of scientific method among the leaders of chemical industry, the fruits of technical education cannot be reaped. Carl Otto Weber, speaking of this move towards a more general scientific education in England, says: "Until the nation, as a whole, recognizes that the prosecution of scientific study as a mere means of money making is a profanation defeating its own end, the history of industrial developments in England will afford the same melancholy spectacle in this, as in the last century, technical education notwithstanding."

The time is past when a factory can be run by rule of thumb; when the chemist is looked down upon simply as a testing machine to be kept at a distance and generally mistrusted. It is true that there are many men to-day who pass under the name of chemists who are little more than testing machines; men who possess the ability to do nothing more than the most strictly routine analysis; but such men will never solve the technical problems of the present or any other time. I would not impugn the dignity or intrinsic value of analytical work—it is the corner-stone of all chemical investigation. But I would emphasize the fact, for it is a fact, that the manufacturer who employs a so-called chemist, one trained to "do" coppers or carbons or acids, and who at the same time expects this chemist to improve his process and keep his business in the skirmish line of the industrial battle, must eventually be numbered among the "not accounted for."

The second factor in this answer is the training of the coming chemist. What is the reply to that now so oft repeated question: What is the best preparaton for a technical chemist? I am personally of the opinion that it is not to be found in the teaching of applied chemistry, as this term is generally understood. This training must provide for something more than simply copying the present—doing as well as others do;

we must look to the future. We must provide men who are prepared to solve the unsolved problems. Within the last few years much has been said and written in America about the lack of adequate instruction in technical chemistry in our universities and colleges. It is assumed that American industries, based on chemical processes, do not flourish for lack of men trained in this branch of science. This, however, is not the case. It is not more instruction in applied chemistry that America needs, but rather a deeper and broader knowledge of pure chemistry, with a more extended training in original research.

In many of the problems we have already noticed, the solution depends upon the discovery of new compounds—the investigation and study of new reactions and relationships. This is the province of pure organic and inorganic chemistry. The foundations of these two departments cannot be too firmly or too broadly laid. The method of attack best followed in each cannot be too well understood. But it is not sufficient that we study only the initial and the final products. It is all important to learn the influence of the variable factors on the process; to study the reaction for itself. This is the province of physical chemistry, a department of science, the importance of which, to technical chemistry, cannot be overestimated. To be able to actually apply the laws of chemistry and to predict the course of reactions from general principles already proven is a tremendous economy of both time and energy.

After we have acquired the tools, however, we must learn to use them; after we possess a sound knowledge of inorganic, organic and physical chemistry, we must have adequate training in work requiring original and independent thought.

As I have already noted, the training to be derived from an investigation may be the same even though the incentive for its undertaking may be different. While I believe that so far as possible the student should be influenced to work for the love of knowledge and for the mastery of science for itself, yet especially in his later years of study there are advantages in allowing him to combine with this a utilitarian aim. In America, at least, most men enter our technical schools with the intention of fitting themselves as rapidly as possible for some useful calling in life. They have a feverish desire to get through and to enter the creative industries and accomplish something. They will work with enthusiasm upon whatever they can be made to recognize as contributing to this end, but by their very directness are intolerant of supposed digressions from their chosen path. The presence of too much of this spirit is to be regretted; but it is a power to be turned to service, not to be opposed. It does not follow that for a training in scientific method and for broadening the mental horizon a research which can have little, if any, practical value is superior to one, the solution of which can find immediate application. For advanced work, as much pure organic chemistry, for example, can be learned from an attempt to convert safrol into eugenol (a consummation in itself devoutly to be wished), as in the transformation of some other compound with a much longer name, but with no higher destiny than to fill a place in Beilstein.

So, also, in physical chemistry. A careful, painstaking investigation of some of our already established industrial processes with a view to determining the maximum yield at the minimum cost is of the greatest educational value. In other words, a problem for research may have a distinctly practical bearing without being any the less a study in pure science, or without having thereby an inferior educational value.

In other problems we have noted, the solution largely depends upon the process, not the reaction. This demands the chemical engineer, a man who combines a broad knowledge of general chemistry with the essentials of mechanical engineering. He must be well schooled in the economics of chemistry; have a knowledge of the strength and chemical resistance of materials; be able to design and operate the mechanical means for carrying out on a commercial scale the reactions discovered, and duplicating the conditions already determined.

With men whose foundations are thus broadly and deeply laid, anxious to enter the industrial arena; and with a generous appreciation of the scientific man on the part of the manufacturer, coupled with a willingness to grant him an adequate return on the money invested in such an education, the problems in technical chemistry of the present must rapidly become the achievements of the past.

Electrolytic Recovery of Tin.

The electrolytic detinning of tin plate (tinned sheet iron) has become an important industry during recent years. In Germany alone, about 35,000 tons of tin scrap are treated annually, at present, by electrolysis, from which 2 to 2.5 per cent, *i. e*,. about 800 tons of tin, are recoverd, amounting to about $500,000. In spite of this extent of the industry, little exact and authoritative information on the details of the processes is available, since the most successful companies in this field keep their methods carefully secret. In view of this situation, any information on experiments in this line requires attention. A recent Faraday Society paper of F. **Gelstharp**, printed below, is interesting in this respect. Some remarks, made in the discussion of this paper, are given on another page of this issue under Faraday Society Meeting.

An experiment was made by Mr. Gelstharp with an electrolyte of pure sodium hydrate in water, strength 10° Tw. (sp. gr. 1.05), temperature about 80° C., anode and cathode of tin plate, area 19½ sq. cm. (3 square inches), placed about 4 cm. (1½ inches) apart.

After switching on the current a small quantity of hydrogen gas was liberated at the cathode, and a little oxygen at the anode, gradually diminishing. The voltmeter stood at 3½ volts, and the ammeter at one-third ampere. The current was allowed to pass for about half an hour to see if the usual energetic electrolysis would commence; but no, the current continued to effect only a slight evolution of gas without stripping the tin from the anode. On examining the anode it was found to be very little attacked, but appeared to be covered with an extremely thin coat of a light brownish tint, which was not easily rubbed off. The current was again turned on without any different result.

The coloration was thought to be due to some low oxide of tin, which is a poor conductor of electricity. Therefore, the effect of reducing it by means of hydrogen was tried, this being done by reversing the current for a moment. Then, after reversing again the direction of the current, the tin-plate anode was completely stripped of its coating of tin in a very few minutes, hydrogen gas being energetically liberated at the cathode, which also became coated with loose, spongy tin. While this reaction was going on the ammeter rose to 1½ amperes, and the voltmeter fell to less than one volt, and when the cathode was agitated so as to knock against the sides of the containing vessel, the ammeter rose to two amperes, and the voltmeter fell to about half a volt.

Various pieces of tin plate were tried under the same conditions, and it was noticed that the clean, bright pieces acted in the same way as that just described; but pieces of old, oxidized, and tarnished tin plate were attacked at once, and quickly stripped.

It was found that instead of reversing the current to make the reaction start, it could be brought aobut by placing in contact with the anode in the electrolyte for a short time a piece of clean iron, most effective being a piece of tin plate which had just been stripped in the bath.

From the results of these experiments the writer conjectures, or suggests, that the clean, bright tin plate (differing from tarnished or oxidized tin) at first becomes thinly coated with a black or brown oxide which is insoluble in sodium hydrate, and that the effect of reversing the current, or making contact with iron, causes a deposit of hydrogen on the anode, which reduces this oxide to metal, thus roughening the surface, and making it in a favorable state for forming the soluble oxide, and when once the formation of soluble oxide has commenced, the electrolysis goes on uninterruptedly. I bring this reaction before the society in the hope that some member who may have investigated it will endeavor to give a satisfactory explanation of it.

After all, the tin was stripped from the anode, the ammeter again fell to about half an ampere, the voltmeter rising to 2½ volts; and when the current was continued the iron became coated with a reddish-colored deposit (probably composed of oxide of iron), which was not easily rubbed off. I noticed that the deposited tin, when squeezed and placed in water, generated hydrogen gas. This evidently points to the fact that spongy tin deposited in a sodium hydrate electrolyte is not only an alloy of tin and hydrogen, but also contains some sodium.

The agitation of the cathode in a special way seems to be an important point so far as the consumption of electric energy is concerned. The resistance due to polarization can be reduced to a minimum, not only by moving the electrode vigorously and by quick circulation of the electrolyte, but by giving it a quick succession of shocks produced by striking sharply. By this means the electric energy can be reduced by 30 to 40 per cent.

Comparing the acid and alkali processes for tin recovery, it is beyond doubt that the latter process is superior. In the alkali process iron apparatus may be used, and mechanical appliances for continuous treatment of tin-cuttings can be conveniently adapted; moreover, the electrolyte is continually being regenerated. In the acid process no suitable metallic apparatus can be used; therefore, mechanical appliances are hardly admissible, and for the electrical treatment of tin-cuttings to be a commercial success they must be handled very cheaply. This can only be done by mechanical means with a continuous process.

For the manufacture of tin paste the acid is superior to the alkali process, as the deposited tin from the former is very light and amorphous, whilst that from the latter is much denser, and consists of minute crystals, which are not so suitable for the uses to which this material is put.

Chloro-Electrolytic Smelting.

We have repeatedly referred in our columns (especially Vol. I., page 412, and Vol. II., page 404) to the Swinburne-Ashcroft process of treating complex sulphide ores. The process consists essentially of three steps. First, treatment of the complex sulphide ores at a temperature between 600° and 700°, with chlorine, with a resulting change of the mixed sulphides into chlorides, sulphur being set free; this step, which is purely chemical, is carried out in what is called the transformer. Second, substitution of one metal for another in the mixed chlorides, so that there is finally only zinc chloride left; this step, which is also purely chemical, is to be sub-divided into as many partial substitution processes as there are metals besides zinc present. Third, electrolysis of zinc chloride, yielding zinc and chlorine. The chlorine is used again in the transformer for treating new quantities of sulphide ore. The diagram given in our Vol. I., page 412, shows clearly that if the process is carried out with theoretical perfection, according to the above scheme, all the metals and the sulphur contained in the original sulphide ore are recovered separately, while the process is cyclic with respect to chlorine, that is, the chlorine is used over and over again.

The first two steps only of the process are applied by the (British) Castner-Kellner Co. (our Vol. II., page 404). The reason for this departure from the original scheme is that this company has a supply of electrolytic chlorine on hand, and wants to use it for the production of zinc chloride. The starting materials in their operation are therefore complex sulphide ore and electrolytic chlorine, the end products zinc chloride, sulphur, and all the metals—except zinc—which were contained in the sulphide ore.

What appears to be the most complete treatment of the whole process, is contained in a paper, recently presented before the Chemical Society in London by Prof. W. Morley Cobeldick, of the Royal College of Science. This paper discusses fully the application of the Swinburne-Ashcroft process to the treatment of Broken Hill ore, which may be described as a mixture of PbS, ZnS, Ag_2S, FeS_2, MnS and gangue. It is ground to a coarse powder, dried and transferred to the transformer, where it is decomposed by the chlorine; this results in a fused mixture of:

$PbCl_2$, $ZnCl_2$, $AgCl$, $MnCl_2$ and $FeCl_2$,

with the gangue held in suspension. The sulphur vapor is drawn out of the transformer and condensed. This is the first step.

In the second step, silver, lead, iron and manganese are consecutively removed by substitution, as follows. When the transformer is full of chlorides, they are placed in the "desilverizer" where the silver is removed by agitating the fused chlorides with molten lead, the following reaction taking place:

$$2AgCl + Pb = 2Ag + PbCl_2.$$

The desilverized chlorides are then transferred to the "deleader" for stirring them with sufficient molten zinc to cause the reaction:

$$PbCl_2 + Zn = ZnCl_2 + Pb.$$

The resulting chlorides contain only:

$ZnCl_2$ $FeCl_2$. $MnCl_2$, and gangue.

In order to separate the gangue matter it is necessary to dissolve the chlorides in water, and advantage is taken of this to precipitate the iron and manganese as hydrated oxides. To effect this replacement there are several ways of proceeding, but the following is the simplest: A current of chlorine is sent through the liquid which oxidizes the ferrous and manganous chloride to ferric and manganic chlorides. Crude zinc oxide is stirred in, and the hydrated oxides are precipitated:

$$Fe_2Cl_6 + 3ZnO + 3H_2O = Fe_2(HO)_6 + 3ZnCl_2$$
$$MnCl_4 + 2ZnO = MnO_2 + 2ZnCl_2.$$

This result may be brought about by using other materials, but in all cases zinc must be added in some form in sufficient quantity to combine with the chlorine of the iron and manganese.

The liquid now contains:

In suspension: All the iron, manganese and gangue originally in the ore.

In solutions: Pure zinc chloride containing all the chlorine used to decompose the original sulphide, combined with the zinc equivalent to all the metals in the sulphide.

It is filtered or allowed to settle and the clear zinc chloride solution evaporated till the chloride fuses, when it is electrolyzed, chlorine and zinc being the products.

Transformer.

The "Transformer," which is illustrated in Fig. 1, consists essentially of a sheet-iron vessel lined inside with firebrick, or other heat and chlorine resisting material. The chlorine is injected at the bottom, the crushed ore is fed in at the top, and sulphur vapor passes out from a pipe in the top and is conducted to condensers. Tapping spouts are provided for running out the fused chloride after the completion of the charge; they are so arranged as to leave behind sufficient molten chlorides for starting the next charge of ore.

Very little is known as to the reactions which take place in the transformer. Sulphur chloride, and ferric chloride are possibly formed, but at the temperature of the transformer it immediately breaks down into sulphur, which is given off in the free state, and chlorine which combines with the metals.

Ferric chloride, if formed, would immediately be reduced to ferrous in the presence of the sulphide ore. The fact that it is possible to obtain chlorides of the metals and sulphur by treating sulphide with S_2Cl_2 or Fe_2Cl_6 at 600° to 700° indicates that some such action as the above is possible.

The management of the transformer is exceedingly simple and can be mastered by an intelligent laborer in eight hours. Thus, if the temperature falls below the prescribed limit, sulphur chloride begins to form, and in coming over with the sulphur vapor alters the color of it. Its presence is also indicated by its characteristic odor, and consequently the man has to increase the speed of working by adding more ore and turning on more chlorine. If the temperature rises above the limit, the sulphur vapor is colored white, owing to the zinc chloride commencing to distil over. This is a sign to decrease the speed of working. When the whole of the sulphide is decomposed, the fact is proclaimed by the production of chocolate-colored vapor of ferric chloride, and the sulphur vapor ceases to come off.

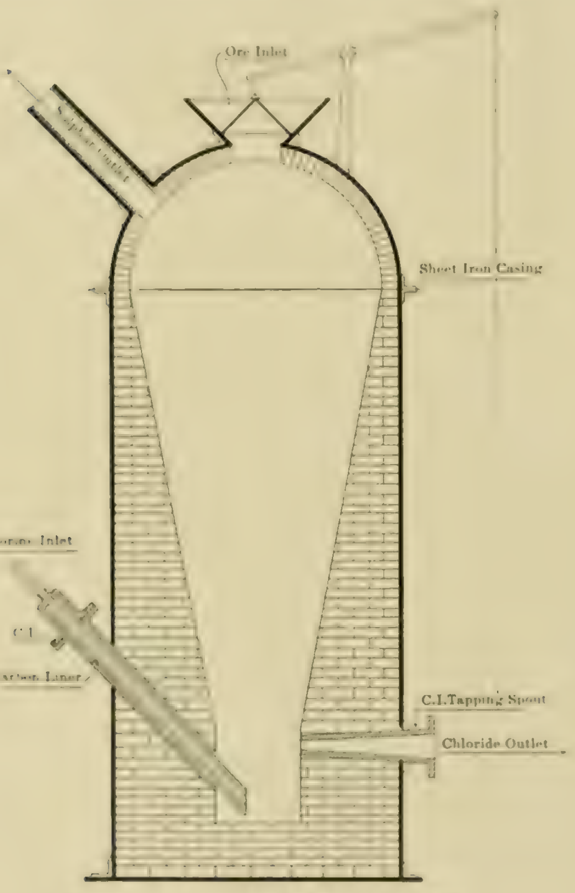


FIG. 1.—CONSTRUCTION OF TRANSFORMER.

The transformer reaction is strongly exothermic, so that no external heat is required to continue the reaction, the temperature being governed simply by the rate at which chlorine is passed through the fused mixture. It seems impossible to pass the chlorine so quickly as to obtain it passing out with the sulphur vapor. Immediate absorption takes place and the only effect of passing the chlorine rapidly is to raise the temperature. It is possible to work so rapidly that the chlorides are raised to their distilling temperature—in fact, with ores very rich in sulphide the rate of working depends on the rate of cooling of the walls of the transformer. Galena is a notable example of this kind of ore.

The efficiency of the transformer operation when properly carried out, amounts to about 98 per cent. If the chlorine gas supplied to the transformer is perfectly pure, all the sulphur is recovered, but this never happens in practice, since the chlorine invariably contains a small quantity of air which causes the formation of a little sulphur dioxide.

The fused chlorides are very fluid and can be easily handled, iron ladles being used to carry them about. The fluidity depends on the amount of gangue present, and it is necessary to keep the proportion within limits. The maximum percentage of gangue in a chloride which is workable is 30 per cent.

CONDENSATION OF THE SULPHUR.

For condensing the sulphur two methods are employed. To make flowers of sulphur the vapor from the transformer is caused to pass through a brick chamber, which is provided with a partition up the center extending from the tap down to within a foot or so of the bottom. The vapor enters one side at the top, travels down to the bottom, and finds its way through the space at the bottom of the partition into the other side, where it rises to the top and escapes by an outlet connected to the works chimney (with a sulphur dioxide absorption tower, if necessary). The sulphur condenses on the walls and is cleaned out periodically.

To make roll sulphur, the vapor is led to a large cast-iron vessel made in two halves. The lower half is set in a brick furnace, and is fitted with tapping spouts. The upper half is exposed to the air and contains openings to allow the vapor to enter and the waste gases to escape. The sulphur condenses in the top part, and owing to a small fire kept in the furnace, it gradually melts and falls to the bottom, where it accumulates and is periodically removed.

THE DESILVERIZER.

The desilverizer consists of a cast-iron pot fitted with a cover, and from this is suspended a cast-iron agitator, which is worked by the gearing on top. The post is set in a brick furnace, in which sufficient fire is kept to maintain the chlorides and lead in the molten state. The silver alloys with the excess of lead in the pot, and this lead-silver bullion is used for desilverizing until it reaches a value of 1,000 ounces per ton. The extraction of silver is quite complete, the operation usually lasting about one hour and a half.

THE DELEADER.

The desilverized charge runs direct to the deleader, which consists of an agitator pot similar to the desilverizer. It is arranged in a brick furnace in such a manner as to have very close control over the temperature. A gas fire is to be preferred on account of the greater ease with which the temperature can be controlled. The reason for this restriction is that, unless the chloride is kept at practically a constant temperature, while the zinc is being stirred into it, there is a great risk of the ferrous chloride being reduced to metallic iron, which forms a triple alloy of iron, lead and zinc, very hard and infusible at even a bright red heat.

The deleading reaction is exothermic, and on starting the charge, the heat given out is sufficient to maintain it in the molten state, but as the proportion of lead decreases, the temperature would fall, unless maintained by externally applied heat. The fire has, then, got to be very small at first, gradually being increased as the lead percentage falls. The operation of deleading takes about two hours.

DISSOLVING THE CHLORIDES.

From the deleader the molten chloride is run into water contained in a wooden tank fitted with an agitator. The zinc, iron and manganese chlorides dissolve and the gangue remains insoluble. The temperature is raised to about 70° C. by the introduction of the fused chloride. Chlorine is blown into the liquid, which is kept mixed by revolving the agitator.

The zinc oxide is obtained by using a cheap form of zinc oxide ore, which is ground and introduced gradually into the liquid.

There are many other ways of precipitating the iron and manganese—zinc oxide and potassium chlorate; bleach powder; lime and chlorine being a few of the oxidizing and precipitating agents which may be employed. In the case of lime, it is necessary to replace the calcium chloride, which is formed by zinc. This is done by adding sufficient zinc sulphate to precipitate the calcium as sulphate, zinc chloride going into solution.

When the whole of the iron and manganese is precipitated the liquor is pumped either into a settling tank or through a filter press. It is found that the best results are obtained when the zinc chloride solution is about 1.48 Sp. G. (=45 per cent $ZnCl_2$). This is the strength used in practice. The solution of zinc chloride is run from the filter press, or settling tank, into the evaporator. The gangue and oxide is washed free from zinc chloride, and the washing water used for dissolving the next batch of deleaded chloride.

An alternative method of proceeding, which omits the risk of iron being precipitated with the lead, is to dissolve the desilverized chloride in water. The iron and manganese is precipitated as hydrated oxide and the liquid allowed to cool It is then passed through a filter press, and the purified zinc chloride forms the filtrate, which is evaporated and fused.

The lead chloride, on account of its sparing solubility in the cold solution, is practically all retained in the filter cakes, and is extracted by leaching these in the press with boiling concentrated brine. The lead chloride readily dissolves in this at 100° C., but separates out again on cooling. To effect this the brine is run from the press to cooling tanks, and the lead chloride separates out, while the brine is used again for leaching another batch of cakes. The lead chloride is dried in a centrifugal separator, melted and deleaded with metallic zinc. Anhydrous zinc chloride is formed, which is added to the main bulk of zinc chloride concentrated from the press filtrate, and the whole eletcrolyzed.

EVAPORATOR.

This consists of a long iron tank fitted with cross partitions and a central partition running the entire length, thus dividing the tank into several compartments. The inside is lined with thick sheet lead.

The tank is set in a producer heated furnace, and is so arranged as to allow the lower pans to be made much hotter than the upper ones, a gradual increase of temperature being produced as the solution travels from the top to the bottom tanks.

The solution enters the topmost divisions in the tank and from them overflows into the next lower down, and proceeds thus until it reaches the lowest tanks. It is gradually concentrated as it travels down and matters are so arranged that it is a frothing mass containing about 90 per cent $ZnCl_2$, when it is in the two deep pans at the end. From these pans it flows into enamelled iron pans more strongly heated, which drive off the last portions of water. The chloride froths considerably at this stage, owing to small quantities being decomposed by the high temperature in the presence of water—HCl and ZnO being formed. This may be reduced to a certain extent by adding strong HCl during the frothing stage.

With careful working the amount of basic material formed is not great. It has been proposed to avoid this decomposition by performing the last stages of evaporation in a vacuum evaporator, but this has not yet been tried. The fused zinc chloride is colorless and appears quite clear through a depth of six inches. When basic it loses to a certain extent its transparency. If colored gray, with traces of organic matter it can be cleared by stirring in a little potassium nitrate.

PRELIMINARY ELECTROLYSIS.

In order to decompose slight traces of water and the basic compounds which are formed at the "frothing stage" the chlorides are subjected to a preliminary treatment by electrolysis.

This takes place in a brick-lined, sheet-iron vessel, fitted with a cover from which is suspended cheap carbon blocks which dip into the molten chloride, and form the anode. The cathode is composed of a bath of molten zinc. The chloride is kept molten by the passage of the current, no external heat being required. The preliminary electrolysis continues until all the water and basic compounds are decomposed. The chloride so treated is perfectly neutral and corresponds by analysis to the formula $ZnCl_2$.

The power required for this preliminary work is about 10 per cent of that required to decompose the purified $ZnCl_2$ into zinc and chlorine.

THE ELECTROLYSIS VATS.

The principle of the vats is shown in Fig. 2. These consist of sheet-iron cases lined with firebrick. The cover is made of cast iron and forms a gas-tight joint with the body of the vat. From the cover are suspended blocks of carbon, which reach nearly to the bottom of the inside of the vat body. An outlet for the chlorine is arranged from the vat cover, or sides.

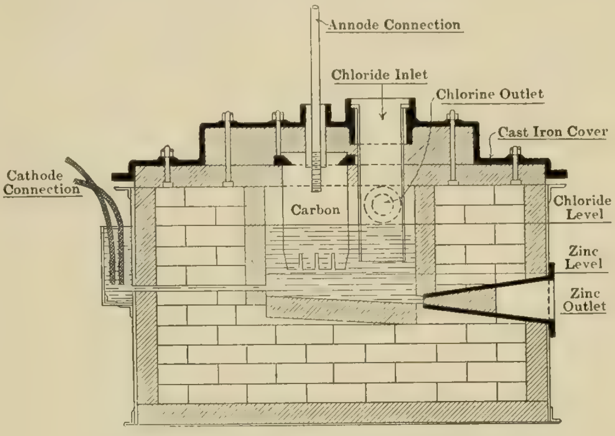


FIG. 2.—CONSTRUCTION OF VATS FOR ELECTROLYSIS OF MOLTEN ZINC CHLORIDE.

The cathode is composed of the molten zinc and connection is made from this to the external circuit by means of a small tank fastened to the outer case, and communicating with the interior by means of a small channel cut through the brickwork. The zinc fills this tank and the conductors dip into it, thus making a good connection, the zinc becoming solid. If the chlorides are pure, electrolysis proceeds quite smoothly, molten zinc being run out and fresh chloride run in at regular intervals.

With proper appliances for handling the chlorides, one man can look after about ten vats. The interior of the vat is maintained under a slight suction, so that in the case of a leaky joint air is drawn in and chlorine cannot escape to the outside.

A current density of about 400 amperes per square foot of cathode surface is employed, and the potential difference between the vat terminals is 4.5 volts. The temperature of a vat running normally is 450°. It is the custom to dilute the chloride of zinc with sodium chloride, the internal resistance of the vat and fuming being thus reduced. The chloride of zinc is diluted until the percentage of Zn is 28 per cent.

Up to the present vats of 3,000 amperes have been used, but designs of one to work at 10,000 amperes have been prepared, and it is expected that this will result in the lowering of the potential difference. It has been found possible, with perfectly pure $ZnCl_2$, to obtain a yield of zinc extremely close to the theoretical. In one experiment, where the current was measured by a copper voltameter, a yield of 1.193 grams of Zn per ampere hour was obtained as against a theoretical of 1.2.

PUMPING THE CHLORINE.

All that now remains to complete the cycle is to pump the chlorine gas from the vat, and put it under a pressure of 25 pounds per square inch for use again in the transformer. Easy as this may appear, it has caused a great deal of truoble and expense to accomplish. One difficulty was that all forms of asbestos or rubber packing were rapidly destroyed by the action of the chlorine, and a system of metallic packing had to be arranged.

Another trouble was caused by the fact that damp chlorine —even if moisture were present in extremely small quantities —caused the destruction of the iron work of the pump. The lubrication of the pump was another great difficulty, since almost all hydrocarbon oils were decomposed by the chlorine, resulting in the valves and other parts of the pump becoming coated with a hard deposit of carbon which stopped the efficient action. All these troubles have been overcome and chlorine has been pumped for weeks on end with no difficulty.

It has been proposed to pump the chlorine by means of a turbine blower, such as is sometimes used in blast furnace work, but this has not been tried.

To supply the waste of chlorine which occurs in the different stages—about 5 per cent per cycle—chlorine has to be supplied from external sources. This is usually obtained from bleach powder. Chlorine so obtained is wet, and in order to extract the moisture it is drawn through a series of lead towers packed with a solid zinc chloride and connected with the suction of the pump. The accumulated moisture gradually causes the zinc chloride to form a strong solution, which falls to the bottom of the tower and makes its exit into a tank from which it is periodically removed, concentrated, fused, and repacked in the towers. The drying of the gas is thus produced at the slight cost of fuel for evaporating and labor for packing.

CONCLUSION

This completes the description of the cycle as applied to a material having the composition of Broken Hill ore. If the ore to be treated is more complex, the number of processes which the chlorides pass through is increased.

Thus, if the ore contains copper in addition to the other metals, the treatment would be as follows: The chlorides from the transformer would be dissolved in water and the liquor stirred with copper plates, replacement of the silver by copper then taking place, the metallic silver being deposited on the plates.

Oxidation and precipitation of the iron and manganese and the separation of these oxides together with the lead chloride and gangue by settling or filter process would then proceed in the manner already described. The clear solution containing only copper and zinc chloride would then be stirred with granulated zinc in sufficient quantity to replace all the copper present, the latter metal after allowing the liquid to settle, being obtained in the form of cement copper, while the clear solution containing now only pure zinc chloride, would be evaporated and electrolyzed in the usual manner.

If gold were present it would be precipitated with the silver and recovered with that metal. Antimony, if present, would distill from the transformer with the sulphur, and condense before the substance in the form of the liquid pentachloride from which it can be replaced indirectly by means of zinc.

The author thinks that treatment of arsenical, antimonious, telluride and other ores for gold could, with modifications, be carried out, and thus render workable a good deal of material which at present cannot be treated.

Cheap power is essential for the economical working of a process like this, and therefore the works should be situated near to a cheap source—such as a waterfall; the works should also be arranged to allow the material to flow from one department to the next, in order to avoid handling. This could conveniently be done by either building the works on a hillside, or by placing them in a building which contains as many stories as there are operations, as in the case of Broken Hill ore. The crushed ore would be elevated to the top story, where the converter would be placed, and the fused chloride flow from this through the different stages, ending with the electrolytic vats on the lowest floor.

Faraday Society Meeting.

The tenth ordinary meeting of the Faraday Society was held on December 19, 1904, at the Institution of Electrical Engineers in London. Mr. J. Swinburne, vice-president, occupying the chair.

The chairman announced that the council had made the following nominations of officers and members of council to serve during the coming year. The election will take place at the annual general meeting, to be held in February:

President.—Lord Kelvin, O. M.

Vice-Presidents.—Prof. A. Crum Brown, F. R. S., Sir W. Huggins, P. R. S., Sir Oliver Lodge, F. R. S., Dr. Ludwig Mond, F. R. S., Lord Rayleigh, O. M., Alexander Siemens, Pres. I. E. E., James Swinburne.

Council.—G. T. Beilby, Bertram Blount, W. R. Cooper, Sherard Cowper-Coles, Prof. A. K. Huntington, Dr. R. A. Lehfeldt, W. Murray Morrison, Dr. W. S. Squire, Dr. O. J. Steinhart, Prof. E. Wilson.

Treasurer.—Dr. F. Mollwo Perkin.

THE ELECTRIC FURNACE.

M. ADOLPHE MINET communicated Part II. of his paper on "The Electric Furnace: its Origin, Transformations and Applications," which was read in abstract by Mr. R. S. Hutton.

In the first and introductory part of his paper, which was read before the Society on June 9 last, M. Minet showed that the historical development of the electric furnace falls into the three following periods:

1. Laboratory furnaces, 1808 to 1886.
2. Industrial furnaces, 1886 to 1890.
3. Development of industrial applications from 1890 to the present day.

The present instalment, after some preliminary general remarks on the e. m. f., current densities, and yields, with arc furnaces and resistance furnaces, respectively, proceeds to deal in detail with the evolution of the furnace during the first period. The subject is sub-divided as follows:

(*a*) *Original Furnaces,* namely, Davy's arc, Pepys's resistance furnace for the cementation of iron, and Davy's and Napier's cathode furnace.

(*b*) *Electrolysis at High Temperatures;* the production of aluminium and its alloys, and the alkali and alkaline earth metals.

(*c*) *Miscellaneous Furnaces,* of which the most important are the early Despretz and Pichon furnaces, Berthelot's electric "egg" for the synthesis of acetylene, Siemens' and Huntington's well-known steel and other furnaces, the Louis Clerc furnace, interesting as the progenitor of the famous Moissan type, and Borchers' resistance furnace.

The paper, which is illustrated with diagrams of the principal furnaces alluded to in the text, closes with a bibliography covering the period dealt with. A final instalment will discuss the furnaces of the second and third periods.

Mr. W. B. Clarke referred to the importance and difficulty of making temperature determinations of electric furnace processes, and expressed the hope that M. Minet would deal with that question in Part III. of his paper.

Mr. L. Gasta handed round some "siloxicon" and descrbed its properties.

Mr. E. Kilburn Scott described some of his experiences in the early days of the manufacture of calcium carbide.

The chairman described a novel type of furnace that he had once hit upon by accident, and which he thought might turn out to be useful. He was sending a current through fused barium chloride, and the temperature having been allowed to get too high the cast-iron crucible melted and formed a fused mass that was kept together by an outer skin of solidified chloride. This suggested the use of a neutral substance like barium chloride for heating to a very high temperature alloys or metals which it was desired not to contaminate with impurities.

Electrolytic Analysis of Cobalt and Nickel.

A paper on this subject, by Dr. F. Mollwo Perkin and Mr. W. C. Prebble, was then read by Dr. Perkin.

Cobalt.—The aim of the experiments was to obtain bright deposits of the metal that should be quantitatively accurate. Various reducing agents were first of all employed, but only with sodium hypophosphite were brilliant deposits obtained, and then phosphorus was found to be present. Salts of the various organic acids were next tried, but only ammonium oxalate and tartrate gave satisfactory results, the latter having the better appearance. Traces of carbon, however, were deposited with the metal. Ammoniacal solutions of ammonium borate were also tried. The most satisfactory results were obtained with a solution containing an alkali phosphate and a little phosphoric acid; not only is the deposit bright, but the analytical numbers are also good. By this method the whole of the metal can be thrown out in from 3 to 3½ hours.

In carrying out this process it was found that only a small quantity of ammonium or sodium phosphate may be added; otherwise, the double salt of cobalt and the alkali phosphate is precipitated, and when once precipitated it can only be redissolved again with great difficulty. The following method of procedure gave the most satisfactory results: About 1 gram of the cobalt salt was dissolved in about 50 c.c. of distilled water, and then 5 c. c. of a 5 per cent solution of phosphoric acid added. To this solution 25 c. c. of a 10 per cent solution of ammonium dihydrogen phosphate [$(NH_4)H_2PO_4$], or better, sodium dihydrogen phosphate (NaH_2PO_4) was added; the solution was then made up to 130 c. c. and electrolyzed, a gauze flag electrode (Electrochemical Industry, Vol. I., page 414) being employed as cathode. After the electrolysis has proceeded for half an hour or so there is usually a brown deposit of cobalt oxide on the anode. In order to dissolve this deposit off, about 0.5 gram of hydroxylamine sulphate or chloride is added to the solution, when the deposit almost immediately vanishes. Very often 0.1 gram of the hydroxylamine salt is quite sufficient. The addition of a few c.c. of formaldehyde solution has the same effect, but the action is much slower. The anode deposit is much more marked when hot solutions are electrolyzed, but there is no difficulty in removing it, as already described. If the solution is heated to too high a temperature, there is a tendency for the double salt of sodium and cobalt phosphate to be precipitated out. The phosphoric acid is added in the first place to prevent the precipitation of this double salt, because if it is once precipitated a considerable quantity of phosphoric acid must be added in order to dissolve it again. But when there is too much phosphoric acid in the solution the deposition of the cobalt is extremely slow. It is, in fact, always an advantage, after the bulk of the cobalt has been deposited, to add a few drops of very dilute ammonium hydrate to neutralize the excess of acid liberated by the electrolysis.

Nickel.—Similar solutions were tried for nickel deposition. In this case splendid results were obtained with the borate solution, while the phosphate solution, which gave such good figures in the case of cobalt, was not at all satisfactory. Using solutions containing ammonium borate and ammonia for nickel, the analytical results, from a quantitative point of view, are magnificent, and although the deposited metal has not the brilliant burnished appearance often obtained by other methods, it is not unsightly, having a matte, silver-greyish appearance.

The authors have not yet succeeded in devising an accurate electrolytic method for the separation of nickel of cobalt, and they find that the methods given in books are unreliable.

In the discussion which followed, Dr. O. J. Steinhart discussed the problem of the separation of the metals. The present method is based on the fact that cobalt is more easily oxidized by chlorine than is nickel. He also referred to the possible industrial bearing of the work of the authors in connection with nickel extraction.

Dr. L. Dobbin, in a written communication, drew attention to the work of Marshall on the same subject. In the case of cobalt he merely added a little nitrate to the ordinary double sulphate solution.

Dr. Perkin said that the presence of nitrate rendered deposition very slow.

Tin.

Mr. F. Gelstharp presented two short papers on the electrolytic preparation of tin paste and a note on the electrolytic recovery of tin, respectively, which were read by the secretary. These two papers are printed in full on other pages of our present issue.

In the discusion which followed, Mr. W. C. Prebble thought an endless band would be a less crude method of removing the paste than that given by the author. Regarding the presence of arsenic, which was hurtful here, it was curious that in the case of zinc this actually causes a spongy deposit to form.

Dr. O. J. Steinhart gave a few particulars of an electrolytic detinning process that was now successfully treating in London seven or eight thousand tons of scrap per annum. A great difficulty found in practice was to dry the tin sponge without its firing.

Dr. T. C. Cain did not think it probable that sodium was deposited with the tin.

Mr. H. Sayer asked what percentage of the tin was recovered. An objection to an alkali process was that it would not remove the tin from the alloy at the surface junction of the two metals.

Mr. A. C. Bawtree sent in a written communication describing an electrolytic method for making tin powder that he had worked out some years ago. His solution contained 0.5 per cent $SnCl_2$ and 0.33 per cent $SnCl_4$, and was worked cold with an e. m .f. of 2.32 volts. This solution successfully overcame the difficulty of crystalline deposits.

Mr. H. S. Coleman also communicated some of his own experiences in tin deposition. He had worked under conditions similar to those described by Mr. Gelstharp. He found, however, that if the bath were worked between 50° and 60° C., no diffculty was experienced in starting the current, and the scrap was stripped clean in twenty minutes.

Improvements in the Treatment of Slimes for Winning Gold.

The interesting article by Messrs. Clement Dixon and M. Torrente, published in our June issue (Vol. II., page 215), contained a concise review how the problem of treating slimes for winning the gold contained in them came up, and was solved in the South African gold fields. This article gave a full review of the development and present practice of gold metallurgy on the Rand.

In a recent paper presented before the Chemical, Metallurgical and Mining Society of South Africa, and published in the August number of the journal of the Society, Mr. M. Torrente, who, as the successor of Messrs. Charles Butters and von Gernet, has been most intimately connected with the developmnet of the electrolytic precipitation process on the Rand, brings forward what he considers a very decided improvement in the treatment of slimes. In the following we reproduce this paper practically in full.

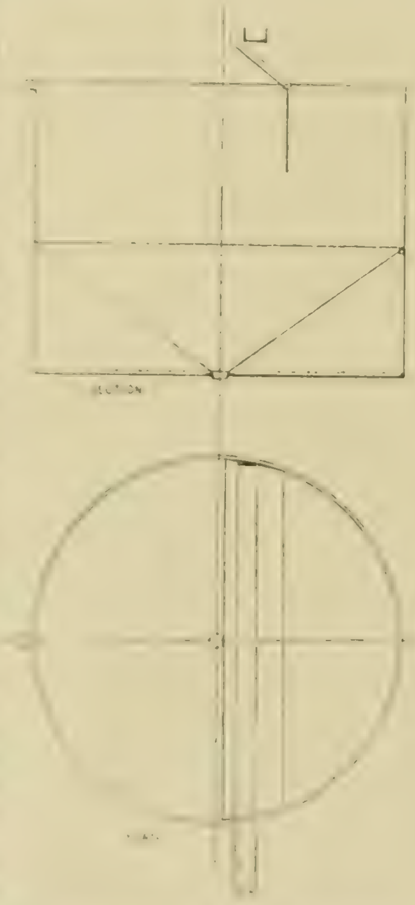
Although the paper only refers to the settlement and separation of the slimes from their solutions, the fact that a good continuous settlement can be obtained, and perfectly these solutions continuously withdrawn (without in any way affecting the value of the residues at present obtained) may, in itself, be considered a step further in the right direction. In addition, some other very considerable advantages may be derived by following this procedure.

In his experience of slimes treatment, Mr. Torrente has arrived at the conclusion that the problem of a continuous treatment presents the greatest difficulties. These may not be so considerable in treating the fresh slimes as they come from the battery, but when dealing with old accumulations where the dissolving of the gold may take anything from twelve to twenty-four hours, the obstacles in the way are evident to anybody experienced in these processes. The first man who experimented on these lines was Mr. Ernest T. Rand, formerly manager of the Simmer and Jack cyanide works. To a certain extent his results were satisfactory, but he had to contend with the same disadvantage which other processes have shown since, viz, that a considerable amount of slimes went out of the plant untreated.

After some experimenting, Mr. Torrent gave up the idea of continuous treatment. The drawbacks of a decantation slimes plant are many, and the margin for improvements still very great. The greatest drawback, in the author's opinion, is the time a plant has to remain idle while decantation is going on. To be able to work continuous settlement on a plant means immediately doubling at least its capacity in one case, and halving its initial cost in the other. By his new method Mr. Torrente considers that he has fairly well attained these two objects.

In the first instance the author mentions that he does not interfere with the treatment of the slimes, which may be carried out as heretofore.

Having first ascertained that the gold is dissolved in the slimes charge, the charge is slowly pumped to the separator, which consists of a tank with a sharp, conical bottom and a vertical partition going down to about two-thirds of its depth or near the level of the inverted cone's base. This partition, which may or may not exceed the height of the liquid in the tank, is continued in the form of an inclined plane for some distance above the level of said liquid. The tank, before it starts work, is filled with water or solution up to the solution outlet, and the slimes pulp is delivered on the inclined plane, where it spreads over the whole of the surface in a very thin layer. The pulp thus enters the solution without the slightest splash, and is bound to continue its downward movement on the one side of the partition. The quantity of pulp, in proportion to the mass of solution contained in the settler, is so small that it becomes diluted to an extent which is practically impossible in any other way. The fine particles of slimes assert to great advantage their specific gravity and continue in their downward movement after they have arrived at the lower edge of the partition, accumulating in the cone of the vat, from the bottom of which they are discharged. The solution, once freed from the slimes it originally contained, having now less gravity, rises in the contrary direction to the column of pulp on the other side of the partition and is discharged through an outlet provided for this purpose near the top of the tank.



APPARATUS FOR SLIMES TREATMENT

The experiments so far made have given the greatest satisfaction. The following tables showing comparative results, will prove interesting. In every case, the pulp taken has come from the slimes plant after the gold has been dissolved, as far as the slimes in question have permitted, during a treatment of twenty-four hours. The washes have also been given as much as possible under working conditions. It must be well kept in mind that the slimes in question are old accumulated slimes, very difficult to treat. Therefore, most figures will be found very high compared with the actual work now done with fresh slimes taken direct from the batteries.

Experiments in Continuous Settlement.

The slimes used for this experiment were taken from the company's dam nearest to the spruit and can be considered as the worst possible for cyanide treatment, being highly contaminated with acid and organic matter, the time necessary for dissolving the gold they contain being eighteen to twenty-fours hours.

Quantity of dry slimes taken, 200 pounds.
Capacity of settler, 800 pounds.

First Wash.—Time taken for settlement, *i. e.*, getting the charge through, 8½ hours.

Ratio of slimes to solution	4:1
Original assay value of the slimes	4.5 dwts.
	Assays.
Solution added before treatment	13 grs.
Solution from pulp after treatment	1 dwt. 12 grs.
Value of undissolved gold in slimes	15 grs.
Value of the residue (59.3 per cent moisture), including gold carried in moisture—	
By calculation	2 dwts. 16 grs
By assay	1 dwt. 12 grs

Second Wash.—Time taken, 10½ hours.

Solution used for washing	1 dwt. 3 grs.
Pulp solution	1 dwt. 7 grs.
Value of undissolved gold in slimes	15 grs.
Value of the residue (56 per cent moisture), including gold carried in moisture—	
By calculation	2 dwts. 3 grs
By assay	1 dwt.

Third Wash.—Time taken, 12 hours.

	Assays.
Solution used for washing	4 grs.
Pulp solution	13 grs
Value of undissolved gold in the residue	12 grs.
Value of residue (49.2 per cent moisture), including gold carried in moisture—	
By calculation	21 grs.
By assay	1 dwt.

This experiment compares with the treatment of slimes as now carried out as follows:

SAME KIND OF SLIMES (AFTER GOLD HAS BEEN DISSOLVED).

First Wash.—Time taken for settlement, 12 hours.

Ratio of solution to slimes	4.1
Original assay value of slimes	4.5 dwts.
	Assays.
Value of solution before treatment	13 grs
Value of solution after treatment	1 dwt. 12 grs.
Value of undissolved gold in slimes	12 grs.

Solution decanted 1½ tons to 1 ton of slimes.

Value of residue, including the gold still carried in the remaining solution	4 dwts. 6 grs.

Second Wash.—Time taken for settlement, 12 hours.

Ratio of solution to slimes	4:1
Value of solution used for washing	4 grs.
Value of solution after pulping	1 dwt.

Solution decanted 1½ tons to 1 ton of slimes.

Value of residue, including the gold still carried in the remaining solution	3 dwts.

Third Wash.—Time taken for settlement, 60 hours.

Ratio of solution to slimes	4:1
Value of solution used for washing	4 grs.
Value of solution after pulping	13 grs.
Value of residue (discharged with 50 per cent moisture), including the gold carried in the moisture	1 dwt.

These results apply solely to charges like the above, which has been treated under the most favorable conditions.

Second Experiment.

The slimes used in this experiment were taken from the dam near the cyanide works. In the ordinary way they are most difficult to settle, although more amenable to cyanide treatment.

Quantity of dry slimes taken, 200 pounds.

First Wash.—Time taken to put the charge through, 24 hours.

Ratio of slimes to solution	1:8.4
Original assay value of the slimes	3.8 dwts.

	Assays.
Solution added before treatment	14 grs.
Solution from pulp after treatment	1 dwt.
Value of undissolved gold in slimes	12 grs.
Value of the residue (61.1 per cent moisture), including gold carried in moisture—	
By calculation	2 dwts. 1 gr.
By assay	1 dwt. 12 grs.

Second Wash.—Time taken, 12 hours.

	Assays.
Solution used for washing	10 grs.
Pulp solution	22 grs.
Value of undissolved gold in slimes	7 grs.
Value of the residue (58.5 per cent moisture), including gold carried in moisture—	
By calculation	1 dwt. 5 grs.
By assay	19 grs.

Third Wash.—Time taken, 11 hours.

Solution used for washing	4 grs.
Pulp solution	10 grs.
Value of undissolved gold in slimes	7 grs.
Value of the residue (55.9 per cent moisture), including gold carried in moisture—	
By calculation	18 grs.
By assay	15 grs.

The first wash was put through in twenty-four hours and with about double the quantity of solution for experimental purposes only. The time taken for the second wash applies to the first for purposes of calculation.

The second experiment compares with the treatment of the same class of slimes as now carried on as follows:

First Wash.—Time taken for settlement, 24 hours.

Ratio of solution to slimes	2:6.1
Original assay value of slimes	3.8 dwts.

	Assays.
Value of solution before treatment	23 grs.
Value of solution after treatment	2 dwts. 2 grs.
Value of undissolved gold in slimes	22 grs.

Solution decanted 1.6 tons to 1 ton of slimes.

Value of residue, including the gold still carried in the remaining solution	2 dwts. 21 grs.

Second Wash.—Time taken for settlement, 60 hours.

Ratio of solution to slimes, 3 tons to 1 of slimes.	
Value of solution used for washing	4 grs.
Value of solution after pulping	18 grs.

Solution decanted 2 tons to 1 of slimes.

Value of residues, including the gold still carried in the remaining solution	1 dwt. 4 grs.

Owing to their nature, the treatment of these slimes had to be curtailed to two washes instead of the former three, the time not being sufficient to allow three settlements.

By following the process above described a slimes plant of almost any size can now be built with two or three treatment vats and three separators. The experiments have clearly shown that slimes to the amount of two-fifths of the total capacity of any given tank can be separated in twenty-four hours, and the return of the pulp from one separator to another with the solutions as now employed constitutes a very thorough washing.

The procedure would be as follows: The treated pulp is pumped to the first separator; the overflowing solution, which constitutes the strong solution, has its gold precipitated in the zinc boxes. From the bottom of the first separator the slimes are pumped to the second separator after the addition of the necessary amount of intermediate solution to make them into a thin pulp. The same operation is repeated in the third separator, substituting precipitated solution or water, as may be most convenient for thinning the pulp. From the bottom of the third separator the slimes flow by gravity into the residue dam, or are pumped there, as may be found most convenient. As every separator constitutes a wash, the number of these will be determined by the value of the slimes and their nature. In some plants two may be found to be sufficient; in others it may be advantageous to put three or even four. So far the author has found that three have done very satisfactory work.

How far the capacity and its initial cost is affected is evident. The author thinks he may safely say that the capacity of any of the existing plants can be doubled at once at a very small cost. In new ventures a plant consisting of three 400-ton treatment vats and three 500-ton separators will easily handle 200 tons of dry slimes per day, that is to say, that for about one-half the present capital expenditure a plant can be erected to treat double the amount of slimes that a plant would treat if constructed on existing lines. The experiments were all carried out at the Crown Reef.

Notes on Electrochemistry and Electrometallurgy in Great Britain.

(*From Our Special Correspondent.*)

The Faraday Society Meeting.

The fact that four papers were down for reading at the December meeting of the Faraday Society, on December 19, led those attending to expect a protracted meeting. The attendance was rather smaller than usual, owing to the proximity of the holidays. Mr. Swinburne presided. The first paper consisted in the second instalment of M. Adolphe Minet's serial on "The Electric Furnace, Its Origin, Transformations and Applications." The first instalment was read on June 9, and a rather scathing communication sent to the *proceedings* by Messrs. Hutton and Morrison. It was therefore somewhat surprising to find Dr. Hutton generously acting as *locum tenens* for M. Minet. A brief summary of the paper was given, and reference was made to the work carried out by Prof. Richards, of Lehigh University as to what actually constituted the economic yield of a furnace. M. Minet's encomium on Borchers was mentioned as a useful confession which probably described Borchers' true relation to electrochemistry. The reference in question was worded as follows: "Like most others who have plenty of ideas, Borchers was attached secondary importance to some of his laboratory results. Then [illegible] other investigators made the conceptions of the German investigator the basis of their researches, without troubling to indicate the source. As a result, Borchers showed a tendency to restrain his appreciation of the work of others, whether they had worked on different lines or had forgotten to do him justice. But time, happily, allows all things to be seen in their right perspective."

The resulting discussion was strikingly irrelevant. One member commenced to read a small paper on Acheson's new product "siloxicon," another spoke of temperature determinations. Mr. Swinburne's remarks were practically confined to an anecdote concerned with a laboratory blunder when some barium chloride was being melted. The property of being intensely heated, sufficiently indeed to reduce iron without being oxidized, suggested that barium chloride might possess valuable properties in electro-thermic work. A vote of thanks was, of course, accorded to M. Minet, whose history of the electric furnace will probably run into a third or fourth paper. It would surely be best not to discuss any of these, but to have a sort of field day at the conclusion and discuss the contributions as a whole.

Dr. Perkin next introduced the paper by himself and Mr Prebble on the "Electrolytic Analysis of Cobalt and Nickel." The introduction was slightly disconnected, not following the same order in the treatment of the subject as the paper. Some rather interesting samples of deposited zinc and nickel were shown. The discussion was confined to the reading of a communication of Dr. Dobbin, and to some remarks by Dr Steinhart. In the former, reference was made to a paper by Marshall, read before the British Association in 1898, in which the addition of traces of nitrate was suggested. A peculiar rise of voltage on the exhaustion of the nickel from the solution was then recorded. Dr. Steinhart dealt chiefly with the commercial side of the question. The difficulty of separation was chiefly due to both metals forming peroxides. Dr. Perkins' success was probably due to the fact that the salts used had a reducing action, limiting the formation of peroxide. In reply to Dr. Dobbin, Dr. Perkin pointed out that the addition of traces of nitrate greatly increased the time needed in any such operation. He had not noticed the rise in voltage cited.

The next two papers by Mr. F. Gelstharp: the first a note on the "Electrolytic Preparation of Tin Paste," and the second a note on the "Electrolytic Recovery of Tin Scrap," were discussed together. In regard to the first, Mr. Prebble suggested the use of an endless band to carry the tin plate outside the cell, in place of the scraper used. The question of the presence of arsenic and difficulties of drying were cited. Dr. Steinhart then intervened, and for the rest of the evening championed the absent author of the paper. Tin recovery processes in England chiefly suffered by reason of the paucity of tin scrap. He endorsed the author's remarks on agitation. The chief difficulties were those of subsequent utilization; the oxidation was terribly bad and smelting difficult. Only a low voltage was needed, and the current efficiency was very high. Indeed, in a case brought under his notice, higher than that allowed for by Faraday's law. Dr. Perkin here intervened to suggest that extraordinary efficiencies were due to iron and copper impurities. No one countered Dr. Steinhart's remark that the current efficiency figures were those of several months by questioning the calibration of the meters. Mr. Swinburne said a few works of his own experiences as an investigator, which were chiefly concerned with the difficulty of getting tin scrap. Mr. Sayer then asked what was the actual percentage of tin recovered? Only from 2½ to 3 pounds of tin were used per box of plates (weighing 100 pounds); figures showing a recovery of 5 per cent to 9 per cent were surely inaccurate. Dr. Steinhart replied that the recovery efficiency was 95 per cent, and, replying to a further query (as to whether the tin left on the scrap was deleterious in iron and steel manufactures), remarked that the iron dust was briquetted and found a ready sale among smelters who insisted on accuracy within a second or third place of decimals in the contents. The meeting then adjourned.

The Germicidal Action of Copper.

Electrical engineers who have followed the recent experiments of the Washington Department of Agriculture as to the value of copper sulphate in the prevention of the growth of algae and certain pathogenic organisms present in water supplies, will, unless interested financially in copper refining, forsee without much enthusiasm another factor which may make for dearness in the future. A perusal of a recent paper by Dr. Rideal and Dr. Baines, read before the last Congress of the Sanitary Institute at Glasgow, can only tend to a strengthening of these forebodings. Dr. Rideal and his colleague show, firstly, that, comparing the action of copper sulphate and copper chloride on water inoculated with the bacillus coli and bacillus typhosus, the bactericidal efficiency is dependent on the absolute amount of ionic copper present, that copper foil having a surface of 100 square centimeters introduced into a 100 c. c. flask containing polluted water, effected a complete sterilization within twenty-four hours. In the paper read no hypothesis is put forward to account for this action. Your correspondent having talked the matter over with Dr. Rideal is not yet prepared to advance any theories at present. He can, however, state that Dr. Rideal's further experiments in this direction will probably be along the lines of pure electrochemistry rather than entirely chemical—if the action be chemical—as heretofore.

The Aluminium Industry in Great Britain.

The British Aluminium Company is about to greatly increase its capital, and the chairman, Mr. J. D. Bonner, stated at a recent meeting that not less than £490,000 additional capital would be needed. Of this, part is to be raised by a debenture issue, and part by the issue of shares. The Loch Leven water-power scheme is about to be undertaken, and the works at Larne, Greenock and Milton extended. It is hoped that the Loch Leven scheme will be finished by the end of 1906, in order to meet the market requirements which it is then anticipated will be considerable.

Metallurgical Papers Read in December.

The meeting of the Institution of Mining and Metallurgy, held on December 15, had before it four papers. The first, by Messrs. R. A. Thomas, and W. P. O. Macqueen, was concerned with "The dust in the air, and gases from explosives in Dolcoath Mine (Cornwall), and the efficacy of methods of dealing with them." For limiting the amount of dust in the air the authors have found a coarse spray most effective in machine drilling. In blasting, in order to clear the air of dust, a water blast of finely divided water and air, discharged immediately after the blast is extremely effective. The same device has also been found to immediately reduce the percentage of carbon monoxide in the air after an explosion from one-third to one-fifth of its volume. In regard to carbonic acid gas, continuous ventilation seems the only remedy.

The second paper, by Mr. E. Harris, dealt with the "Permanganate Chlorination Process at Bethanga" (New South Wales). The solvent employed is a modification of the Etard solution and consists of 110 pounds of commercial sulphuric acid, 90 pounds of common salt, and 12 pounds of potassium per 1000 gallons of water. These figures are more or less experimental, as the present process has only lately superseded the chloride of lime method. The advantages claimed for it over the older methods are (1) the less amount of free chlorine with consequent easier work to the men; (2) quicker extraction and less clogging of the filter beds; (3) the ready means of gauging the solvent strength afforded by the deep ruby color of the working solution; and (4) the obtaining of a cleaner precipitate with easier smelting.

Of the remaining papers, one was descriptive of the methods of mining carried on in the St. David's Mine in North Wales, and the other describing a new slag car.

The Institution of Mechanical Engineers met on the following evening, December 16th, to conclude their discussion on the "Impact Tests on the Wrought Steels of Commerce" (to which reference was made in my last letter) and to receive a new paper by Captain Sankey and Mr. J. Kent-Smith on "Heat Treatment Experiments with Chrome-Vanadium Steel." The paper is a long one, occupying some 48 pages of printed matter, illustrated with eleven plates of micro-photographs and diagrams of tests. Of the steels tested by the authors the best was that made from an 18-ton open-hearth furnace, and numbered 349. It contained 0.297 per cent carbon, 0.059 per cent silicon, 0.394 per cent manganese, 1.066 per cent chromium and 0.169 per cent silicon. Its tensile strengths, when raw from the rolls, showed a yield point of 369 tons, and an ultimate stress of 54.1 tons per square inch. Its elongation on 2-inch gauge was 24.0 per cent. On submission to impact tests, the figure for the average foot-pounds absorbed was 3.1, the test pieces 3/8 inch broad, the depth of the notch being such that 0.137 inch of metal remained. When tested on Professor Arnold's alternation testing machine, a test piece 4 inches long and 3/8 inch square withstood 1906 alterations before breaking down. The motion of the free end being 9-16 inch on either side of the center line.

The heat treatment was carried out in a muffle of the electric resistance type, consisting primarily of an earthenware tube around which was spirally wrapped thin platinum foil. The current was at 100 volts pressure, 27 amperes being required when the temperature of the muffle was kept at 1200° C. and 1250° C.

Except as regards such treatment as ruins the steel the chrome vanadium steel is distinctly less sensitive to heat treatment than is a pure carbon steel, while the impact figure is considerably larger. A great dissimilarity exists between annealed chrome vanadium steel, and the best oil-tempered carbon steel (containing 0.468 carbon). In the case of the annealed chrome vanadium steel the characteristics are maintained throughout the mass, whereas in the case of an oil-tempered steel in large masses the characteristics depend entirely on the depth to which the effect penetrates. The oil-tempering of chrome vanadium steel, however, further improves its quality in regard to elastic limit and impact test; on the other hand the alternations (Arnold's test) are diminished. When water-quenched, chrome vanadium steel agrees more closely with 0.254 carbon steel.

The diminished sensitiveness to heat treatment is a matter of the greatest practical value. It should be noted, however, that if the tensile strength is to be maintained, the annealing temperature should not exceed 650° C., while if the best all-round result is required, the annealing temperature should be 900° C.

Rather over six months ago a leading article in *The Engineer,* entitled "Ideal Steel," attracted considerable attention. It almost seems as though Captain Sankey and Mr. Kent-Smith have succeeded in producing a steel which answers to the ideal of *The Engineer.* Who, however, will be rash enough to say that further progress is unattainable, because so much has been achieved?

Market Quotations.

Chemicals have been remarkably steady during the month of December, the quotations for December 28 agreeing very closely with those for November 30. Caustic soda, copper sulphate, and bleaching powder are unchanged. Shellac has, however, fallen from 240s. to 200s. per cwt.

Fine Para rubber is still firm; there was a slight fall in the middle of the month to 4s. 10d.-5s. 0d., recovering to 5s. 3d. Meanwhile, small lots from Ceylon are fetching 5s. 10d. to 6s. 0d. per pound. Gutta percha is quoted at 8s. 0d.

Platinum still remains 81s. 6d. per ounce. Aluminium is stationary, and electrolytic copper bars rod and sheet are unchanged. Copper refined by the wet process weakened a little in the middle of the month, since recovering. Cleveland pig iron has risen further to 51s. 0d. Block tin, after declining to £133 per ton, has risen to £134-£135, or £3 less than at the end of November. Zinc sheets are stationary at £29.7.6 per ton. Brass sheets and tubes are practically unchanged.

London, January 4, 1905.

ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS.

By George P. Scholl, Ph. D.

Electric Furnaces and Furnace Products.

Manipulation of Refractory Material.—E. Thomson, Swampscott, Mass. Assignor to General Electric Co. Patent 778,286, December 27, 1904. Application filed August 28, 1902.

The invention relates to the manufacture of articles of various shapes, such as tubes, plates, etc., from a very refractory material, such as fused silica, in a furnace, with metallic terminals at opposite ends of the furnace. When it is desired to produce a tube, a carbon rod is introduced between the terminals, and is embedded in a charge of granulated quartz or sand. A very strong current is then passed through the rod, in order to melt those portions of the charge contiguous to it, the length of the time during which the current is passed determining the depth to which the fusion takes place, and consequently the thickness of the tube walls. It is stated that the fused coating of quartz around the carbon rod generally does not adhere to the rod, so that the latter can be removed, thus leaving a tube of quartz. If a curved instead of a straight tube is desired, the heating rod is made to curve correspondingly. In such case the heating rod, if it cannot be conveniently withdrawn, is burned out by heating the whole to a high temperature. When the tube to be produced is of relatively large diameter, the heating rod is made hollow. In this case it is provided with a number of fine perforations to allow for the escape of any gases generated during the process. These perforations are so fine that the granulated quartz of the charge cannot pass through them, but they prevent the possible injury to the outline of the tube due to the action of these gases. The exterior of these tubes, which is more or less granular according to the fineness of the raw material used, may be smoothed down by progressively heating it in the electric arc from end to end. The method for the production of a plate or a slab of fused quartz consists in the heating of a perforated plate of carbon, upon the face of which a layer of granulated quartz has been placed, by a heavy current. After the mass has cooled, the plate of fused quartz may be removed, and its surface smoothed off by the electric arc. A series of devices are described in the specification for the production of more complicated structures.

Production of Metallic Magnesium.—T. L. Roberts, New York. Patent 778,270, December 27, 1904. Application filed May 31, 1904.

The process is carried out in a vessel shown in cross-section in Fig. 1. It consists of an iron pot *A,* which is set into a furnace *B* of refractory material and heated from the outside by means of a burner K. It is covered with a slab of slate D, provided with an opening E through which the anode rod G, its surface [illegible]. The vessel itself constitutes the cathode. An aperture *H* in the lid serves for feeding the electrolyte, while a tube *N* is provided for carrying off the gases generated during electrolysis. A draw-off tube *O* is used to remove the molten metal, which accumulates on the surface of the bath. During this operation, of course, the heating arrangement has to be withdrawn. Another draw-off cock *J*, at the bottom of the vessel, serves for the withdrawal of the contents of the pot; the cock may be used for withdrawal of the metal, if the electrolyte is of such low specific gravity that the metal will sink to the bottom. A ring *K*, of non-conducting material, preferably porcelain, is used to keep the magnesium, which rises along the sides of the vessel, from coming into contact with the anode. The anode should fit tightly, as the less air is admitted to the pot the better it is, the air seeming to accelerate or assist volatilization more than does an atmosphere of the electrolytic gases. The main point of importance about the process is the composition of the electrolyte, it being claimed that the addition of fluoride of magnesium to the bath will result in the recovery of the metal with little or no loss of the salts of volatilization. The composition of the bath is preferably as follows: Lithium fluoride 120 parts, potassium chloride 80 parts, sodium chloride 40 parts, ammonium chloride 20 parts, magnesium chloride 10 parts, magnesium fluoride 80 parts, and magnesium oxide or carbonate 20 parts. The mixture becomes liquid at a dull red heat, and is then electrolyzed, chlorine and oxygen being formed at the anode. It is stated that the above mixture may be varied considerably and good results still be obtained; certain of the constituents may be omitted or others substituted, or the proportions carried, so long as magnesium fluoride and a chloride of the alkali group are used. A certain amount of lithium salts is advantageous in lowering the specific gravity of the bath, so that the liberated magnesium sinks readily to the bottom, while it also lowers the fusion point of the electrolyte. The ammonium and magnesium chlorides also lower the specific gravity and fusing point of the bath, but as they are very volatile, only limited amounts should be used. The oxide or carbonate of magnesium is not essential, but if added in such amounts that they will be dissolved or suspended in the electrolyte, more or less of it will be reduced along with the fluoride and chloride. They, however, raise the specific gravity of the bath, a fact which has to be taken account of if the metal is to be drawn off from the bottom of the apparatus. No data concerning voltage or current strength are given

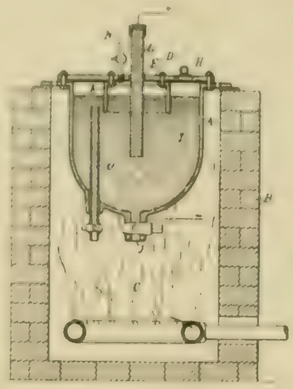


FIG. 1.—APPARATUS FOR PRODUCTION OF METALLIC MAGNESIUM.

Electrolytic Production of Metals and Compounds. Electrode.—E. J. Briggs, Everett, Mass. Patent 776,400, December 6, 1904. Application filed September 19, 1903.

The electrode is intended to be used in apparatus for the decomposition of the alkaline chlorides, particularly in the [illegible] apparatus described in Electrochemical Industry, Vol. I., page 392. The construction provides for an efficient electrical connection, to prevent loss of current as much as possible, while at the same time preserving the feature of easy accessibility for cleaning and adjusting purposes. A block [illegible] of angle iron, upon the horizontal flange of which it rests. The individual carbons are provided with elliptical tops and arranged in pairs, the tops of each pair being connected by a ribbon of copper, which can be tightened by bolts extending through washers into threaded copper lugs. The latter are integral with a copper plate resting on top of all the electrodes in the block. This plate is connected with the cable from the dynamo.

Process of Producing Salts of Hydrosulphurus Acid.—A. Frank, Charlottenburg, Germany. Patent 777,669, December 20, 1904. Application filed March 8, 1900.

The process aims at producing the hydrosulphites of alkaline earth or earth metals by electrolyzing a practically neutral solution of the bisulphite salt of the metal, the hydrosulphite of which is desired. Electrolysis is carried out in a rectangular vessel which serves as the anode compartment, a series of porous cells, which rest on the bottom of the vessel, constituting the cathode compartments. The solution in the anode compartment consists of dilute sulphuric acid or a suitable salt solution, while the cathodic electrolyte is the bisulphite salt solution mentioned above. The reaction taking place during electrolysis, when calciumhydrosulphite is to be produced, is stated to be as follows: $CaH_2S_2O_6 + 2H = CaS_2O_4 + 2H_2O$. The output of salt with respect to the quantity of current is stated to be the greater, the less acid is contained in the cathode liquid during electrolysis, it having proved advantageous to employ a solution of bisulphite of calcium, which, if possible, does not contain more sulphurous acid than corresponds to the formula $CaSO_3 + H_2SO_3$. The hydrosulphite of calcium, CaS_2O_4, is stated to deposit at the cathode in the form of a heavy crystalline mass, which falls down to the bottom of the cathode receptacle. In this way a decomposition of the salt, either by electrolysis or by the influence of the atmosphere, is avoided. The cathodic electrolyte circulates through the cathode compartments, flowing from a reservoir to the first one and successively through the others, being returned from the last one of the series to the reservoir by means of a pump. Salts of barium, strontium, magnesium and aluminium are stated to be produced in a similar way. No particulars are given concerning the current density or the voltage required for the process.

Process of Reducing Lead Ores.—P. G. Salom, Philadelphia. Patent 778,901, January 3, 1905. Application filed April 17, 1903.

The process, though claimed to be applicable to all lead ores, including argentiferous lead, is described as applied to the reduction of galena, presumably as carried out in Mr. Salom's works at Niagara Falls, which have been described repeatedly in Electrochemical Industry. The finely ground ore is spread continuously on the cathode in a very thin layer, and is left there until it is completely reduced, the reduced lead being also removed continuously. The apparatus in which the reduction is carried out is shown in cross-section in Fig. 2. Its construction varies to some extent from that described and illustrated in Electrochemical Industry, Vol.

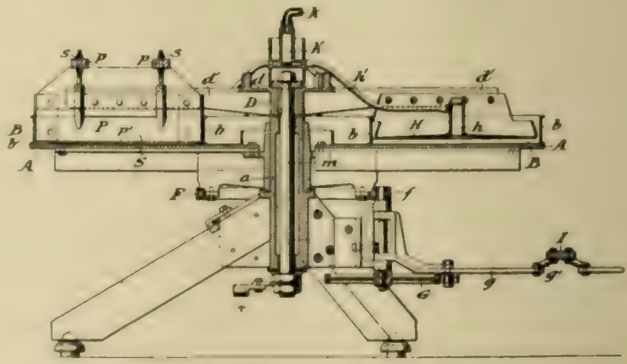


FIG. 2.—APPARATUS FOR CATHODIC REDUCTION OF LEAD FROM GALENA.

I., page 392, especially as regards the movable part, the anode being movable in the former construction, while the cathode was stationary, whereas in the one under consideration the cathode is movable and the anode stationary. The apparatus consists of a rotating table *A*, which is supported by a fixed central port *a* and carries concentrically an annular electrolytic cell *B*, with inner and outer peripheral walls *b b*. The cell is composed of antimonial lead, and is made the cathode bv electrical connections through the central post, which, as shown, is insulated from the floor on which it rests. Movement is imparted to the table by a gear wheel *F*, which meshes with a pinion *f* sitting on the same shaft as the ratchet wheel *G*. The latter is moved by a pawling lever *g*, which is slowly oscillated by a bar *T*, through link connection *g*. The bar moves to and fro, and a whole series of apparatus may thus be kept in motion. The post *a*, around which the table swings, carries above the table a spider frame *D*, insulated from the post and consisting of a central disc *d* and a number of radially-extending arms *d′ d′*. The spider frame supports, within the electrolytic cell, an anode in the form of an annulus, which, however, does not form a complete ring, but lacks one segment between two of the arms of the spider frame. The annulus is made up of a series of segmental plates *H*, of antimonial lead, each plate bridging a space between the arms of the spider frame, and having its under surface slightly dished upwards towards a central aperture *h*. This opening leads to a short pipe, which communicates by a rubber tube *h′* with a central chamber *K*, and serves the purpose of conducting away the gases generated during electrolysis. The entire series of anode plates is constituted a single anode by connection to the spider frame by means of the central insulated spindle *m*. The ore is spread upon the rotating table by a supply box *P*, supported from two bars *p p*, which bridge the opening between those two arms of the spider frame, where the segmental plate is lacking, all connections between the box and the spider frame being insulated. The side wall of the box *P*, away from which the cathode rotates, does not reach quite to the bottom of the cell, thus leaving an adjustable slot *p′* for feeding the ore onto the cathode plate, while outside of the other side wall of the box *P*, towards which the cathode rotates, there is fixed a vertical scraper plate *S*, adjustable by set screws *s s*. Finely ground galena of about 70 to 80 per cent of lead is fed continuously at about the rate of ten pounds per hour for a current of 1500 amperes, distributed so as to provide about 60 amperes per square foot of active cathode surface. The width of the feed slot is so adjusted that the total amount of ore resting on the cathode at any one time is about 15 pounds, the layer of ore being 1-10 to 1-30 inch thick. The rate of rotation is about one turn in an hour and a half. The electrolyte is a 10 per cent sulphuric acid solution.

Electrolytic Apparatus.—C. P. Townsend, Washington, D. C. Patent 779,383, January 3, 1905. Application filed April 12, 1902; divided and refiled May 10, 1904.

Electrolytic Process.—C. P. Townsend. Patent 779,384, January 3, 1905. Application filed April 12, 1902; divided and refiled May 11, 1904.

The process is intended especially for the electrolysis of aqueous solutions of alkali metal salts, and its distinguishing feature is the liberation and recovery of the products formed at the cathode, in the presence of a liquid which is substantially immiscible with and inert towards the products. The apparatus, as applied to the electrolysis of an aqueous solution of sodium chloride with production of caustic soda and chlorine, is shown in transverse vertical section in Fig. 3. It comprises a central anode compartment 1 and two cathode compartments 3, 3. The anode compartment is filled with the sodium chloride solution, a continuous flow from below upwards being maintained by means of pipe 11 and outlet 12. The anodes 2 are plates of Acheson graphite, carried by a conductor bar 2′ of similar material. Diaphragms 5 of asbestos or other suitable pervious material separate the anode compartment from the cathode compartments. They are supported between the perforated sheet metal or wire gauze cathodes 4 and the plates 6, which latter have numerous small perforations, except at their upper portions. These plates may be of hard rubber, earthenware, hardened asbestos or other suitable non-conducting material. Plates 7, 7, preferably of iron, are placed parallel to the cathodes in such a manner as to provide a comparatively narrow space between them and the cathodes, while leaving a relatively wide space towards the outer wall of the cathode compartments. Coils 8, 8 are introduced for heating or cooling the liquid in order to maintain its temperature at any desired point. The liquid in the lateral compartments should be of such character as to separate readily from the caustic soda and to be so inert with respect to the caustic that it will not undergo any rapid, injurious change or modification in use. As such liquids, the non-saponifying oils are mentioned. The sodium set free at the cathodes is in part or entirely oxidized by the solution in the diaphragm, or percolating through the diaphragm, the caustic soda solution thus formed under the oil detaching itself freely from the cathodes. The hydrogen resulting from the reaction of the sodium upon the water, in passing upwards through the narrow spaces between the cathodes and the plates 7, induces a strong circulation of the oil, which further aids in detaching the globules of caustic sôda, and thus removes them quickly from the field of electrolytic action and prevents their diffusion into the anode compartment. It is stated that current densities equal or exceeding one ampere per square inch of cathode surface are utilized with a high degree of efficiency.

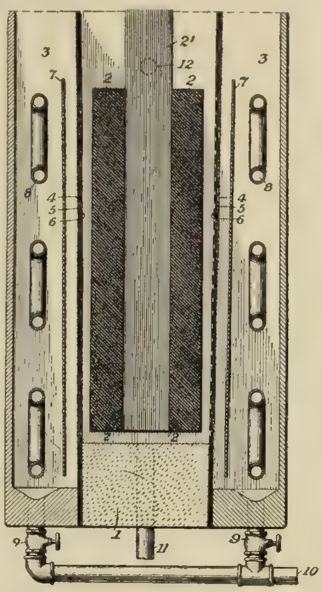


FIG. 3.—DIAPHRAGM CELL FOR ELECTROLYSIS OF SODIUM CHLORIDE.

Process of Electrolytically Refining Lead.—A. G. Betts, Lansingsburg, N. Y. Reissued patent 12,301, January 3, 1905. Original 713,277, November 11, 1902. Application for reissue filed April 22, 1903.

Patent 713,277 was described in Electrochemical Industry, Vol. I., page 133, while the process was mentioned repeatedly, and also fully described by Mr. Betts in Electrochemical Industry, Vol. I., page 407. In the present specification, he states that in practicing the invention he prefers a current strength of ten to twenty amperes per square foot of cathode surface, and a corresponding tension of .15 to .35 volts for each element (presumably cell). With higher currents the lead becomes harder and more brittle, and takes on a whitish color and a silvery luster. In general the deposited lead is apt to be slightly stiffer than ordinary lead, but the stiffness may be regulated at will by varying the current density and the quantity of organic reducing agent, such as gelatine, etc., in the solution, one part of gelatine, by weight, to five thousand parts of solution having been found preferable. It is also claimed that the process is well adapted for electroplating metals with lead, as the deposit obtained is a firmly-adherent, dense coating.

Storage Batteries.

Process of Making Spongy Lead for Storage Battery Elements.—C. J. Reed, Philadelphia. Patent 778,894, Jan. 3, 1905. Application filed May 7, 1903.

In carrying out the process one or more compounds of lead [illegible] pulverized soluble [illegible] such as the oxide, chloride or sulphate of lead, are mixed with finely divided metallic zinc or zinc dust, in chemically equivalent proportions. This mixture is subjected to the action of liquid of such composition as to decompose the lead salt into metallic lead, the acid radical uniting with the zinc to form a soluble zinc salt, which is then leached out with water. Among the materials which may be used are red lead and zinc dust in chemically equivalent proportions, and a solution of zinc chloride. The mixture of powdered materials may be treated with the solution while in a loose form to produce the finely divided lead for subsequent application to a battery plate. The mixture may also be slightly dampened and applied to the plate or battery frame as a solid coherent mass. It is then dried out sufficiently to make it suitably absorbent and subjected to the reducing liquid until the composition has been completely converted into metallic lead and a soluble zinc salt. It is then finally subjected to a water bath for a sufficient length of time to remove the zinc salt, and the plate is then ready for use in a storage cell. It is sometimes found advisable to mix with the powdered material a soluble, inert, granular material, such as sugar, salt or zinc sulphate, which may be readily dissolved out after the conversion of the lead compound to metallic lead has taken place, thus leaving pores for the entrance of the battery electrolyte.

Protective Coating for Storage Battery Plates.—Achille Meygret, Paris, France. Patent 779,553, January 10, 1905. Application filed April 6, 1904.

The coating described in the present specification embodies an improvement as compared with those described in the inventor's preceding patents (ELECTROCHEMICAL INDUSTRY, Vol. III., page 31; Vol. II., page 206), inasmuch as it is not applied directly to the active material of the plate, which might thereby be subjected to ill effects, but a film of a different material is interposed between the active material and the coating. This material is of a soluble nature, so that, when the battery is in use, it will be dissolved by the battery liquid, thus leaving the protective coating still surrounding the active material of the plate, but not firmly adhering to it. The substance preferably used for this intermediate film is a syrup formed of common lump sugar or saccharose, or glucose dissolved in water to such a consistency as to furnish a bath, into which the grid or metal of the plate, with the active material upon it, can be dipped. The syrup adheres to the surface and forms a film, the plate being then dipped into the solution, which is to give it the protective coating, and which solution consists of tetraacetate of cellulose or tetrabutyrate of cellulose or castor oil, essence of turpentine and nitrate of cellulose. The protective coating is not affected by the battery liquid, but the syrup film is dissolved out as described above.

Galvanic Elements.

Electric Battery.—W. H. Gregory, Vallejo, Cal. Patent 777,851, December 20, 1904. Application filed May 14, 1904.

The invention relates to a battery of the dry type; provision is made in its construction for resupplying the cell with moisture, after it becomes exhausted. For this purpose a glass tube is embedded within the cell, which lies against the inner surface of the zinc or metal casing, and is filled with water or any desired liquid. When the battery is exhausted, this tube is broken by striking the casing.

Electric Battery.—A. Gabrielson, Milwaukee. Patent 778,653, December 27, 1904. Application filed July 21, 1904.

The battery is a dry cell for open circuit only, and consists of a zinc cup coated with mercury on its inner surface. The cup is provided with a lining of paper soaked in dilute sulphuric acid. The cell is filled with a mixture of coke dust, powdered graphite and sal-ammoniac crystals, moistened with potash solution. This mixture is packed around a battery car- [illegible]

[illegible] Philadelphia. Patent 778,893, [illegible] Application filed July 12, 1904.

The battery is divided into two compartments by a porous partition, preferably of zigzag form. One compartment contains a solution of one or more ferrous salts, such as ferrous chloride, or ferrous sulphate, or a combination of two or more of such salts, while the other compartment is filled with a solution of one or more ferric salts, such as ferric chloride or ferric sulphate. In the ferrous electrolyte there is located an iron electrode, while the electrode in the ferric solution is electronegative to iron, and may consist of carbon, platinum, platinized carbon, etc. When the battery circuit is closed, the iron electrode becomes the anode and dissolves, thus forming a ferrous salt, while the ferric salt in the other compartment is also changed into a ferrous salt. This solution may be regenerated by oxidizing agents, such as oxygen, chlorine, bromine, ozone or hydrogen peroxide, preferably blown through the solution. When the battery is exhausted through exhaustion of the ferric salt, it may be regenerated by passing a current in the opposite direction through the electrolytes, which action deposits iron from the ferrous salt upon the iron electrode and reconverts the ferrous salt in the other compartment into ferric salt. When regenerated in this manner, the battery becomes an accumulator. In that case an electrode is preferably employed in the ferrous salt, which has a base of copper, carbon or other suitable electronegative substance, upon which a coating of iron has been electrolytically deposited. The battery may then be discharged until nearly all the iron is dissolved, and may be recharged without any substantial alteration in the form of the electrode, whereas with an iron electrode the latter might be dissolved irregularly, and would be unlikely to retain its original form when subjected to a series of charges and discharges.

Dry Cell.—Walter Stoeckigt, Greiz, Germany. Patent 778,912, January 3, 1905. Application filed May 12, 1904.

The cell is provided with a filling and replenishing tube for the purpose of introducing the electrolyte, and replenish it whenever desired. The tube has a number of perforations distributed along its length.

Electric Battery.—F. P. Dewey, Washington, D. C. Patent 779,589, January 10, 1905. Application filed March 25, 1902, renewed May 19, 1904.

The battery comprises a zinc plate supported from the top of a battery jar by a thimble of gutta percha or similar material. Within this thimble and above the zinc plate is placed a porous cup, upon the bottom of which rests a piece of asbestos cloth, which in turn carries the other electrode, consisting of a coil of copper or nickel ribbon. The latter is wound in the form of a spiral. The cathode is not immersed in the electrolyte, the latter being brought to its lower edge by capillary action of the pores of the diaphragm and allowed to spread in a thin film over its whole surface. Polarization at the cathode is claimed to be overcome by this means.

Battery Zinc.—F. J. Delarie, Pittsburg. Patent 779,693, January 10, 1905. Application filed July 1, 1904.

The invention relates to means of supporting the battery zinc, which consist in a bridge piece of stamped sheet metal resting on top of the battery jar, provided with shoulders and with a central key slot adapted to receive and securely hold the stem of the zinc electrode.

Battery.—N. M. Watson, Detroit. Patent 777,985, December 20, 1904. Application filed June 13, 1904.

The invention relates to a battery more particularly designed for electrotherapeutic purposes. The construction is therefore simple and compact, in the shape of a cylindrical casing containing a dry cell and the induction coil. No external wires are necessary, and the ends of the casing need only be grasped by the hands if it is desired to subject the system to the effect of the induced current. The constrcution of the cell is described in detail.

Thermo-Electric Element.—A. L. Marsh, Lake Bluffs, Ill. Patent 779,090, January 3, 1905. Application filed June 4, 1904.

The inventor states that he has found that the chromium group of metals, comprising chromium, molybdenum, tungsten and uranium, when alloyed with nickel, may be formed into elements, strongly electronegative to elements composed of an alloy of nickel and copper. The positive element is preferably made of an alloy of 35 per cent nickel and 65 per cent copper, which has a fusing point much above 1050° C., the approximate fusing point of copper. An alloy of chromium and nickel, or molybdenum and nickel, the nickel predominating in each case, is preferably employed as the negative element. Any of the alloys of nickel and a metal of the chromium group is electronegative to the alloy of nickel and copper, and a bar of either may be readily welded to a bar of the other to form a highly-refractory non-oxidizable, tough, durable and efficient couple, which may be subjected to a concentrated, intense heat, without danger of injury.

Miscellaneous.

Production of Paper Pulp. I. Kitsee, Philadelphia. Patent 775,829, Nov. 22, 1904. Application filed May 1, 1903.

The object of the invention is to effect an improvement in the present process of extracting the resinous and other intercellular matters from wood fiber which is to be used for the manufacture of paper pulp, which extraction, up to the present time, has been accomplished by boiling the wood in caustic alkali with addition of caustic lime and carbonate of soda. For this purpose it is proposed to use the ordinary digester as used for the treatment of wood chips, with such alterations as are necessary to carry out the process outlined below. The boiler shell is connected into an electric circuit as the cathode, and a structure is arranged in the center composed of a series of pieces of porous pipe, superposed upon each other and cemented at the joints so as to constitute a central tubular compartment. An anode consisting of platinum or carbon is introduced into this compartment and the whole of the apparatus is filled with a strong solution of common salt, after the outer compartment has been filled with the wood chips or fibers intended to be treated. Steam or another source of heat is then applied and the current passed through the apparatus, with the result that alkali will be formed in the outer compartment, which will extract the resinous and intercellular matter from the wood, while chlorine is given off in central compartment. It is proposed to use the chlorine for bleaching the pulp coming from the digester, after it has passed through the usual process of washing. It is stated that a current density of about 100 amperes for a digester of say 8 feet in diameter by 6 feet in depth is sufficient for all practical purposes, but where a stronger alkalinity is required in a shorter time the density should me raised accordingly.

Apparatus for the Treatment of Gases.—E. A. Werner, St. Louis. Patent 777,987, December 20, 1904. Application filed June 13, 1904.

Process of Generating Gases from Air.—Same inventor. Patent 777,988, December 20, 1904. Application filed August 6, 1904.

The apparatus consists of a chamber with an insulating bottom, on which are supported two current terminals, formed of thin strips of metal, between which an arc is struck. A gas supply pipe, by means of which compressed air can be introduced into the chamber, terminates below the arc, which is by this means deflected and elongated in a lateral direction, such deflection varying with the pressure and amount of the air supplied. At the same time, the latter serves the purpose of cooling the terminals. A further regulation and control of the back pressure in the apparatus and the amount of deflection of the arc is furnished by outlet valves on the conduits which conduct the treated gas to the various points where it is to be used. The second patent protects the process of carrying out the above operations.

Apparatus for the Treatment of Gases.—Same inventor. Patent 777,989, December 20, 1904. Original application filed June 13, 1904. Divided and refiled August 11, 1904.

Apparatus for the Treatment of Gases.—Same inventor. Patent 777,990, December 20, 1904. Original application filed June 13, 1904. Divided and refiled August 11, 1904.

Device for projecting air against electric discharges.—Same inventor. Patent 777,991, December 20, 1904. Application filed August 29, 1904.

The apparatus described in the first patent is the same in principle as that described above, except that a conduit leads from the electrode chamber to a cylinder in which is located a piston held in position by a spring. The piston rod is connected with a switch lever, which in turn cuts in or out, as the case may be, a certain number of sections into which the secondary coil of the transformer is divided. Accordingly, as the pressure in the apparatus rises or falls, a decreasing or increasing number of sections is cut into the circuit, and the amount of current supplied to the arc is thus correspondingly automatically lessened or increased. The second patent provided means for cooling the transformer by locating it in a chamber situated below the treatment chamber and passing the air to be treated around it before it passes through the arc. The third patent protects the device for injecting the air, so as to divert the course of the arc and to cause it to spread apart, for which purpose the supply pipe is covered by a cap provided with a number of small holes. The pressure of the compressed air is preferably from twenty to fifty pounds.

RECENT METALLURGICAL PATENTS.

Iron and Steel.

In our Vol. II., page 503, an account was given of James Gayley's now famous large-scale experiment of the application of dry-air blast to iron production. The air is dried by artificial cooling, a patent to this effect being mentioned in our Vol. II., page 469. A new patent of Mr. Gayley (779,037, January 3) gives some further details which throw light on some causes of the great success of his method. When the air is first dried, and the moisture reduced to a small and uniform percentage, it is, of course, possible to reduce the percentage of the coke employed in the charge by an amount equal to what would have been required to dissipate the moisture from the air. For instance, when the moisture contained in the air has been lessened from an atmospheric content of 6 grains of water per cubic foot of air to 1.5 grains per cubic foot, the saving of fuel calculated from the heat required to dissipate the 4.5 grains of eliminated moisture is about 7 per cent. Mr. Gayley has found, however, that if he goes further and reduces the fuel by 16 to 20 per cent, he not only maintains the furnace in successful running order, thus saving the cost of the fuel, but he improves actually the quality of the iron by lessening the extent to which it is contaminated with sulphur and phosphorus; he also obtains greater uniformity of work, prevents waste of ore carried away from the furnace by the gases, reduces the temperature of the gases leaving the throat of the furnace, and increases the output of metal about 20 per cent. The inventor attributes "these results, in part, at least, to the increased reducing power impared to the furnace gases by the removal of the moisture from the air-blast; but it may be due in part to other causes." Some notes on critical remarks by foreign authors with respect to this subject are given on another page of the present issue in the Synopsis of Periodical Literature.

J. A. Mathews (779,171, January 3) patents "high-speed steel containing not less than 0.10 per cent of vanadium." (First claim).—High-speed steel is defined as steel hardened by air and containing, in addition to carbon, either molybdenum or tungsten, either alone or in combination with chromium. Vanadium and ferro-vanadium has proven extremely attractive to steel men in recent years, though very little exact information is so far available as to the action of vanadium in steel. The present inventor adds about 0.50 per cent of vanadium to high-speed steels, containing from 6 to 10 per cent of molybdenum (or in the alternative from 12 to 20 per cent of tungsten) and from 0.5 to 1 per cent of carbon. He states that by the addition of vanadium, "the efficiency and endurance of the tool were more than doubled; there was also a great increase in the resisting power of the tool as against the sudden cross-sectional strains and shocks which are not infrequent in severe usage.

C. W. Cowen (779,074, January 3) patents details of construction of a frame for covers for crucible steel-melting furnaces.

F. A. Reynolds (780,128, January 17), in a brazing compound for welding cast iron, makes use of the reducing properties of aluminium. Equal parts of aluminium powder and carbon are mixed with oil or water, so as to form a paste. This paste can be used for painting the ends of the parts to be united, after which these ends are placed in the fire and heated to a red heat. After this borax and the ordinary brazing compound or spelter can be melted on over the joint, and allowed to cool. These parts are then filed off, as usual. "The aluminium has so great an affinity for oxygen that it reduces all other oxides present, and leaves an absolutely metallic surface at the parts which are to be united, so that there is nothing to prevent a perfect union." He prefers to use the brazing compound in stick form; the ordinary spelter or brazing mixture, a quantity of aluminium powder, and some oxide of iron are mixed together with a little copper to toughen the material. This mixture is then heated up to a red heat, when it melts down and forms an alloy. This alloy, in the form of sticks, is applied directly to the heated ends of the parts to be brazed.

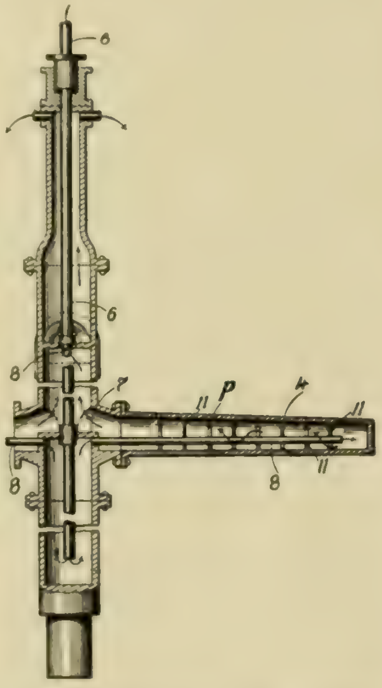


FIG. 1.—CONSTRUCTION OF RABBLE-SHAFT AND ARMS.

IRON AND ZINC.

S. Peacock (779,310, January 3) proposes to treat ores containing sulphides of iron and zinc as follows. The ore is pulverized and roasted. The roasted ore is then either heated in a furnace while mixed with carbon or subjected to the action of a reducing gas in a furnace at such a temperature that while the iron oxides are reduced to metallic iron the zinc oxides remain unreduced. After all the iron has been practically reduced to a metallic condition, the furnace is cooled down in a reducing atmosphere. The metallic iron is then removed from the ore, either by a magnetic separator or by gravity. The oxide of zinc which is then practically pure, so far as the presence of iron is concerned, is then subjected to the distillation process in the usual way.

ROASTING FURNACES.

Frank Klepetko (779,717, January 10) patents air-cooling devices for the McDougall type of ore-roasting furnace. Fig. 1 shows a section of the rabble-shaft and arms. The air-feed pipe 6 is located within the shaft and extends to a short distance from the closed bottom of the latter, the lower end being open, so that air blown through six discharges into the shaft, near the bottom, as indicated by the arrows. The shaft itself is divided into a series of chambers, separated from one another by the transverse partitions 7, slightly above the bottom of the adjacent rabble-arms 4. The air is thus forced from the shaft into the pipes 8, which extend into the hollow arms 4. The inner opposite wills of these arms are provided with an alternating series of deflecting ribs 11, the free edges of the ribs being concave, whereby the concavities of the entire series of ribs from an opening for the free passage of the pipe 8. The air, when discharging from the end of 8 into 4, is then forced to pass through the space between the walls of 8 and 4 in a zig-zag way, and is then discharged into the next higher chamber of the hollow shaft. In this way the cooling air is forced to come sufficiently long into contact with the surfaces to be cooled.

A. R. Meyer (780,115, January 17) also patents cooling devices for the McDougall type of roasting furnaces. For cooling he employs air charged with moisture, water flowing downward along the inner face of the rabble-shaft, while air is circulated in such a manner that the vapor rising from the heating of the water is carried by the air in contact with the shaft, and with other parts to be cooled. Water is introduced at the upper end of the shaft from a supply pipe, and is discharged into an annular gutter within the shaft near its tow. Similar gutters are provided at intervals, so as to hold limited amounts of water along the line of shaft. The connection between the shaft S and the hollow rabble arms is shown in Fig. 2. Into the hollow arms a current of air or air charged with vapor, is blown through 14 from the supply pipe P, centrally located within the shaft. The plate separates the interior of the shaft from the interior of the rabble arm, and the latter space is partly divided by the partition 22 into a lower and upper half. The cooling air introduced through 14 passes through the lower half to the end of the rabble-arm and returns through the upper half to be discharged through 18. In order to avoid the overheating of the necks or bosses, some of the water which flows downward is discharged around the inner part of the bosses.

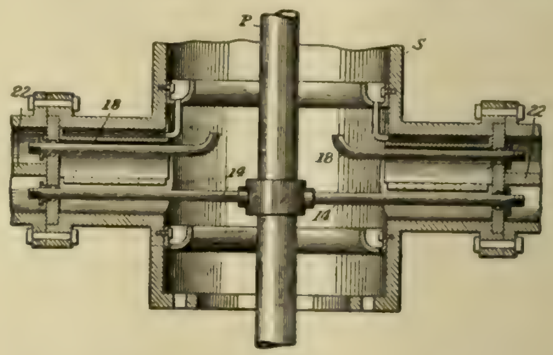


FIG. 2.—CONNECTION BETWEEN SHAFT AND RABBLE-ARM.

H. C. Davey (779,119, January 3) patents mechanical details of construction of a roasting furnace of that class in which a horizontal or slightly inclined cylinder is revolved upon its axis, the ore passing gradually from one end to the other. The inventor provides a series of revoluble inclined barrels, each having stationary heads and furnaces connecting therewith, through which heat is delivered to pass through the barrels. These barrels are set with relation to each other and provided with intermediate automatic discharge chutes from the end of one cylinder, and receivers into which the ore passes and by which it is delivered to the succeeding cylinder.

Recovery of Metal Values in Slags and Mine Waters.

R. Baggaley (779,252, January 3) patents a process for recovering the copper held in solution in mine waters—as, for instance, those discharged from the mines in the Butte district, and for recovering simultaneously the copper, gold and silver values in the ferruginous copper-smelter slags commonly called in the trade "foul slags." From the latter he separates by a special smelting process, the iron which is drawn from the separating furnace and granulated by pouring it in a molten condition into water. This granulated iron is an excellent agent for precipitating the copper from the mine waters. These waters are passed through successive vessels containing the iron and are caused to drop from one vessel to the other; the copper is thereby precipitated.

Cyanide Process.

T. B. Joseph (780,293, January 17), patents a modification of the ordinary cyanide process for the extraction of metals from ores. His leaching solution is a mixture of "water, sodium cyanide, bicarbonate of ammonium, bromo cyanide, barium dioxide, and hydrate of calcium" while compressed air is forced upwardly in the tank from the bottom. "The cyanogen and the cyanogen bromide in this solution dissolve the precious metal, while the ammonium dissolves not only some silver, but also the copper, nickel and zinc, if there is any therein, and thereby causes a better extraction of the silver, as well as of the gold, with this solution than the straight cyanide process would obtain, as the ammonium therein holds the most of the dissolved copper, zinc and nickel leached from the ore, and prevents them from occupying so much of the cyanide as they otherwise would. When it is desired to extract much copper, silver, nickel, or zinc, as well as the precious metals from the ore with this solution, one should increase the quantity of the ammonium bicarbonate therein accordingly; but when it is only the precious metals in the ore to be extracted then only a small quantity of the ammonium bicarbonate need be used." As a general rule, the author says that there should be used in proportion to one ton of the water of solution, about 2 pounds of sodium cyanide, 1 pound of ammonium bicarbonate, enough of the bromo-cyanide solution to contain about 2 ounces of bromine, about 4 ounces of barium dioxide, and enough of the calcium hydrate to contain about 1 pound of the oxide of lime before being dissolved in the water.

L. E. Porter (778,547, December 27), treats cyanide solutions containing precious metals and copper as follows. He first precipitates gold and silver by means of zinc shavings; this leaves a solution of copper cyanide, which is subjected to the action of zinc dust and ammonia precipitating metallic copper, and leaving an alkaline cyanide solution suitable for further use. The process is said to be particularly applicable when the solution already contains lime. The resulting solution is used to leach a further quantity of ore, and in this operation the zinc cyanide and alkaline cyanide are all effective in dissolving the copper, silver and gold of the ore. By a continued repetition of this process zinc cyanide would, of course, accumulate in the regenerative solution; to overcome this, sufficient carbonate of soda is added from time to time to precipitate a portion of the zinc, as a double carbonate of zinc and lime, which, being insoluble, is precipitated. The inventor says that the process has been sucessfully used in practice, and that "the following theoretical conditions explain the operativeness of the process. When the alkaline or free cyanide is completely saturated with the metal cyanides which are known to be good solvents for metals—especially the cyanides of zinc and copper, which are good solvents for gold and silver—then ammonia will cause the metals to precipitate in the presence of zinc. The ammonia acts in the place of free cyanide. The zinc dust used is an impalpable powder, and each particle seems to be entirely consumed with the first precipitation of copper. The zinc-shavings are comparatively coarse, and after the surface has been entirely coated over, first by copper and later by gold and silver, there still remains a mass of zinc, which continues to precipitate gold and silver from the copper-cyanide solution without further precipitation of copper. The general operation may therefore be assumed to be as follows: The copper is first precipitated on the zinc-shavings, and is then dissolved, being replaced by gold and silver, and the surface exposed by the zinc-shavings should be sufficient to enable precipitation of all the gold and silver from solution by the time that the zinc-shavings are full coated. When all the gold and silver has thus been deposited, the addition of zinc in the form of dust exposes a far larger amount of surface, so as to precipitate all of the copper present.

I. Anderson (778,348, December 27), patents a process of recovering precious metals from solution, based upon the fact that when sulphuric acid is added to cyanide solutions of the precious metals in which sulphides and chlorides are present, the sulphuric acid so added will combine with the potassium of the potassium cyanide to decompose the double cyanides formed by the action of the potassium cyanide on the precious metals. The precious metals freed from combination with the potassium cyanide will then combine with the sulphur and chlorine to form insoluble sulphides and chlorides, which will be precipitated, leaving the cyanogen in the liquid in the form of hydrocyanic acid or in combination with the radicals with which the sulphur and chloride has been combined. The precious metals having been precipitated as chlorides or sulphides, the supernatant solution may be decanted off and the potassium cyanide regenerated by decomposing the potassium sulphate formed by the action of the sulphuric acid and removing the sulphuric acid from the solution. This decomposition of the potassium sulphate and the removal of the sulphuric acid is accomplished by introducing lime into the solution decanted off the precipitated gold and silver values, the lime so added causing the decomposition of the potassium sulphate with the formation of insoluble calcium sulphate, and the simultaneous regeneration of potassium cyanide.

Briquetting.

U. Wedge (780,464, January 17), patents a method of briquetting iron pyrites fines for desulphuration. He has formerly patented a method of briquetting with sulphate of iron as a binder. He now states that iron pyrites reduced to a powder and mixed with sufficient water to render it plastic will, when subjected to air and especially under the influence of heat, rapidly oxidize and produce iron sulphate which acts as binder so that they may be easily pressed into the form of briquettes.

A. Ronay (778,899, January 3), patents a process for briquetting finely granulous ores, or the like by the following method. The powder or granular material is subjected in a dry or slightly damped condition to a pressure of at least 800 atmospheres up to 2000 atmospheres in such a manner that the pressure is gradually increased for the purpose of letting the air completely escape. The highest pressure at which material becomes plastic is applied only at the last stage of the pressing process, whereupon the briquettes are further subjected to the action of carbonic acid gas. The apparatus for applying the pressure is described.

Miscellaneous.

A. E. Manchester (779,953, January 10), patents details of construction of a smelting furnace, the feature being the provision of a new cut-off valve which is interposed between the furnace and the fore-hearth, whereby the furnace and the fore-hearth nozzles may be brought into open communication when the valve is in one position, and be closed when the valve is in another position.

[illegible] patents details of construction of a billet-heating furnace of the type comprising an air-heating oven by which the air fed to the heating chamber as a constituent part of the gaseous fuel is heated. The feature of the construction is the provision of a supplementary flue between the heating chamber and the stack, independently of the air heating oven, and the provision of suitable dampers for controlling the products of combustion.

F. B. Parkison (779,307, January 3), patents a process of manufacturing mineral wool. Ordinarily, mineral wool is produced by directing a strong blast of air or steam through molten slag as it issues from the cupola to blow the molten mass through a blow-tube and into a blow-chamber or receiver. The material settles in the blow-chamber in the form of a mass of fine fiber interspersed with considerable quantities of minute globules or beads, the presence of which is due to the imperfect conversion of the slag into a fibrous form. The inventor has found that the bead-like particles are produced by the cooling of the slag before the latter is completely converted into fibrous form, the minute beads solidifying at the ends of the filaments. He also found that by subjecting the material during its flight to the action of hot vapor, preferably smoke or a combination of smoke and steam, the fibers are softened or annealed, materially lengthened, and given additional tensile strength, with the result that the body of the wool when taken from the settling chamber is of a soft coherent texture, and is almost wholly devoid of beady particles.

F. C. Weber (778,345, December 27), subjects the mixture of powdered aluminium and an oxide, preparatory to the aluminothermic reaction, to a drying, dehydrating and briquetting process.

A. Custodis (778,846, January 3), patents a process of making coke, which consists in mixing coal with top dust of blast furnaces containing iron, lime, silicic acid, and alumina, grinding the mixture, compressing it into cakes, and coking the cakes, whereby a slag is formed and the iron is reduced to form a sponge.

SYNOPSIS OF PERIODICAL LITERATURE.

A Summary of Articles Appearing in American and Foreign Periodicals.

INDUSTRIAL APPLICATIONS OF ELECTROCHEMISTRY.

Lead Refining.—In our Vol. I., page 407, we published a long illustrated description, by Anson G. Betts, on his process of electrolytic lead refining; some notes on recent developments were given in our Vol. II., page 509. The January issue of ***Mines and Minerals*** contains an illustrated description, by R. L. Whithead, of the plant at Trail, B. C., as remodeled in 1903 by the author. Up to that time 28 tanks had been used; then forty-four new tanks were installed, differently arranged and with different electrical connections. The electrolyte contained 10 per cent free hydrofluosilicic acid, with 5 per cent of lead in solution as lead fluosilicate. The potential between the electrodes was 0.25 volt. The anodes were cast 1 inch thick and placed 4⅛ inches from center to center, each tank containing 20 anodes and 21 cathodes. The cathodes, 1-16 inch thick, were obtained by a deposit of 30 hours on a steel plate, lead-coated and paraffined. For making starting sheets a current density of 4 amperes per square foot of cathode surface was used, while the current density on the working tanks was 15 amperes per square foot. The tanks are 7 feet 2 inches long, 2 feet 6 inches wide, 3 feet 6 inches deep, inside measure, and are made of 2-inch selected fir, painted inside and out with P. & B. acid-proof paint. Circulation is obtained through 1½-inch hard rubber pipes from one tank to another, as is customary in the cascade system. The arrangement of the original twenty-eight tanks was in two rows of fourteen each, two tanks being placed side by side, connected by a small copper frame, the current entering the tank on a bar of copper, 4 inch x 1¼ inches, and leaving the second tank by the same size bar, as shown in Fig. 1, where *a* are the anodes, *c* the cathodes, *w* the tank wall and *i* wood insulators. The new forty-four tanks were arranged side by side, twenty-two in a row, each tank being connected with the tank ahead by thin sheet copper, 1-32 inch thick. The current, of course, enters on heavy copper and leaves the tank on same size copper as shown in Fig. 2, where *a* are the anodes, *c* the cathodes, *w* tank walls, *d* copper conductor, *e* copper between tanks 8 inches x 1-32 inch, *i* insulation. The circulation being through two tanks, instead of through fourteen, as in the arrangement of the twenty-eight tanks, the advantage gained here was the use of about 60 per cent less copper than in the old system, and gave a more uniform circulation. A rapid circulation of the electrolyte was found to be best, since it gave a more uniform percentage of lead in solution. It was found that as soon as the lead content at the bottom of the tank differed more than 1 per cent from the top, the deposit would begin to tree, thereby causing a short circuit.

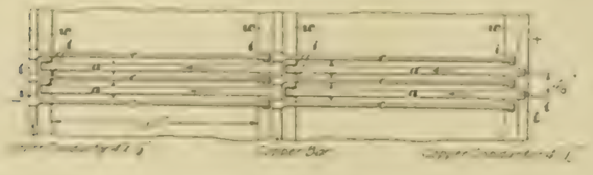


FIG. 1.—CONSTRUCTION OF ORIGINAL TANKS.

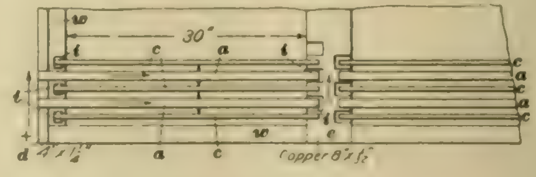


FIG. 2.—CONSTRUCTION OF NEW TANKS.

The author then describes the melting house. The anodes weigh each 300 pounds and require six days before deposition is complete, the amount of scrap obtained being 22 per cent of the total weight of anode lead. By casting the anodes in closed molds and inserting copper hooks in the molten lead, so that they are supported by cross-bars instead of cast lugs, the amount of scrap can be cut down to 10 per cent. The cathodes weigh, when taken out of the tanks, from 150 to 200 pounds, and each plate is washed separately with hand brushes. During the sweating down of the cathodes in the refining lead kettle, about 4 per cent of dross is formed, which contains the antimony and tin that has been deposited; to insure the complete removal of both, the heat is raised until the lead shows red on the surface, then follows poling for 30 minutes, during which time the metal is constantly agitated and exposed to air, whereby any antimony or tin that has escaped the sweating process is removed, with the large amount of lead oxide formed.

Of special interest is the treatment of the slimes, and it seems that a perfectly satisfactory process has not yet been worked out. The slimes which adhere to the scrap of the anodes are removed from the tanks by lifts; the slimes are then washed off with stiff brushes. As the amount of lead fluosilicate absorbed by the slimes during electrolysis is quite large, the percentage of lead and acid in the electrolyte is decreased from day to day, and necessitates that free acid and white lead be added each day to the electrolyte in order to maintain a uniform percentage of each ingredient. It was also necessary to add glue, about 2 pounds every other day, to each 2000 cubic feet of electrolyte; this prevents treeing and short-circuiting, and also gives a beautiful crystalline deposit, as shown in Fig. 3. After the slimes have been washed from the scrap, they are allowed to settle, and this first wash-water is run back into the electrolyte. To free the slimes from fluosilicic acid and lead fluosilicate, they are washed three times by decantation, with boiling water; after that, the slimes still contain 1 per cent of fluosilicic acid. The dilute solution of wash-water is concentrated by evaporation until it reaches a strength of 30° Be, giving by analysis 16 per cent metallic lead, and 8 per cent free acid; it is then run into a settling tank, cooled, and during the next twenty-four hours returned into the electrolyte in small amounts, so that it becomes thoroughly mixed. During electrolysis lead oxide is readily formed at the anode, and as readily goes in solution with the free fluosilic acid present, forming lead fluosilicate. A large amount of this salt is absorbed by the slimes, adhering to the anode, and does not enter the body of the solution, thereby depleting the lead content of the electrolyte. Another cause of further depletion of the lead content is the large amount of antimony in the slimes, from 25 to 30 per cent dry weight, only about half being soluble as oxide of antimony, Sb_2O_3, the remainder being combined with lead. This causes a further loss of lead in the slimes, and for each 10 tons of lead refined about 100 pounds are lost; this, in Canada, amounts to about 25 cents per ton. The author then discusses the characteristic advantages of the Betts process, which we have already repeatedly discussed. He finally discusses in great detail the methods now used for the refining of the slimes; this reference may suffice, since the author remarks himself that these methods are still incomplete, for they do not recover the impurities completely. He is now perfecting a process "in which electricity plays a prominent part," and which is intended to solve the question of melting silver in the presence of arsenic or antimony.

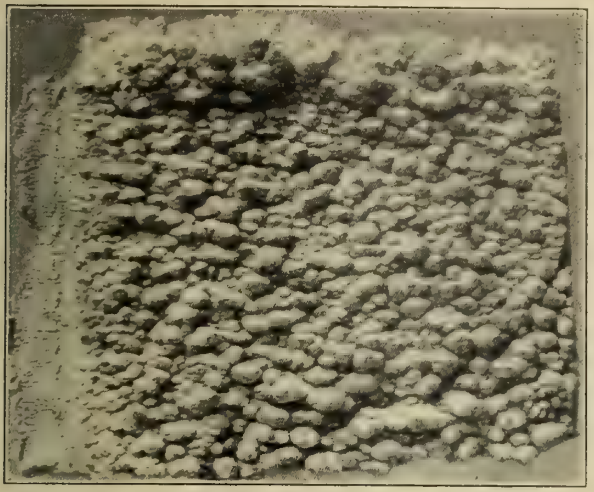


FIG. 3.—ELECTROLYTIC LEAD.

Calcium Cyanamide as Fertilizer.—The *Elektrotech. Zeit.* of December 29 contains some notes by Erlwein of the Siemens & Halske Co. on the production of calcium cyanamide, $CaCN_2$. The product is stated to be made a "rather (ziemlich) large" commercial scale in a plant in Germany, while a large plant for producing 4000 tons per year is being erected in Piano d'Orta in Italy. This product may be used as a cheap substitute for Chili saltpeter, and is stated to have proven satisfactory in extended fertilizer experiments made for three years by Wagner and Gerlach. There are two methods for making calcium cyanamide, the one starting from calcium carbide, invented by Moissan, Frank, Caro, Rothe, Pfleger (see our Vol. I., page 423), the other starting from carbon, lime and atmospheric nitrogen, invented by Erlwein (see our Vol. II., page 204). The German trade name of calcium cyanamide is Kalkstickstoff.

Ferro-Alloys.—In our Vol. II., page 309, we gave an illustrated description of the electric crucible furnace of P. Girod for making ferro-alloys and special steels. In an article by Schmidhammer in the *Oest. Zeitschr. f. Berg-u. Hüttenwesen*, of Nov. 12, some analyses of ferro-alloys made by Girod are given as follows: Ferro-chromium for fine steels: 67.27 per cent Cr, 31.82 Fe, 0.41 C, 0.17 Si, 0.11 Mn, 0.18 Mg, 0.007 S, 0.003 P. Ferro-chromium for armor plates: 67.10 Cr, 26.81 Fe, 4.2 C, 0.61 Si, 0.47 Mn, 0.23 Al, 0.17 Ca, 0.31 Mg, 0.01 S, 0.02 P. Ferro-tungsten: 85.47 W, 13.94 Fe, 0.35 C, 0.13 Si, 0.09 Mn, 0.005 S, 0.009 P. Ferro-molybdenum: 80.80 Mo, 16.79 Fe, 2.27 C, 0.11 Si, 0.02 S, 0.007 P. Ferro-vanadium: 49.50 V, 49.15 Fe, 1.07 C, 0.09 Si, 0.07 Mn, 0.10 Ca, 0.009 S, no P.

Chlorination versus Cyanidation.—The discussion started by W. E. Greenawalt's articles, abstracted in our last issue, is continued in the correspondence columns of the *Eng. and Min. Journal*, December 29. P. Argall writes: "The substitution of chlorine gas, or even chlorine water, in place of chlorine liberated in a nascent state, directly in the barrel, is certainly nothing new, nor can the electrolytic generation of chlorine be considered new, even when generated in the barrel. One of Cassel's numerous patented processes takes in that feature, while the Greenwood and other processes cover about all that is left, and, so far as I know, they are all commercial failures. The electrolytic decomposition of salt outside the barrel as a means of providing the chlorine for the charge, has been in use in the largest plant of the 'mill combine' for about a year; but it has not been, up to this time, introduced into the other mills of the combination, so I infer it is not good enough." He then defends the cyanide process against various objections raised against it, and reiterates his statement that the chlorination process "cannot begin to compete with modern cyanide practice. One of the simplest proofs is that the tailing discharged from any of the chlorination plants, after being subjected to the double process of chlorination and concentration, will pay handsomely to cyanide." William H. Davis refers in the same issue to a case in which an electrolytic chlorination process mill was dismantled and refitted for cyanide; "the electrolytic chlorination process, though preceded by roasting, had failed to extract the gold from ore which was a simple telluride." He then contradicts the view that neither the acid nor the bleach has any indispensable efficacy in the chlorination process, other than to deliver chlorine; he thinks that this view "overlooks the difference between molecular and nascent chlorine."

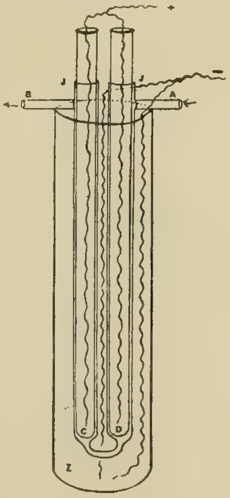


FIG. 4.—OZONIZER.

Ozone.—To the London *Elec. Review*, of December 30, J. B. C. Kershaw contributes another article on the production and utilization of ozone. He first describes a modified form of the Berthelot tube ozonizer, now being employed by Bone and Drugman in research work at the Victoria University in Manchester. This consists of a U-tube formed of two thin walled glass tubes, one inside the other, their surfaces nearly touching, and sealed together at *J*, as shown in Fig. 4. The outer U-tube has branches at *A* and *B*, for the inlet of air, and exit of the ozonized product. The conducting medium is a slightly acid solution of copper sulphate, the inner tubes, *C* and *D*, and the outer thick glass containing vessel *E* being nearly filled with this solution. A stout copper wire passes down each of these tubes, and these wires are connected to the positive and negative poles of the induction coil or transformer supplying the current. The air passes along the very narrow annular space between the inner and outer tubes. The advantages of a liquid contact in place of metal are, that there is a more regular and silent discharge of the electric current in the space between the two glass tubes, and that the temperature of the apparatus can more easily be kept under control, a flow of the sulphate solution through the inner and outer tubes being very simple to arrange, if desired. Any number of single ozonizer tubes can also be grouped together, and for experimental work this form of ozonizer is said to be superior to any that has been yet tried. For industrial purposes, its fragile construction is, of course, a serious defect, and would probably lead to trouble. The author then describes the ozonizers of Elworthy and Vosmaer-Bebret, which were described in our Vol. II., pages 199, 153, 511, and then refers to the researches of Goldstein, who has recently discovered that it is the ultraviolet rays of light which-are-chiefly-instrumental in the formation of ozone. (This has been confirmed by others.—Ed.) Goldstein has designed a quartz-glass tube ozonizer which is said to utilize these ultraviolet rays to the fullest advantage. Two platinum electrodes are fixed in the ends of this tube, a vacuum is produced in it, and upon allowing the electric discharge to occur between the two electrodes, the tube becomes filled with the usual colored haze. Only the ultra-violet rays of light, however, can pass easily through the quartz-glass envelope which surrounds them. If the tube be now placed in an atmosphere of pure oxygen, cooled below 100°, almost the whole of the gas is transformed into ozone, and this condenses upon the outer walls of the quartz glass tube, as a blue liquid. Not only is the action of the silent discharge stated to be much intensified under these conditions, but the liquid ozone is said to be more stable than gaseous ozone, and less liable to be decomposed by the rays which produce it. The reaction, in fact, ceases to be a reversible one at — 100° C. The author then refers to Rosenberg's ozonizer noticed in our last issue, page 36, and gives the following table, which brings together all the hitherto published figures for the yields of the various forms of ozonizer, with the calculated cost of electrical energy, when the e. h. p.-hour is supplied at 3 cents:

COMPARATIVE YIELDS AND COSTS OF OZONE.

Form of Ozonizer	Yield of Ozone in Grams per E.H.P. Hour.	Cost of Electrical Energy in Cents.	
		P'r Kilogramme of Ozone.	P'r Kilogramme of Active O.
Rosenberg.......... ..	184	16.3	48.9
Yarnold............ .	175	17.1	51.4
Otto...........	155	19.3	58.0
Andreoli	91	31.8	95.4
Elworthy............	48	62.4	187.2
Siemens & Halske.....	25	120.0	360.0
Marmier & Abraham.	20	150.0	450.0

The lowest figure in this list for the cost of the kilogram of active oxygen is still "greatly in excess of the cost by the usual oxidizing and disinfecting agent, namely, hypochlorite of lime (bleaching powder or bleach), namely, between 28 and 34 cents." It must also be remembered that the figures in the last two columns in the above table, give the cost of electrical energy only. The author then begins to make some remarks on the utilization of ozone for water sterilization, referring to the Marmier-Abraham system.

[illegible]

Electro-Analysis with Circulation of the Electrolyte.—The *Zeit. f. Elektrochemie*, of December 23, contains an account of [illegible] and R. J. Boddaert on electro-analysis with strong circulation of the electrolyte, produced, for instance, by rotation of the electrodes, in order to hasten the deposition. They experimented with nickel, zinc, copper, bismuth, cadmium, lead, silver, mercury, antimony, and tin. A perfect precipitate of nickel is obtained with rotating electrodes from an ammonium oxalate solution, the time of analysis being reduced from three hours to fifty minutes, with a hot solution, and to forty minutes with a cold solution. The method cannot be applied to Classen's oxalate method for zinc precipitation, since spongy zinc would be obtained; good results are, however, obtained with sodium acetate and acetic acid; the precipitate becomes best, and the result most exact with an alkaline solution with an addition of potassium cyanide, a nickel cathode being used. Copper is precipitated satisfactorily at a high rate from a nitrate solution with or without sulphuric acid; for precipitating the last quantities of copper, an addition of ammonia is necessary; the time is reduced from six hours to twenty minutes. With respect to bismuth, the authors found that strong stirring accelerates the precipitation from a nitrate solution and diminishes the formation of sponge without preventing it altogether; for this reason an exact analysis by this method is not yet possible. With cadmium, an oxalate solution is not suitable for precipitation at a high rate, since an increase of current density results in a spongy deposit; sulphate solution is more suitable, although the formation of sponge cannot be avoided altogether; in the latter case the time of analysis was ten minutes. By stirring with increased current density, the time of precipitating lead from a nitrate solution was reduced from one hour to fifteen minutes, the result being equally satisfactory. Strong circulation accelerates greatly the precipitation of silver from a potassium cyanide solution, but the quality of the silver is thereby impaired. Precipitating with stirring gives good results for mercury in a nitrate solution at ordinary temperature, and with not too high a current density; the time was thereby reduced from forty-five to fifteen minutes. In general, if the best methods are compared for both cases, the time required for precipitation of the metals investigated, with stirring and revolving cathodes, is four to eighteen times smaller than without stirring.

Electrolysis of Cobalt and Nickel Tartrates.—An account of experiments on this subject is given by J. E. Root in *Jour. Phys. Chem.*, January. In determining the apparent decomposition voltages under the conditions used in electrochemical analysis, the value of 1.75 volts was found for cobalt in an alkaline tartrate solution and 2.8 volts for nickel. However, the range of voltage in separating cobalt from nickel is not as great, since, in the presence of cobalt, the decomposition voltage for nickel in an alkaline tartrate solution is lowered. If the voltage be kept below 2.1 volts in separating cobalt from nickel, the metallic cobalt deposit contains no nickel; at higher voltages the cathode deposit always tests for nickel. While pure cobalt is obtained at voltages below 2.1 volts, it has proved impracticable to get all of the cobalt out of the solution in any reasonable time. One reason for this is the oxidation of cobaltous salt to a green cobaltic salt at the anode. Coehn's results as to the composition of cobaltic oxide, $Co_2O_3.3H_2O$, are confirmed by the author. The practical results may be summed up in the statement that the author found several methods for the electrolytic determination of cobalt alone, and that it is possible to obtain nickel and cobalt pure from an alkaline tartrate solution if one is willing to take the time and pains.

Calcium.—J. H. Goodwin, who has already published experiments on the electrolytic production of calcium (*Jour. Am. Chem. Soc.*, August, 1903) has recently presented another paper before the Amer. Philos. Soc'y., which is published in the *Proceedings*, Vol. XLIII., No. 178. The experiments were made at the laboratory of the University of Pennsylvania, and the method used was the electrolysis of fused calcium chloride. The furnace is shown in Fig. 5, where a bottom of solid calcium chloride was maintained by the cooling effect of a copper coil *E*, through which is a thick piece of asbestos *C*, holding by the blocks *D*, the copper coil *E* well up in the Acheson graphite anode *F* and insulated from it by the asbestos *G*. The iron bands *H* conduct the current from the positive cable *I* to the graphite vessel *F*. The calcium grows and forms the stick *J*, which is started by the cathode *K*, connected with the negative cable *L* and supported by the clamp *M*, which is drilled and tapped at *N* to receive the screw *O* by which it can be raised or lowered. *P* is a tube sliding freely on the rod *Q* of the retort stand, and against which *R* is firmly screwed to make the clamp *M* fairly rigid without interfering with its vertical motion. Fluospar covers the copper coil and fills the space about it, while the furnace is filled with calcium chloride, which becomes solid at *S*, and is kept molten at *T* solely by the current passing through the furnace. The whole apparatus was set up inside an empty wind furnace, from which the grate bars had been removed. In this way the escaping chlorine was drawn from the room. Pure anhydrous calcium chloride was used, melted in a Dixon graphite crucible and added from time to time, but this was found to introduce much iron, aluminium and silicon from the clay binding used in the crucible, so that finally the chloride was added cold and melted by drawing an arc from the iron rod *K*. Thus, the furnace was filled sufficiently for a run. The anode was turned in a lathe from a 6-inch length of Acheson graphite electrode six inches in diameter. Being pure and able to withstand the high temperature and chlorine without disintegration, made this material by far the most suitable for constructing the furnace. In two experiments, the author succeeded in obtaining an ampere-hour efficiency of nearly 42 per cent, while the average ampere-hour efficiency was 29.1 per cent; the average volts were 17.7.

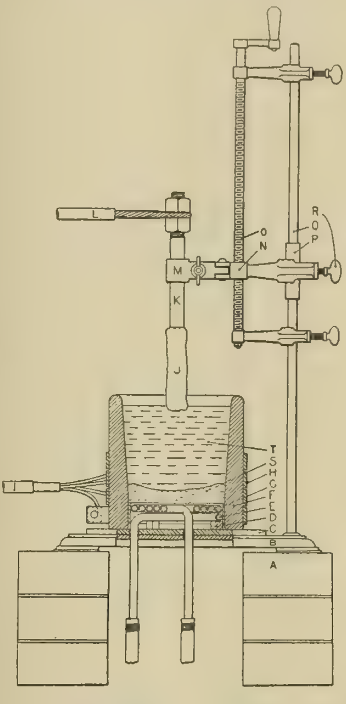


FIG. 5.—ELECTRIC FURNACE FOR PRODUCING CALCIUM.

The analysis of the calcium produced is Si 0.03 per cent, Fe 0.02, Al 0.03, Ca 98.00, Mg 0.11, Cl 0.90, O (by difference) 0.91. The author thinks that the difficulty experienced, until recently, in making metallic calcium, was probably due to the small scale on which the operation was tried. The simple and satisfactory operation of this furnace would lead one to believe that, technically, the process would be still more efficient and easily controlled. A furnace, five times as large, using about 1200 amperes, would require about 8 volts, and the screw mechanism could be electrically controlled, keeping the current constant and the product perfectly uniform, as the rotary furnaces of the Union Carbide Co. are controlled. A water-cooled shield might be necessary to cool the large calcium cathode as it was drawn from the bath. The two essential conditions of operation are—first, rapid withdrawal of the metal formed to increase the yield and minimize recombination; second, narrow temperature limits. The bath must be hot enough to deposit the metal molten, not spongy, and cool enough to let it congeal upon the cathode and be raised without breaking off.

"The solid metal can be worked like other metals and is much more stable than imagined." It can be heated red-hot continuously in a triple Bunsen flame without igniting, but at this temperature its texture is like clay, and it can be easily squeezed apart with tongs, sometimes igniting at the edges and burning feebly till the lime formed smothers the flame. When cold, a bright calcium surface becomes dull rapidly in ordinary air, but if hot the metal can be brightened with a file or polished in the lathe with emery cloth and will remain bright as long as it is hot. The specific gravity is 1.5446 at 29.2° C., so that it is only four-sevenths as heavy as the light metal aluminium (specific gravity 2.68). Calcium is the fifth best conductor of electricity, being surpassed by silver, copper, gold and aluminium, if wires of equal diameter and length are compared; but for wires of equal weight and length the order is entirely different, calcium being second and exceeding silver by 67 per cent, copper by 62 per cent, gold by 86 per cent, and aluminium by almost 20 per cent. Calcium is harder than sodium, lead or tin, almost as hard as aluminium, but softer than zinc, cadmium or magnesium. The ultimate tensile strength of calcium is 8,710 pounds per square inch. A similar method of making calcium is understood to be used on a commercial scale at the Elektrochemische Werke of Bitterfeld in Germany (see, for instance, the paper of Rathenau, Electrochemical Industry, Vol. II., page 276). The metallic calcium is sold in form of sticks and is said to be consumed mostly in the iron and steel industries and for reduction purposes.

Action of Amalgams on Solutions.—The controversy between G. McP. Smith and G. Fernekes on the action of amalgams on various aqueous solutions and on the possibility of explaining the facts by the electrolytic dissociation theory is continued in the *Jour. Phys. Chem.*, G. McP. Smith contributing an article on the action of barium amalgam on solutions of sodium and potassium salts to the January issue. The author, who defends the dissociation theory, describes some new experimental facts and discusses at some length some points of the theory. "While all the phenomena of solutions may not be capable of explanation by the ionic theory, it is at present, nevertheless, very rash to claim it to have been shown 'conclusively' that this theory has outlived its usefulness."

Thermochemistry.—L. J. Henderson publishes a paper in the January issue of the *Jour. Phys. Chem.* on the heats of combustion of atoms and molecules. He shows that the heat of reaction of a substitution depends not alone upon the nature of the atoms or atom groups which take part directly in the reaction, but also in a high degree upon the chemical composition of the remainder of the molecule, and upon the arrangement of the atoms therein. Thus, the same substitution —H by OH—which leads from paraffine hydrocarbon to primary alcohol, with a loss of 40 calories to the heat of combustion of the molecule when it converts an aldehyde into an acid diminishes the heat of combustion by 72 calories. On the other hand, marked and parallel differences exist in the heat of formation from the hydrocarbons of primary, secondary, and tertiary alcohols, and of aldehydes and ketones. The heat of combustion of an atom chemically bound in a molecule is dependent not only upon its nature and the nature of the atoms with which it is directly united, but also upon the nature and position of every other atom of the molecule. If there are not present within the molecule considerable energy relationships of unknown character, the energy of a valence is variable, and a function of all the atoms of the molecule and of their positions.

METALLURGY.

IRON AND STEEL.

Fine Ores.—A standing order of the day, in the production of pig iron, is the discussion of obviating the difficulties caused by using fine ores, such as the Mesabi. It is probably the most serious question at present confronting American blast-furnace practice. Engineer Aloys Weiskopf, of Han-

[illegible] discusses the question in a recent issue of *Stahl und Eisen*, November 1. He reviews the question of chemical equilibrium between Fe_2O_3, Fe_3O_4, FeO, Fe, C, CO and CO_2, and finds the more serious cause of the difficulties of smelting fine ores to be in their very easy reducibility, by which they are reduced high up in the furnace, and the spongy iron thus produced has a large range of action during which it acts as a catalytic agent to cause carbon di-oxide to deposit carbon, while the finely deposited carbon clogs up the gas passages and causes "hanging" of the charge and explosions from accumulation of pressure and subsequent rapid oxidation of the fine carbon.

Desperate attempts have been made to overcome the difficulties by changing the lines of the furnace and the manner of charging, but the author thinks that the most hopeful solution lies in practicable briquetting processes. The sintering process is being operated at a cost of $0.75 per ton, in Germany. Very cheap fuel is gasified in gas producers, and the fine ore heated in furnaces to the point where it becomes sticky; this temperature is lowest for spathic ore, higher for purple ore, and highest for magnetite. The hot ore is then pressed at this temperature into briquettes, which are hard, strong and porous, while also free from water, sulphur, arsenic or carbonic acid, since the heating may be conducted as an oxdizing roasting. The process gives very good results, but requires careful supervision.

The slag-briquetting process consists in mixing 10 per cent of finely ground iron slag with the ore, making up briquettes and keeping them in a steam chamber for ten hours, at 120 pounds pressure per square inch. The action is a hydrochemical one, the slag being partly combined with the ore, under the influence of heat and moisture, and so causing a setting into a hard, porous briquette. The steam washes out some injurious ingredients, such as sulphur, but the slag added increases the work which the furnace must do in smelting the ore. This solution is not yet altogether satisfactory.

The Tunner Statue.—Of interest to all metallurgists, and particularly to iron and steel workers, is the unveiling of the beautiful statue to Peter von Tunner, at his beloved Bergschule in Leoben, Austria, on November 20. Numerious officials, dignitaries, colleagues and a host of students, miners and townspeople participated in the occasion, while the successor of Tunner in the chair of metallurgy at Leoben, Professor von Ehrenwerth, delivered a masterly oration. The statue consists of a granite pedestal and a bronze bust of Tunner, with life-size figures of a student on one side and a miner on the other, entwining a wreath about it.

Born in 1809, Tunner traveled on metallurgical journeys throughout most of Europe, in 1835 to 1838, and was named professor on the opening of the Mining Academy at Leoben in 1840. Here he taught, worked and lived until his death in 1897, at the age of eighty-eighty years. His best-known scientific works were his classical researches on the finery process, the puddling process, and the working of the blast-furnace. In thus honoring the memory of her celebrated metallurgist, his native land does honor to herself.

Verein deutscher Eisenhüttenleute.—The general meeting of this well-known German society of iron and steel metallurgists was held in Dusseldorf on December 4. The papers read were "On Large Gas Engines," by Prof. Eugene Meyer, of Berlin; "On Drying Blast for Furnaces by Refrigerating Machines," by Professor Carl von Linde, of Munich, and "On the Classification of Foundry Pig Iron," by Dr. F. Wust, of Aachen. We will reprint these papers as soon as they reach us in full

Dry Blast.—The unexpectedly valuable results attained by Mr. Gayley when using dried blast in a blast-furnace (see report of proceedings Iron and Steel Institute, ELECTROCHEMICAL INDUSTRY, December) have fairly stirred up metallurgists in every direction to account for them by proper calculations. The point of this whole question is, that theory, so far as developed and understood, would account for a saving of only 2.2 per cent of the fuel by removing the moisture of the blast, whereas Mr. Gayley actually saved over 20 per cent. Professor Le Chatelier discusses this question in the *Revue de Metallurgie*, for December, and rather plainly intimates that some other conditions, not described in Mr. Gayley's report, must have caused the saving, or else that the results were deliberately falsified as a stock-jobbing scheme. The puerility of these objections would render them undeserving of notice, if they come from a less distinguished scientist; it is a pity that the critic could not have preserved a more judicial frame of mind. W. Schmidhammer, in *Stahl und Eisen*, for December 1, attacks the same subject, but in a much more judicial and judicious manner, and, in fact, does really come near to finding the correct solution of the enigma. He bases his calculation on the theoretical temperatures obtainable before the tuyeres, using dry blast and moist blast, and calculates them as 2465° C. and 2294° C., respectively. The calculations are not free from errors, but they show what is undoubtedly true, a much higher theoretical temperature before the tuyeres when using dry blast. This higher temperature, Schmidhammer argues, causes the furnace to work faster, extends the smelting zone, and so improves the efficiency of the furnaces as to account for the saving in fuel obtained. Schmidhammer is miles nearer to the correct solution of this question than is Prof. Le Chatelier, with his unwarranted insinuations, as is evident from the logic of Schmidhammer's remarks, and as will be abundantely proven by developments in the near future. By the latter we allude to a forthcoming discussion of this whole matter at the next meeting of the American Institute of Mining Engineers, at which it will be shown just what causes the improved working with dried blast, and where it will be shown that Schmidhammer has reached conclusions which are about half right.

Quenching Tests.—Le Chatelier has made a series of very interesting tests showing the rate of cooling of pieces of steel in various liquids and under varying conditions, with the resultant effects upon the hardness of the specimens. (*Revue de Metallurgie*, September.) The temperatures were taken by a thermo-couple embedded in the center of the specimens, and recorded autographically by means of a reflecting galvanometer and a movable photographic plate. The variations in rate of cooling were considerable; in 8 seconds after immersion, the temperatures fell from the 800° to 900° C. at starting to 40° C., when sprayed with cold water, 100° when dipped into water at 20°, 115° when dipped into cold brine, 160° in water at 35°, 190° in water at 50°, 190° in 10 per cent sulphuric acid, 460° in mercury, 550° in linseed oil, 670° in boiling water, and 725° in melted lead. The speed of cooling varies greatly with the temperature of the cooling fluid. Cooling in still air or in an air blast showed great variations; while a specimen cooled in cold water fell to 100° in 5 seconds, in boiling water to 320° in 5 seconds, and in oil to 320° in 17 seconds, or to 100° in 45 seconds, the pieces cooled in air remained at 700° at the end of 60 seconds, rising to 750° after that, by the effect of heat evolution at the allotropic critical point.

One practical result of these tests is to show the similarity of the cooling curves in oil and in hot water, showing the possibility of obtaining similar effects, which has already been proved in practice. The results obtained show the relative rates of cooling for bars of the size taken, viz., 18 millimeters square, and in these there is a certain relation between the rate at which the piece can conduct its internal store of heat to the surface and the heat-absorbing capacity of the surrounding fluid, the latter factor consisting of a combination of the specific heat of the surrounding fluid and its rate of heat conduction. For smaller objects, with the cooling surface relatively much smaller, the conductivity of the surrounding fluid plays a relatively much greater *rôle*, and the relative values of the fluids as hardening media are different from those found by Le Chatelier. For small, fine steel objects, mercury is known to be a much greater hardener than water, which is the converse of Le Chatelier's results, and this is undoubtedly due to the greater influence exerted in this case by the larger heat-conducting power of mercury.

Copper, Brass and Bronzes.

A Swede, Axel Westman, made a "metallurgical journey" into Germany, and his report to the *Jornkontorets Annaler*, translated into German in *Metallurgie*, of December 8, 1904, contains, among other things, the following information:

The Kupferwerke Deutschland, at *Oberschönweida near Berlin*, was erected in 1897, and is provided throughout with modern machinery. The copper used for copper ware is exclusively American electrolytically refined; for brass, Mansfeld copper and Silesian zinc is used. In the casting house, the crucible furnaces are sunk level with the ground, and the crucibles used carry up to 100 kilograms (220 pounds) of metal; coke is used as fuel, and blast is introduced by means of small nozzles just above the grates. The copper is first melted, then the scrap brass added, and finally the zinc, slightly warmed with vigorous stirring. The loss of copper is 0.5 per cent, and of zinc 6 per cent. The brass is cast into round ingots 3 inches in diameter by 36 inches long, in thick-walled molds, so as to chill the metal and make it stronger; it has been found that when using thin-walled molds, in which the metal set slowly, copper tends to sink to the bottom and zinc to accumulate on top. One article made on a very large scale at this works was rods of drawn brass, drawn with a section like the profile of a gear wheel. These rods are bored in the center and then cut transversely into thin sections, forming small toothed wheels for clocks and watches. Copper rods are rolled into forms with complicated cross-sections, which, when cut into sections in a similar manner form the copper blocks needed for commutator bearings. In casting brass into chill molds, a sort of sieve or filter filled with charcoal is put on top of the mold, which strains out dirt, oxide and other impurities, and divides up the falling metal into small streams. In rolling out sheet brass, the rolls become worn away more in the middle than on the edges. They are brought back to proper condition by screwing the rolls hard down together, and running them empty with a plentiful supply of water; they thus grind each other straight.

The Allgemeine Elektricitäts Gesellschaft has a large, newly erected copper and brass works at *Oberspree, Oberschönweide, near Berlin.* The casting house contains, in a row, thirty sunken crucible furnaces, and with fire-tipping crucible furnaces, in which the crucibles, holding up to 650 pounds, remain permanently until they are used up, after about 100 heats. When pouring ingots into chill moulds, it is usual to wash the mold with a wash of finely ground wood ashes. In this works, aluminium bronze containing 10 per cent aluminium was being manufactured, and cast into cylinders of 2 inches diameter, by 36 inches long. The copper used is the purest electrolytically refined, which is first melted, and then aluminium stirred in, resulting in a lively boiling, and the raising of the contents to nearly a white heat, a phenomenon ascribed by the author to formation of aluminide of copper [but which is in reality mostly due to the oxidation of some aluminium by dissolved cuprous oxide.—*Abstractor*]. The bronze thus made is so brittle that it cannot be worked either hot or cold, and it is only malleable after being remelted two or three times. It is general experience that the aluminium bronze is better the oftener it is remelted. The final casting is made in cast-iron molds, smeared with a mixture of graphite, pipe-clay and lard; and greatest care is taken that no skin of oxide runs into the mold. The waste is about 5 per cent. Finally, the aluminium bronze rod is placed in another mold and well deoxidized copper cast around it; the compound bar is rolled and drawn down to wire, which is sold as "Doppelbronzedraht" or "double bronze wire," used extensively in Europe for telegraph and telephone service, the bronze core giving great strength and the copper envelope high electric conductivity.

Phosphor bronze is made in this establishment by melting together a mixture of 93 per cent electrolytically-refined copper, 2 to 3 per cent of 10 per cent phosphor copper, and 4 to 5 per cent of tin; practically no phosphorus remains in the product. The melt is stirred with a copper rod well rubbed with powdered graphite, and cast into slabs for rolling. These slabs are rolled down to nearly circular discs of 3 millimeters thickness, by passing through the rolls in different directions; the nearly circular sheets are put into a circular cutter, which, starting from the outside cuts off a continuous strip 3 millimeters wide, forming, when straightened, a continuous rod 3 millimeters square, leaving a small circular waste in the middle. These strips are then rolled down to wire, which, after annealing, can be drawn down to the finer sizes. This phosphorbronze wire has an elastic limit which practically coincides with its ultimate tensile strength, and does not crystallize or weaken under rapid agitations of load. With these properties it makes a splendid material for wire rope used in mines, and particularly because it resists corrosion better than iron or steel.

The same firm rolls and draws down American copper wire bars to pure copper wire of 98 per cent guaranteed conductivity, using 21 heavy American drawing benches, with roller draw plates, which are described as the most perfect wire-drawing machines made. The finer wire, from 1.4 down to 0.05 millimeter diameter, is drawn through diamond dies, the larger sizes necessitating a 5-karat diamond for one die; the life of these dies is 6 months to a year. The hard-drawn trolley wire, of 8 millimeters diameter, made here has a tensile strength of 60,000 pounds per square inch and conductivity of 98.45 per cent.

The *"Letmather Messingwerk" at Letmathe* makes brass sheet and wire. There, brass is cast into slabs in sand molds, using 94 to 96 parts sand, mixed with 4 to 6 parts of washed clay, and made moist. The molds are dried in hot chambers above the crucible furnaces, which are kept hot by the waste heat of the latter. The brass contains 63 parts copper to 37 parts zinc, and is cast at as low a temperature as possible. The slabs are rolled to 5 millimeters thickness, the corners cut off, and then a spiral cut made which furnishes a long strip 5 millimeters square in cross-section, which is annealed, rolled and drawn.

Basse und Selve in Altena, Westphalia, use a crucible furnace of specially effective construction. A shaft of refractory material is surrounded by a jacket of double sheet iron, the space between the iron walls is filled with asbestos. Between this mantle and the fire clay shaft is a space into which air is pumped, with holes through the shaft to admit the air to the crucible space at four different levels and all around the crucible. The latter is packed in with coke, and stands on a pedestal of fire clay which has also air conduits to admit air around the base of the crucible. In this manner the fuel around the crucible is burnt very regularly and economically, it being found that only one-third as much fuel is used as in the old form of wind furnace; or, giving the actual figures, there were used for melting nickel, bronze or brass, per 100 pounds of metal melted, 85, 63 and 37 pounds of coke in the old-style wind furnaces, and 35, 19 and 11 pounds, respectively, in the new furnace. The latter is, moreover, portable, can be carried around the shop and the metal poured wherever neded, without removing the crucible.

The *Elmore Metall-Aktien Gesellschaft,* in *Schladern on the Sieg.,* make copper tubes electrolytically. The refined copper is granulated in water, and placed in a layer 16 inches deep on the bottom of a lead-lined tank, 19 feet long by 6 feet wide. The solution is copper sulphate with 3 per cent free sulphuric acid, and the horizontal rolls forming the cathode run in glass bearings at a distance of 1.2 inches above the copper. The agate burnishing roller moves over the depositing copper once for every 0.03 to 0.04 millimeters of increasing thickness. The current density is 200 amperes per square meter of immersed cathode surface, or 800 amperes per tank; the voltage 1 to 1.25 volts per tank; at which 4.4 millimeters thickness of copper is deposited per week. It is highly important that there be no interruption during the deposition, as this would produce immediately an oxide layer, which would prevent complete adhesion of the subsequently deposited copper. This plant makes 1500 metric tons of copper tubes yearly; the largest tube so far made is 2.5 meters diameter by 6 meters long. The mandrels used as cathodes are of easily fusible brown alloy, the composition of which is kept secret; they are covered with Stanniol before being used. After deposition is finished, they are put into a superheated steam chamber and the alloy melted out. The deposited copper shows 36,000 pounds tensile strength per square inch, with 15 to 16 per cent elongation; after annealing, 30,000 pounds, with 50 per cent elongation. The slimes remaining in the tanks contain gold and silver, and are a source of profit, being sold for $450 per ton.

The English Electrometallurgical Co., Ltd., at *Leeds*, also use the Elmore process, with a capacity of 100 tons per day. The main precipitating room is 200 by 250 feet, and contains 216 precipitating baths. The mandrels here are polished copper tubes, well rubbed with grahite to prevent adherence of the deposited copper. The raw material is Chili blister copper containing 95 to 96 per cent of copper, and is cast at the works into slabs some 17 feet long, to fit the bottom of the tanks. Tests of the copper here deposited show from 50,000 pounds tensile strength per square inch, with 6 per cent elongation to 30,000 pounds, with 50 per cent elongation.

Julius and August Erbslöh, in Barmen, manufacture polished, silvered and gilded copper plate, rods and sheet of brass and tembac, and of aluminium. The aluminium slabs are from Switzerland or America, and are 3 to 4 inches thick. They are rolled down cold. After being annealed and then cleaned in caustic soda solution they are trimmed by hand, to free them from particles of slag, flaws, etc., and then rolled down further, to sheet, without warming, but being several times cleaned and annealed. The annealing furnaces are muffle furnaces with cast-iron muffles.

Smith's Dock Company, Ltd., South Shields, cast brass and bronze, and make a specialty of copper blast-furnace tuyeres. For producing sound castings of copper, the purest copper obtainable is used, since very small quantities of iron, arsenic or sulphur are very injurious. The copper is also preserved as carefully as possible from oxidation, while melting down, for when cuprous oxide is formed it dissolves in the metal and renders the production of sound castings impossible. This firm adds 1 to 1½ per cent of tin or zinc to the copper thus being cast; lead is not usually added, except in cases where the casting has to stand the corrosive action of acid water; phosphorus is used to the amount of 0.5 per cent for castings, with very thin walls. Lately, 1 to 5 per cent of manganese has been used, with best results in such castings, using 30 per cent manganese-copper; such castings are dense, hard, tough and easily worked. The castings under discussion are cast in dried sand molds.

[The preceding abstracts from Herr Westman's account of his "metallurgical journey" demonstrate the value of the Swedish custom of sending out experts in various lines to examine as far as possible and report as extensively as is permitted, on the metallurgical industries of other nations. It is a most admirable custom, both for the "traveler" and for the "stay-at-homes," and one which might be copied with great advantage by our American universities, societies and large industrial concerns.—Editor.]

GOLD.

Cyanide Process.—The advantages of using filter presses are rapidly becoming recognized, and their use is being extended from the treatment of slimes, where they were first employed, to the treatment of fine sands; and it is even proposd by some enthusiasts in their use, to grind the whole of the ore to fine sand or slime and treat it all in the presses. C. Göpner, of Hamburg, writes interestingly on this subject in Borcher's *Metallurgie* for December 22.

According to Herr Göpner, he was the first to recognize the advantages of the filter press treatment, in 1892. The mines of Western Australia, just then opened, were having extraordinary difficulties with the amalgamation of their gold ore, the gold being so finely disseminated that in some cases, although containing over $100 worth of gold per ton, scarcely $10 per ton was extracted by amalgamating milling. The author obtained samples of this ore, had an experimental filter press built for him, and by agitating the very finely ground ore for two hours, with a 0.25 per cent cyanide solution, and filtering in the press, found that he extracted 96 per cent of the gold present, obtaining it in clear cyanide solution ready for precipitation.

The Hannans Brown Hill Gold Mining Co., of Kalgoorly were the first to put the idea into practical use, with a plant of 100 tons daily capacity. The extraction obtained was 82.7 per cent of the gold from the coarse sands, 88.5 per cent from the average sands, and 93.1 per cent from the finely ground flour.

The presses used have 50 frames, each 40 millimeters thick and 1 meter square. The cubic content of these frames is 1.656 cubic meters, corresponding to about 3.5 tons of ore at one treatment. Much trouble was found in getting suitable filter cloth, until, after extensive tests, a cotton-duck cloth from the United States, such as is used for sail-cloth, was found to answer all requirements, and is now used exclusively. Using air pressures of 50 to 75 pounds per square inch, the press can be filled in 7 minutes; the washing, emptying, cleaning and reclosing take two hours and thirty minuthe; nine batches are treated per day, when the workmen become skillful.

The above work was all done by first agitating the slimes in tanks with cyanide solution, and then filtering in the presses. Experiments were made in running the ore into the presses with water, and then washing with cyanide solution in the press itself, but 10 per cent less gold was extracted in this manner, and the original practice was reverted to. For the maximum extraction by filter-press treatment, it is recommended that the ore be pulverized in roller-mills. The filter press is not, however, universally applicable, each case of ore and slimes must be studied for itself, and the best method of treatment for it discovered and applied.

At the present time, 80 per cent of all the ore mined in the Kalgoorly fiield is being treated in filter presses, amounting to 70,000 tons of ore and 15,000 tons of accumulated tailings monthly and yielding over $3,000,000 monthly: The ore is either ground directly to fines, agitated with cyanide solution containing cyanogen bromide, and filtered; or it is amalgamated, concentrated, the concentrates roasted and treated with cyanide solution. In any case, the slimes are treated altogether by filter presses.

VANADIUM.

H. Herrenschmidt communicates to the *Comptes Rendus* of the French Academy an interesting paper on the extraction of vanadium (1904, Vol. 139, page 635), of which we abstract the following, from a translation in Borchers' *Metallurgie* (*December* 8). The process described is that used at the works of Bas-Coudray, near Le Genest (Mayenne). The ore is lead vanadate, from the Santa Marta Mine in Spain, and contains 12 to 14 per cent of vanadic oxide to about 50 per cent of lead. It is melted in a reverberatory furnace with carbonate of soda and coal, forming metallic lead, which collects also the silver in the ore, and a slag containing sodium vanadate, sodium aluminate and iron silicate. The slag is kept melted in another furnace, and air blown over it until the vanadium oxide is fully oxidized. The mass is then run into boiling water, granulated and leached. After repeated washings, the residuum does not contain over 2 per cent of vanadic acid; 95 per cent of the vanadic acid in the ore is thus obtained in solution. Silica is removed from the solution by careful addition of sulphuric acid, and filtering. The vanadic acid is then precipitated by adding excess of sulphuric acid, evaporating and washing. The precipitate contains some 5 to 8 per cent of impurities.

If ferro-vanadium is the object, the solution of sodium vanadate can be precipitated by adding iron sulphate and sodium carbonate, in such proportion that two parts of iron are precipitated for every 1 part of vanadium. The precipitate is filtered out, dried, mixed with aluminium powder and reduced to an alloy of 33 per cent vanadium to 66 per cent iron, by Goldschmidt's process. If nickel sulphate, copper sulphate or cobalt sulphate are used, instead of iron sulphate, alloys of vanadium with nickel, copper or cobalt are obtainable in a similar manner. Vanadium-nickel alloy is also made by mixing vanadic acid with nickel oxide, adding carbon, mixing, pressing into cubes, and strongly heating in a carbon-lined crucible. The alloy thus obtained, however, is not so homogeneous or free from impurities as that made by reduction by aluminium, but the process is cheaper.

BOOK REVIEWS.

Grundzüge der Siderologie: III. Die Hüttenmännischen Prozesse. By Hanns Freiherr von Jüptner, Leipzig: Arthur Felix, 1904. 422 pages. Price, marks 18.00.

This third part completes von Jüptner's treatise. It has appeared in two sections, the first treating the behavior of iron towards various reagents or other elements, the second of the metallurgical processes for producing iron and steel.

These treatises have very marked characteristics, making them take a place in the metallurgy of iron never before so satisfactorily filled. They are, in brief, treatises on the chemistry and physics of iron, presenting the scientific principles on which its metallurgy is based. For descriptions of processes, mechanical details and illustrations, the reader must look elsewhere, such as to the works of Turner, Sexton, Wedding or Yedebur; for elementary explanations for a beginner, Bauerman's many-editioned work would be recommended in preference; but for the experimental results by which a thorough insight is given the metallurgist into the scientific principles involved in the study of iron, von Jüptner's work is incomparable.

The subject is so vast, and its scope so various, that no one man could be an expert on the whole of it. Von Jüptner's work has therefore been mostly that of a compiler, and, as such, there will be found minor shortcomings, such as a repetition of results at places without as intelligent a criticism of their value as might have been given. But, the work will be of most use to that class of metallurgists who simply want the data, and are often as competent as the author himself to make the evident criticisms, and to properly weigh the value of the information. While not a work of genius, yet it has been patiently, carefully and meritoriously done, and the author has gained by it the well-deserved thanks of the metallurgical world.

The Brass World.—The first number of a new journal, under the title *The Brass World and Platers' Guide,* has just been issued. The journal will be devoted to the art of refining, alloying, casting, rolling, founding, and electroplating of all the non-ferrous metals. Mr. Erwin S. Sperry, of Bridgeport, Conn., is the editor. Some contributions to the first issue refer to fire cracks, their cause and prevention; black nickel-plate and its production; a new cheap white metal for sand-casting, and numerous smaller notes. We wish every success to our new contemporary.

Producer Gas Engine Electric Plants in Chemical and Metallurgical Works.

By Frank C. Perkins.

Electric power is now being extensively used, not only in the largest chemical works but also in all of the recently erected iron and steel plants. The high power gas engine is being introduced in the largest plants in England, Germany, Austria and other Continental countries, directly for power purposes as well as for generating electrical energy for lighting, power and chemical processes. Thus, two 250-hp. Deutz double-acting, single-cylinder engines, direct connected to continuous current electric generators, have recently been installed at the chemical works at Kalk, near Cologne, Germany.

In large chemical works gas producers are frequently employed for supplying the fuel for high power gas engines in the electric power stations, the economy of the entire installation being equal to, or better, it is claimed by prominent engineers, than the best steam engine power house equipment. Fig. 1 shows a gas producer plant of 10,000-hp. capacity erected at the Castner-Kellner Alkali Co.'s chemical works at Runcorn. This installation was furnished by the Power-Gas Corporation, Ltd., of London, and is used for furnace work as well as for supply gas for a number of Koerting gas engines, each having a capacity of 700 B. H. P.

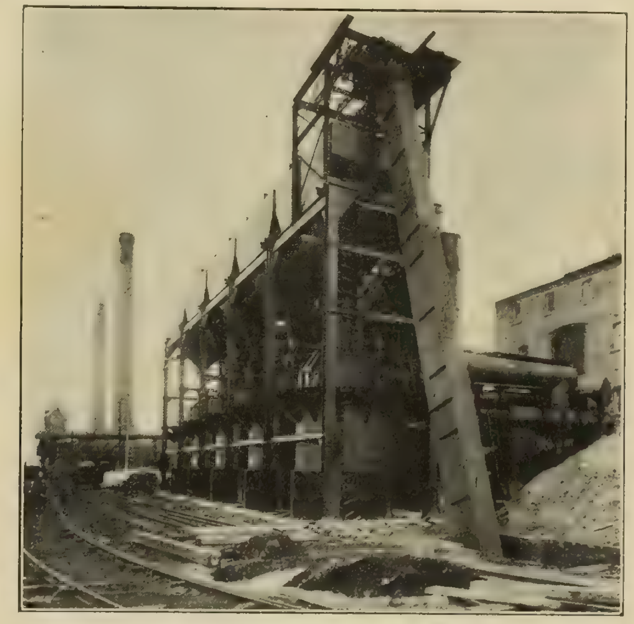


FIG. 1.—GAS PRODUCER PLANT.

Another large gas producer plant has recently been installed by the same company at the steel works of William Beardmore & Co., of Glasgow, Scotland. The ultimate capacity of this installation will be 200 tons per day, but at the present only five producers are in operation. The gas is used for the steel furnaces as well as to drive Koerting and Oechelhauser gas engines of 1000 and 500 B. H. P. each.

Gas producers are necessary at iron and steel works for supplying fuel to gas engines when the waste blast furnace gases are not adequate for providing sufficient power, but at present high power gas engines utilize these waste gases extensively, and with the greatest of economy. Fig. 2 shows a 300-hp. single-cylinder, double-acting Deutz gas engine driving a direct-current generator at the power plant of the Lauchhammer Iron Works.

The suction gas producer is being largely employed; the advantages claimed for it are that no boiler nor gas holder is necessary, and that it can be erected without danger in [illegible] plant, which it is easy to start and operate. With the suction gas producer plant only a small floor space is required, and there is no soot, smoke or odor, and the highest and best utilization of the fuel is obtained, there being automatic generation of the gas by the engine. Combinations of [illegible] gas producers and gas engines are built, for instance, by the Gas Motoren Fabrik Deutz, represented in this country by the Otto Gas Engine Works, of Philadelphia.

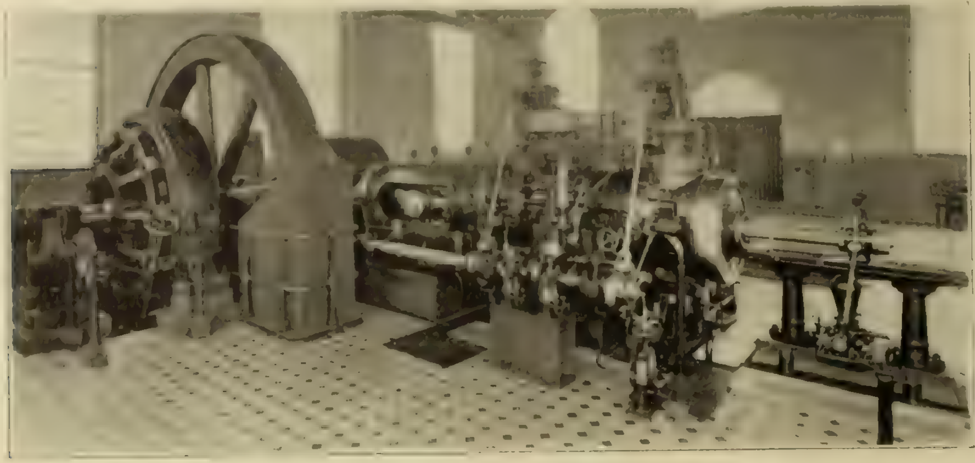


FIG. 2.—BLAST FURNACE GAS ENGINE ELECTRIC PLANT.

It is claimed by prominent engineers that a high advantageous utilization of fuel results from employing gas engines which take their supply from producer gas power plants constructed for generating out of a suitable fuel a proper mixture of hydrogen and carbon monoxide. Some of the gas producer plants of the pressure type have boilers for providing steam which is mixed with air; by means of a fan or steam jet blower it is forced through the glowing fuel, with the resulting oxidation of the fuel to carbon monoxide, while hydrogen is simultaneously obtained from the steam. The gas engine receives these combustible gases from a gas holder, to which they are conducted after passing through scrubbers.

The gas producers of the suction type are said to be cheaper and simpler, requiring no gas holder or boiler. In these plants an evaporator takes the place of the boiler, being constructed on top of the producer in some cases and in the form of a tubular evaporator in larger installations. The steam is generated in the evaporator by taking advantage of the heat of the producer and of the gases, the firing of an independent boiler being avoided. It is claimed by engineers favoring this type of producer that it utilizes 80 per cent of the fuel and the grate is kept cool by passing the steam under the same. It is also claimed that the suction gas producer can [illegible] from a part load to an excessive load, as the [illegible] le is formed and the steam provides the hydrogen very rapidly. The gas producer is said to work well with either coke, charcoal, or anthracite coal, supplying the gas engines with an abundance of fuel at high economy.

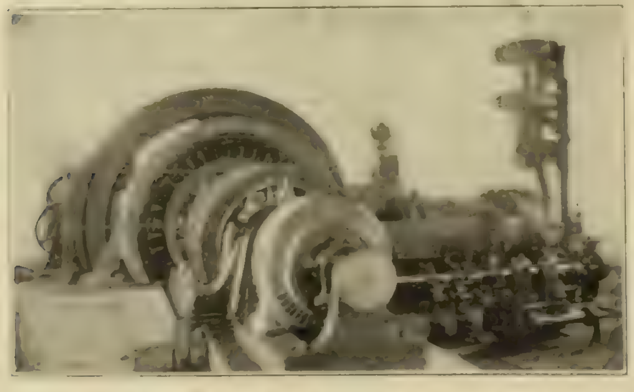


FIG. 3.—COKE OVEN GAS ENGINE ELECTRIC PLANT.

Where waste gases are obtainable from either coke ovens or blast furnaces, the gas engine is particularly desirable and is largely employed in Europe, and is now being introduced in this country. Fig. 3 shows a twin-cylinder, single-acting gas engine of the Cockerill type operating in Witkowitz. This coke oven gas motor operates a 150-kw. three-phase generator by direct connection at a speed of 100 r. p. m. The gas engine was constructed by Breitfeld, Danek & Co., of Prag-Karolinenthel, Austria, Hungary, and the three-phase alternator by Ganz & Co., of Budapest-Leubersdorfs, the frequency being 50 periods per second, and the pressure 575 volts.

The earlier electric stations operated by gas engines supplied with producer gas were not of the direct-connected type, but were belted to the dynamos as at the gas engine electric power plant at Uster, near Zurich, Switzerland. A number of gas producer systems are now in operation successfully in Europe and America, and there is every reason to believe that during the next decade the gas engine will occupy an important place in the electric light and power stations throughout the world as the high efficiency of the entire installation compares well with the best plants using other prime movers.

Revolving-Hearth Roasting Furnace.

The development in late years of so many mining properties containing large quantities of comparatively low grade and refractory ores, has created a demand for a furnace of large capacity, in which such ores can be roasted preparatory to chemical or further igneous treatment.

Heap and stall roasting and the hand reverberatory roasting furnaces have, in turn, made way for furnaces in which the ore is stirred or rabbled by mechanical means. The designs of the first mechanical roasting furnaces followed closely along the lines of the hand reverberatories, but the later designs have been, in nearly every instance, furnaces having a round hearth and revolving rabbles. In the Holthoff Revolving-Hearth Furnaces, the design of Mr. Henry C. Holthoff, of Milwaukee, Wis., this principle is reversed. The hearth, which is annular in form, is made to revolve about a centrally located gas producer. The wall of this producer forms the inner wall of the roasting chamber and support for its arched roof, while the outer wall is supported on cast-iron columns or structural steel.

The gas generated in the producer enters the roasting chamber through a number of pipe channels radially disposed in the walls of the producer. Air pipes are also located in the walls of the producer, the lower ends opening to the atmosphere through a regulating valve, the upper ends extending into the roasting chamber in close proximity to the gas exits. The air, heated in its passage through the producer walls, mixes with the gas upon extrance to the roasting chamber, and combustion is spontaneous. The flame traveling radially across the hearth finally discharges through a series of tile pipes into a large circular smoke flue supported on steel frame work just above and around the outer edge of the furnace. It will be noted that while the hearth resolves, the main furnace structure is stationary.

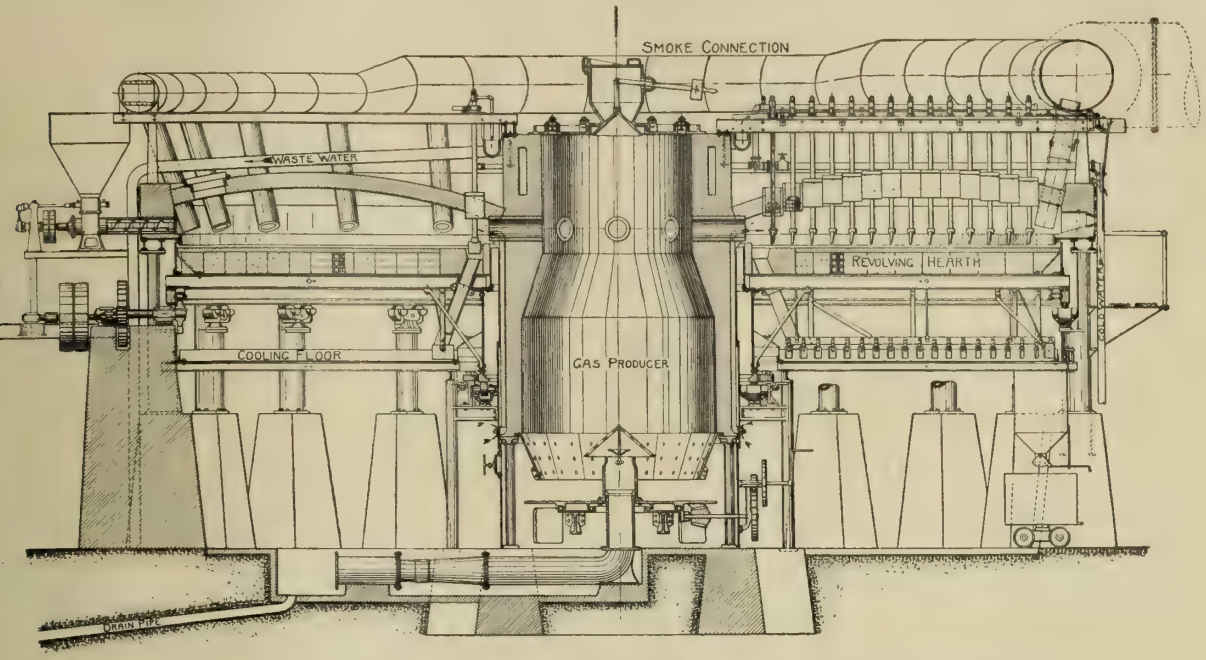


FIG. 1.—CROSS-SECTIONAL VIEW OF HOLTHOFF REVOLVING HEARTH ROASTING FURNACE.

The rabble arms, of which there are four sets, extend down through the arched roof of the roasting chamber, and all cooling water connections and adjustment of the angularity of the rabble blades are accomplished outside of the furnace. The rabble arms consist each of two pieces of cast-iron pipe of different diameters, one within the other, the outer pipe being tapped into the hollow rabble blade. This arrangement is a simple and effective means for water-cooling the rabbles.

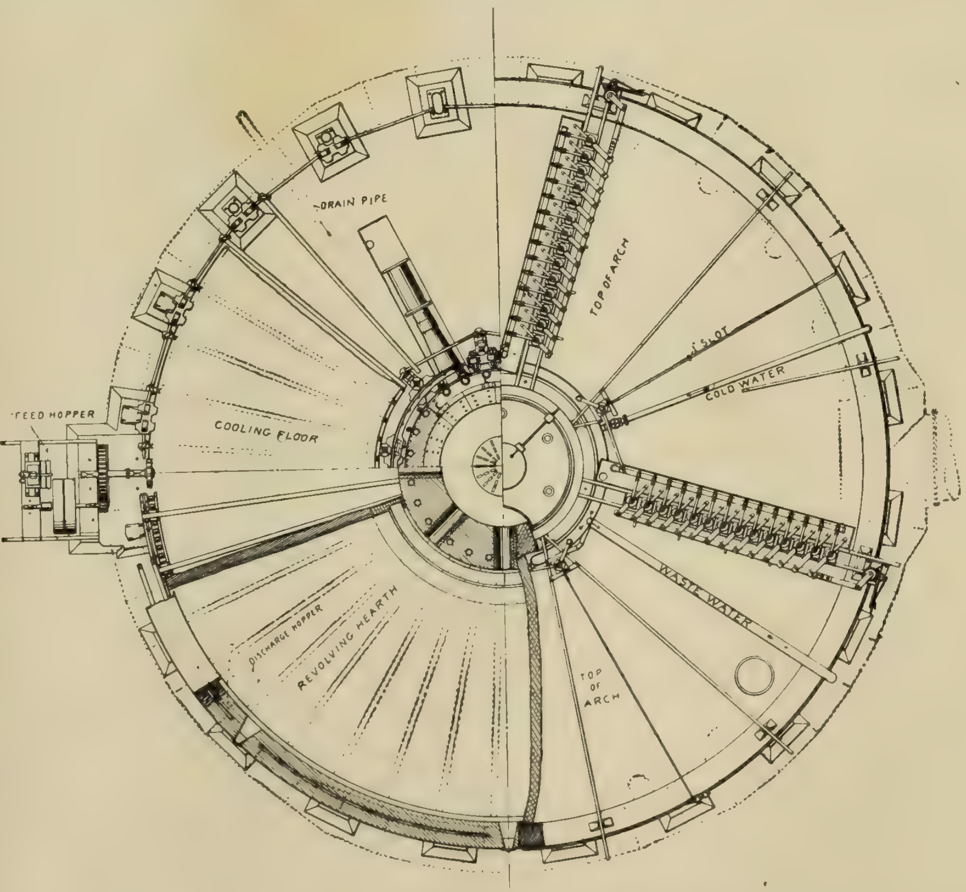


FIG. 2.—PLAN OF HOLTHOFF REVOLVING HEARTH ROASTING FURNACE.

The ore, fed through a feeder having an adjustable eccentric and a spiral conveyor extending through the outer wall of the furnace, drops on to the outer edge of the hearth, and, as the hearth revolves, is rabbled across same in a spiral path. Reaching the inner curb of the revolving hearth, it drops through discharge spouts on to the inner edge of the stationary cooling floor located directly under the revolving hearth. It is then moved across the cooling hearth and finally discharges at the outer edge of the cooling floor into a revolving hopper or to a belt conveyor or elevator.

The hearth revolves upon equalizing roller bearings of conical form, and is revolved by means of a roller rack and a pinion. From the inside of the mantel upon which the outer wall rests, a steel metal apron is hung, extending down inside the outer curb of the hearth, and the top of the ore bed being higher than the bottom of this apron, the opening between the hearth and the wall of the roasting chamber is effectually sealed against the entrance of cold air. By means of valves or dampers located in the air inlet and smoke discharge pipes, the temperature in the roasting chamber is under complete control of the operator at all times, and, as the angularity of the rabble blades can be changed quickly and easily, the time the ore remains in the furnace is also under complete control. The angularity of all the rabbles of one set is changed by the movement of a lever conveniently located for the operator, and which slides in a quadrant on which are figures indicating the degree of angularity of the rabbles.

The furnace above described was designed especially for treatment of low sulphur ores, such as are found in the Cripple Creek district and was built about a year ago for the Portland Gold Mining Company's mill at Colorado City.

In a two months' test run of this furnace the following results were obtained, the figures given showing the average per day of twenty-four hours during the entire test.

Tons of raw ore charged	99.82
Per cent Sulphur in raw ore	2.572
Per cent Sulphur in roasted ore (sulphides)	.142
Tons of coal consumed	9.45
Pounds of coal per ton of ore charged	189.80

These furnaces are also built in three other types, one similar to that described above, except that a plain fire-box is substituted for the gas producer. Another type designed for roasting sulphide ores preparatory to smelting, is built along the same general lines as that described above, except that the cooling floor is omitted and the roasted ore discharges automatically into a stationary receiving hopper from which the calcines are drawn off into cars or buggies and taken to the smelting furnace.

The fourth type of furnace is designed for roasting high sulphur pyritic ores. In this type of Holthoff furnace the cooling floor is omitted and a plain fire-box is substituted for the gas producer. This fire-box, however, is used only to ignite the sulphur in the charge. After ignition, the sulphur burns spontaneously, and the fire-box is used merely as an air flue.

The advantage claimed by the builders of these furnaces, the Power & Mining Machinery Co., at Cudahy, Wis., are: First, great capacity. Second, ability to control the temperature in the roasting chamber, as well as the rate at which the ore is moved across the hearth, thus giving the operator complete control of the roast during the entire operation. Third, by using a gas producer incorporated in the furnace structure, no heat is wasted in piping, and the expense of installation and repairs for piping, found in furnaces in which the fire-box is separate from the furnace proper, is eliminated. Also, by generating gas in the producer, the combustion of which takes place in a roasting chamber, the greatest possible utility is made of the heating value of the fuel, and it will be noted from the figures given above that the fuel consumption per ton of ore charged is exceedingly low. Fourth, the small amount of manual labor connected with the use of these furnaces is an important item. The only labor required is for lubricating and such general attention as is due any piece of machinery however simple, and one man can attend several of these furnaces. Fifth, throughout the general construction of these furnaces, provision is made for expansion and contraction of the various parts, so that the users are saved the annoyance and expense of repairs from this cause, which are so frequent in other furnaces.

The Power & Mining Machinery Co., Cudahy, Wis., are issuing their Bulletin C-1, describing this furnace, which, we are informed, will be sent gratis on request. This bulletin contains complete detailed description of the design, construction and operation of these furnaces.

Personal.

Mr. **Ernest Kilburn Scott**, of London, England, has been appointed to organize and take charge of the Electrical Engineering Department at Sydney University. Mr. Kilburn Scott's name is well known to our readers as a virile technical writer of very wide and detailed knowledge. He recently contributed to our columns some information regarding the manufacture of a refractory furnace lining material from magnesite. Mr. Scott carried out this work for Mr. Turner, the owner of the magnesite mines at Salem, in Southern India, who has recently been awarded a gold medal at the St. Louis Exposition for such furnace linings.

Prof. **Walter Nernst**, of the University of Goettingen, the distinguished founder of the osmotic theory of galvanic cells and inventor of the Nernst light, has accepted a call to a chair for physical chemistry at the University of Berlin.

While going to press, we learn with deep regret of the sudden death, on January 25, of Mr. **E. H. Mullin**, of the New York office of the General Electric Co. Mr. Mullin had been ill with grip for four or five days.

Digest of U. S. Patents.

Prior to July, 1902.

Compiled by Byrnes & Townsend, Patent Lawyers, National Union Building, Washington, D. C.

Molten Electrolytes, Miscellaneous.

(Concluded from page 48.)

679,253, July 23, 1901, A. H. Cowles, Cleveland, Ohio.

Volatile elements such as sodium, potassium, zinc and phosphorus, are extracted from fused compounds by electrolysis, the specific example given being the extraction of sodium from a bath containing sodium fluoride and aluminium fluoride in which sodium carbonate is dissolved. Upon electrolysis oxygen is set free at carbon anodes, while the sodium is deposited upon a porous hearth produced by slowly baking a mixture of coke, granular carbon, and a heavy hydrocarbon. The bath is maintained above the volatilizing temperature of sodium, under which conditions the sodium passes through the porous hearth and is condensed in the surrounding casing. The condenser is preferably heated in order to permit the removal of the sodium by tapping. The vessel is of metal, the sides being cooled, and protected by a layer of the solidified electrolyte. A receptacle employing twenty carbon anodes, each 2½ inches in diameter, will carry about 3000 amperes at 5 volts. The bath should be periodically renewed by addition of sodium carbonate.

679,997, August 6, 1901, George P. Scholl, Philadelphia, Pa.

For the electrolytic separation of sodium employs a bath containing caustic soda and sodium sulphide in equal parts. The bath has a somewhat higher melting point than sodium hydrate, but has the advantages that the e. m. f. required for decomposition is lower, and that less hydrogen at a lower temperature is liberated at the cathode. It results from the latter advantage that a single cell may be safely enlarged and greater economy of operation secured. The electromotive force is regulated to decompose the sulphide only, the liberated sulphur reacting with the hydrate to again form sulphide. Additional quantities of hydrate are supplied during the operation. The method is stated to be applicable to metals other than sodium, the required condition being that the electrolyte comprise a sulphur compound and a fusible compound readily converted into a sulphide.

699,851, May 13, 1902, C. W. Roepper and G. P. Scholl, Philadelphia, Pa.

An apparatus for the electrolysis of fused salts, designed particularly with a view to use in large installations. The chief features are the provision of separate intercommunicating heating and electrolytic chambers; the employment of a number of electrolytic chambers grouped around or otherwise arranged in conjunction with a single heating chamber; a system of jacketing whereby a thin incrustation of the solid electrolyte protects parts from the corrosive action of the bath; and a construction which provides for the ready replacement of parts which are subjected to the most rapid deterioration. Many alternative constructions embodying the foregoing features are shown. The fusion of the electrolyte is accomplished by gaseous products of combustion passing over the material; the latter may be protected from the gases by a floating layer of carbon, sand, or melted slag. No specific example of the use of the apparatus is described.

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NEW YORK, MARCH, 1905.

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Power From Blast Furnace Gases.

In this issue we publish an admirable article from the pen of Mr. F. duP. Thomson who speaks authoritatively on this subject on account of his connection with the installation of the pioneer blast furnace gas power plant of this country. The subject is specially interesting in its relation to the problem of the use of the electric furnace in the steel industry. Aside from special cases with particular local conditions, it seems at present that the proper place of the electric furnace in the iron and steel industries is as an adjunct to, not as a competitor with, the blast furnace. The enormous amount of power which may be rendered available from blast furnace gases will be seen from the following estimate. The output of pig iron in this country in 1904 was over 16,500,000 long tons; on the basis of Mr. Thomson's quite conservative figure—468 hp. per ton of iron made per hour—this represents the possibility of developing nearly 900,000 hp. above all the requirements of the blast furnaces themselves. The significance of this figure is evident in view of the fact that, according to the last census report (1902), the total power in all electric central stations in the United States was 1,800,000 hp.

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Metallurgical Calculations.

We are glad to announce that, beginning with the present issue, we will publish in our columns a serial on metallurgical calculations from the pen of Prof. J. W. Richards, whose enthusiasm in progressive metallurgy is coupled with a happy faculty of elucidating intricate problems, gained through his many years' experience in metallurgical teaching and consulting work. The scope of the serial is given in the introduction of the first article, and a comprehensive scope it is. We trust that this serial will gain many friends and may do much good; for the future of metallurgy depends chiefly on the sound amalgamation of practical metallurgical experience with the fundamental principles of chemistry and physics, which must underlie all metallurgical thinking. The time is over when the "rule of thumb" rules. The future belongs to the man who plans accurately what he is going to do and who controls what he is doing by checking the figures obtained in his work with the figures calculated in advance, to see how far he is successful in accomplishing what he is after. Metallurgical calculations were formerly mistrusted by practical men, and criticisms of this kind were occasionally backed up by references to cases in which a certain formula taken from a certain pocket-book and applied in practice did not give correct results. The fault, however, was less with the formula than with the man who tried to apply it at random. It is a most dangerous thing to apply any formula whatever without completely understanding its derivation and all the suppositions on which it is based. Formulæ are nothing but a kind of short-hand writing; there can be no value in a formula if it is not based on sound reasoning, and the best formula is of no value in a special case if the conditions in this case are different from those assumed in the derivation of the formula.

What a metallurgist should aim to acquire is not a collection of formulæ—a treasure of doubtful value—but rather a faculty of making calculations for himself. This requires above all sound common sense, which cannot be taught; but it also requires a knowledge of facts and principles. There are, indeed, a certain number of fundamental principles of reasoning which will fit a great many apparently different cases, and which come up again and again in metallurgical calculations, like a Leitmotiv in an opera of Wagner. These leading principles of calculation can be taught and learned, and we trust that Prof. Richards' serial will contribute to their dissemination throughout the metallurgical world. When a metallurgist is no longer ashamed nor afraid nor unable to check his work by calculations, then and then only is he enabled to determine exactly the weak points of his process and the chances of improvement. And even in such cases where a sound metallurgical calculation yields results which are not confirmed by practice, there is no reason to sneer at the theory. Gayley's famous dry-blast experiment is a case in point; he obtained a saving of fuel several times greater than was to be expected from calculation. But this result simply shows that we did not yet know everything which was really worth knowing concerning the phenomena in the blast furnace—that the explanation of this disagreement must lead to a better understanding of the blast furnace. We can promise our readers that Prof. Richards' articles will not be all easy reading, they will at times require study; but we think we can also promise that whoever masters them will gain much and varied practical information of metallurgical principles.

Continuous Versus Intermittent Process.

In our last issue, we made editorial reference to the far-reaching and fundamental question of the metallurgical difference between a continuous process and an intermittent process. There are to be found in America's industrial development numerous good examples of each type. We mention among the continuous processes that of the lead, copper and iron blast furnaces, of the cement kiln, the mechanical ore roasters, crushing machinery, ore-dressing machinery and gas producers. In the intermittent processes, are found the desilverizing base lead bullion, bessemerizing pig iron, or refining pig iron to steel in the open-hearth furnace, bessemerizing high-grade copper matte to "blister" copper, or doing the same operation in the reverberatory furnace (all four of the last instances showing a curious parallel between copper and steel metallurgical practice), etc. In this classification it is seen that, as a general rule, for doing the rough work on the ore with the formation of a crude concentrated product and of a waste product containing little metallic value, the continuous furnace has won in the great metals, iron, lead and copper, undisputed sway, while for the finishing or the refining [illegible] the intermittent process [illegible].

For such a generalization as this there must be some reason. It was a dictum of the late Herbert Spencer that nothing in this world of chance results in an orderly arrangement without a cause for this order. The cause for order in this case is that in a continuous process the conditions are held nearly rigid while the material in the process flows, while in an intermittent process the material is fixed, while the conditions can be varied to suit the needs of the treatment. In the first case, the quality of the product is a matter of some indifference, the goal to be attained is a large tonnage handled at a low cost. In making lead bullion, the small amounts of such impurities as copper, arsenic, antimony, etc., will vary greatly from day to day without serious effect on the cost-sheet of the lead smelter. The real problem is to increase average monthly tonnage of plant by keeping furnaces open and running at full capacity, and to make a large proportion always low in silicon and lead, so that only small percentage needs be retreated. The first treatments are really concentration of values. On the other hand, in the refining process quality is of all importance. To control quality of product, the operator must be able to vary conditions at will. No refinery can sell metal of a purity of 99.9 per cent one week, and the next week of 99.0 per cent. As the refining process requires such to be the state of affairs, the intermittent process fits the category nicely and exactly. A central furnace reservoir, like the "mixer" of the modern steel plant, will average the impurities, but the open-hearth furnaceman, by his control of flame and fluxes, can eliminate them to the extent demanded by the use of the finished steel. By putting a chemical laboratory near the furnace when steel of desired quality is produced, the treatment is stopped and the furnace tapped.

To expand this idea, we see that the cost of labor per ton of output on a continuous furnace is low, for if the conditions be held constant and the charging of the furnace be constant also, mechanical means can be devised to do the work. The units are made of tremendous size, for everything is known. There is no time lost in charging the furnace, and thus capital invested in it is not idle for a part of the time. The continuous furnace is not subjected to strains in cooling during charging, and thus its life is longer. "Continuous processes make continuous profits." Two curious half-exceptions are found to this generalization. These are zinc and aluminium. The reduction of zinc is done in an intermittent final process. The refining has been done largely in the ore-dressing, which is a continuous and concentrating process. In the metallurgy of aluminium, the refining is really done in the first chemical process extracting by caustic soda the pure aluminium oxide from the crude bauxite. The final reduction is done by the Hall continuous electrolytic process. Both these exceptions have common properties: great difficulty of reduction and their volatility at temperatures of reduction. One requires heated carbon in a closed retort; the other requires less heat, carbon, and that great reducer—the electrolytic potential of the direct current. So both these "half-way" exceptions show a logical similarity in their partial dissimilarity. In industrial electrochemistry, it is seen that high temperature furnaces, *e. g.*, graphite and carborundum furnaces, are intermittent because their extremely high temperatures apparently make a permanent continuous furnace nearly an impossibility. The carbide furnaces are semi-continuous in operation. Electrolytic copper refining possesses peculiar characteristics, but it can be put in the intermittent class. The Castner-Kellner caustic soda cell is beautifully continuous. In the electrometallurgy of steel, the electric furnace is of the refining intermittent type—which is just its proper place. The great example of intermittent treatment applied to ores is the leaching of ores by solvents, such as cyaniding gold ores. Here, we find a strong desire for a continuous leaching plant and filter press. Such a plant would be cheaper in construction and operation. Another point to be noted is that most primitive processes and most processes in the experimental stage are intermittent. This is but natural, for when the right conditions of the reaction are yet unknown, it is impossible to do the hard task of working under rigid conditions. Thus it is with the old Chinese processes of making iron by heaping up coal about a crucible containing iron ore, piling clay about it and tearing down the crude furnace at the end of the operation. This is quite analogous to our present high temperature electric furnace. As Americans have only been making carborundum at some 3000° C. for ten years, while the Chinese have been making iron at some 1400° C. for a few thousand years, we may not be ashamed to be compared to the Chinese in this respect. The perfectly natural law for the classification of metallurgical reactions shows that the same underlying philosophical principles pervade applied science, as well as all human relations. They are always in the mind of the industrial pioneer, sometimes consciously, but often unconsciously; for, in general, intuition, not analysis, guides the inventor.

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Trusts and Metallurgical Advance.

Evolution is defined as a process of change from a state of homogeneity to a state of heterogeneity through a series of differentiations and subsequent integrations. By such a process has nature evolved out the primordial "world-stuff"—the simple electron—the large complex molecule of albumen which contains several million electrons. By such a process man has come to his present physical and spiritual being from the first little proto-plasmic bit. And every organism is now growing from the simple to the complex. As long as man has been ministering to his needs the changes in his methods of ministering have been along these same lines of organic growth. Not so long ago each man was his own "butcher, baker and candle-stick maker," but the growth of society has caused a rapid classification of industry. This progress has reached its apogee in a corporation employing over one hundred thousand men, owning its own mines, steamships, railroads, iron and steel works, and plants for turning the steel into many finished products. Such a corporation is a wonderful and complex organism, even more wonderful and complex than the albumen molecule. Thus the large corporations commonly called "trusts" are the legitimate outgrowth of economic evolution. Under present conditions they cannot be repressed by hostile legislation. The intelligently and honestly directed trust is omnipotent. Possession of large working capital, large-scale production, ownership of large sources of raw material, control of the market from unnecessary fluctuations are a few of its advantages. Strong financial connections, distributing orders to its plants according to favorable geographical locations, employing labor regularly 300 days a year (the only efficient way), pooling metallurgical experiences of all plants, all these present a striking array of secondary points of industrial superiority. Concerning the last point—the pooling of experience due to consolidation of companies—the following true story is instructive: A company absorbed recently two rather widely separated street railway systems. The superintendent of the one system was instructed to report on the operation of the other road, and suggested a number of improvements which had escaped the attention of the resident manager. The result was such a decisive reduction of the cost of operation that the manager of the second road was instructed to report on the first road. He recommended economies of an entirely different kind, but which were equally effective in lowering the operating cost.

Like every giant, however, the trust has weakness. One weak point will be like Goliath's, in the head. The present able managers and engineers were developed by strongest and keenest competition. They are men who operated plants under adverse conditions; men who installed new machinery; bold innovators and pioneers; men full of resource, patience and courage. Their endeavors built the great metallurgical plants of the country. Thus comes up the question, how will the successors of these men be trained? The policy pursued as to training young men, of seeming unimportance now, can make or unmake the corporation in the future. If the managers and engineers of the future are not trained in the spirit of our age, and are not of the highest type, "dry rot" will spread and the trust will be, 100 years from now, an extinct species, like the other large animals, such as the Brontosaurus in the New York Museum of Natural History. Obviously, there must be new blood infused into the technical and business staff. How to do that logically and effectively is one great problem for the directors of a trust. Much of course is done by metallurgical competition between the different plants. This will effect the details of the process, so the things are not hopeless. But the great basic improvements will come in large part from outside competition. In many instances the metallurgical nestors turn down a new process from natural conservatism. A new process will, if successful, nullify the investment value of many millions of dollars; while, if turned down by the trust, it may fail for commercial reasons. This has often been the case. On the other hand, the independent can scrap the small old plant and build a new plant on radically new lines of quadruple capacity. This move has often been made, and is a good stroke of business. For the "largest interest" to do this would be impossible, for the increased capacity would exceed consumptive demand of the country. The trust could not do more than build to its present capacity. The smaller concern would have much better reasons and stronger incentives for taking chances on the first work of a new process. Were it not for the fact that the logical attitude of the belligerent independent is to stimulate metallurgical activity and that this will react favorably on the trust, the outlook for advance would be most gloomy. The large corporation without active competition has the best of reasons to rest on its oars. Any legitimate enterprise can obtain capital for industrial purposes. So we have reasonable hopes for believing that American independence of action will keep up the continuous march of progress in applied science.

Notes.

American Electrochemical Society.—In the February meeting of the Board of Directors, Mr. F. A. Kjellin, of Gysinge, Sweden, and Mr. Simon G. Engle, of Monticello, Ind., were elected members of the society. A preliminary announcement of the programme of the Boston meeting will be found further below.

Society of Chemical Industry.—The programme of the meeting of February 24, of the New York Section of the Society of Chemical Industry, comprises papers by C. Richardson and C. N. Forest on tetrachloride of carbon and its use as a solvent in the differentiation of bitumen; H. Schweitzer on the history of artificial musk; W. G. Berry on tannin.

Electrometallurgy of Iron and Steel.—At a recent meeting of the Canadian Society of Civil Engineers, Dr. E. Haanel, superintendent of mines, of Ottawa, Canada, read a paper on the electrothermic production of iron and steel. He summarized concisely the work and the conclusions of the Canadian Commission of which he was the head (abstracted at length in our December issue, 1904). The author predicted a very rapid development of electrical methods. He pointed out that the older metallurgical methods of steel-making had almost reached their limit of economic development, and no very startling reductions in cost are probable, whereas electrothermic methods, although new, have already been so successful as to justify the belief that they will soon become of great commercial importance.

Clays.—In circular No. 17, issued by the United States Department of Agriculture, Bureau of Chemistry, Dr. A. S. Cushman gives a very useful summary of our knowledge of clays. The contents of the pamphlet are as follows: Formation of clays, production and importation, kinds of clay, physical properties of clay (plasticity, binding power, slaking, air shrinkage; firing qualities, namely, fire shrinkage, fusibility, color; absorptiveness, purification and treatment of clays, use of clays, testing and examination of clays.

Worcester Polytechnic Institute.—We have received the thirty-fifth annual catalogue of the Worcester Polytechnic Institute for 1904-5. The total number of graduates of the Institute is now 1057, the number of members 2020, so that the percentage of graduates is 52.

Optical Pyrometry.—We recently referred to the most excellent summary by Dr. C. W. Waidner and G. K. Burgess of the present status of optical pyrometry (bulletin No. 2, Bureau of Standards). This paper has now been published as a pamphlet of the Department of Commerce and Labor, Bureau of Standards.

Electric Steel Furnace.—The Héroult process for making steel in the electric furnace was the subject of a recent paper of Mr. Charles Combes, presented before the Society for the Encouragement of National Industry in Paris.

The Technical School and the University.—A most interesting paper on this subject was recently presented by Prof. Henry M. Howe before the Association of American Universities in Baltimore: "Will technical schools serve the interests of the community better if they are parts of great universities, or if they are isolated institutions? Is association or isolation more to the public good?" * * * "The technical man is always occupied with questions of profit and cost, of making money for his employer, or of building as cheaply as his standard of quality will permit. The merit of every plan is measured in dollars, be it mining and smelting, manufacturing, transportation or agriculture. The subject must be taught from this standpoint. Our graduates must be sufficient money-makers; but it is still more important for the community that they should be liberal citizens." * * * "The technical teachers should have the softening and broadening influence of contact and work with teachers of the humanities and of pure science. * * * Conversely, it is for the good of the public that the teachers of pure science, and more particularly those of the humanities, should, in turn, be broadened by contact with the technical school."

Prof. Howe then spoke of the interaction of different classes of students. "The serious technical student unexpectedly learns from his fellow of the college, of the delights of this or that writer; of the existence of this or that school of philosophy; his interest in things beautiful is awakened; a chat with the student of architecture sets him thinking about the genesis and meaning of logical finished Doric and soaring spiritual Gothic. Each student from another department cries '*Audi alteram partem;* however good and healthy your interest may be, they are only one group out of many.' The technical student's horizon is broadened; and truly it needs broadening. According to my observation, narrowness is the chief defect of the isolated technical school. Like students of the other learned professions, the technical students, are as a class more earnest, more zealous, than their fellows of the college; they are more mature, and they see more clearly than the college student the bearing of their studies upon their life's work." However, there may be conditions where the matter is not so simple, for instance, if in creating a new technical school, the choice should lie between isolation and association, not with a university, but with some long-established college which has as yet no professional schools, and thus is itself isolated. Prof. Howe then discussed the economy resulting from association—in saving salaries and in widening the use of the more expensive instruments, as well as in fitting work to worker, and best of all, in more fully supplying the eminent with work on their own plan. Prof. Howe concludes that "wisely guided association, while it need neither deprive the technical school of character and individuality, nor injure any of those in interest, should benefit the community, whose welfare here deserves our chief thought."

American Electrochemical Society.

The annual meeting to be held in Boston, Mass., on April 25, 26, 27, promises to become a most memorable one in the history of the society. The local committee, with Prof. W. H. Walker as chairman, is making extensive arrangements.

The following preliminary list of papers has been announced:

A. B. Albro, "Microstructure of Silicon and its Alloys."

J. C. Blake, "The Migration and Coagulation of Colloidal Suspensions."

H. R. Carveth, "Chromium."

Ernst Fahrig, "Treating Low Grade Ore and Tailings by Electrolysis."

H. M. Goodwin, "On Billitzer's Method of Determining Absolute Potentials."

C. Hambuechen, "An Optical Method of Observing Diffusion in an Electrolyte."

Carl Hering, "Removing Solvents During Electrolysis."

G. A. Hulett, "A Low Voltage Standard Cell."

L. Kahlenberg, "Osmosis."

A. Lodyguine, "Some Experiments with the Reduction of Titaniferous Ores Into Steel and Titanate of Iron."

A. Lodyguine, "Experiments with the Reduction of Different Oxides of Lead by Electric Current."

A. Lodyguine, "Some Results of Experiments with the Electrodeposition of Metals on Aluminium."

A. Reuterdahl, "The Interdependence of Atomic Weights and Electrochemical Equivalents."

J. W. Richards, "Conduction in Electrolytes."

C. C. Richardson, "Water Powers of the Canadian 'Hinterland.'"

R. C. Snowden, "Electrodeposition of Silver."

R. C. Snowden, "Adherence of Nickel to Nickel."

C. P. Townsend, "A New Diaphragm Cell."

Other papers are expected from Messrs. W. D. Bancroft, C. F. Burgess, A. J. Rossi, and others.

We are also informed that access can be had in or near the convention hall to a plentiful supply of electric current for demonstrations and exhibition purposes, should any manufacturers of special electrical apparatus, electric furnaces, etc., desire to exhibit. While, on account of limitations of space, a general invitation cannot be extended to manufacturers and dealers to display their products, the committee will be glad to make all possible arrangements for such exhibits, should those interested apply to Prof. H. P. Talbot, secretary-treasurer of the committee, whose address is at the Massachusetts Institute of Technology.

The hotel headquarters will be at the Hotel Lenox, Boyleston and Exeter Streets, Boston.

The complete programme of the meeting will be published in our next issue.

Faraday Society Meeting.

The eleventh ordinary meeting of the Faraday Society was held on Monday, January 30, 1905, at the Institution of Electrical Engineers, in London, Prof. A. K. Huntington being in the chair.

MASS ANALYSIS OF MUNTZ'S METAL BY ELECTROLYSIS.

The first paper of the evening was presented by Mr. John G. A. Rhodin, the subject being "Mass Analysis of Muntz's Metal by Electrolysis," and some notes on the electrolytic properties of this alloy.

The first portion of the paper describes an apparatus which was specially designed by the author for the purpose of the accurate and rapid determination of the copper content (which should lie between 60.5 and 61.5 per cent) of Muntz's metal. As about 40 to 60 "heats" of metal are cast every day, and it being necessary to obtain the results within twelve hours of the time of casting, the apparatus had to be such as to enable 100 analyses to be made in twenty-four hours—with a mean probable error of not more than ± 0.1 per cent. The author decided that an electrolytic method would best fit in with these stringent requirements.

The electrodes employed (Fig. 1) consist of concentric cylinders of very fine platinum gauze, supported by stout frameworks in order to ensure even current distribution. The anode rests on a ring at the bottom of the containing vessel, so that the cathode can easily be slipped away and removed, and both electrodes are held in modified Classen stands. The present installation consists of thirty such unit cells, and current is supplied to them from six pairs of accumulators, each of which supplies current to five pairs of electrodes, through suitable nickelin resistances.

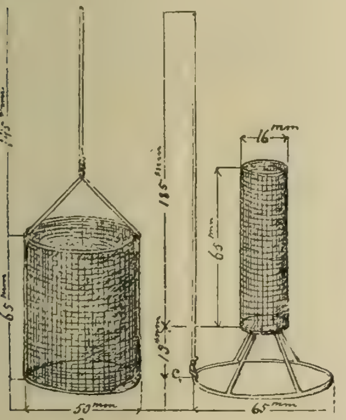

FIG. 1.—ELECTRODES.

Fig. 2 is a diagram of the connections of two sets of five pairs of electrodes. The batteries B_1 and B_2 are connected to the conductors L_1 and L_2, and L_3 and L_4, respectively. The conductors are enclosed in wood casing, screwed down to the table. Parallel branches run to the different electrodes, a_1 and c_1, a_2 and c_2, etc., but the main conductors L_1 and L_2, and L_3 and L_4, are insulated from each other between the sets, so as to prevent the batteries from discharging into each other. In each one of the secondary branches there is a resistance, R_1 or R_2, etc. These resistances can be shunted at S_1, S_2, S_3, etc. All the resistances are made from No. 20 nickelin wire, stretched on porcelain bobbins in front of the table. Hence, they can easily be got at, and this is convenient for repairs, which have frequently to be made when electrolytic work is carried on continuously.

One gram of the alloy, dissolved in nitric acid, is used for an analysis, and a current either of 0.5 amperes or else of 2.0 amperes is employed for the deposition. In the former case deposition is complete in twelve to fifteen hours, in the latter in three hours. Electrolysis proceeds in three phases: (1) Copper deposition and ammoniacal reduction of nitric acid. (2) Ammoniacal reduction only. (3) Deposition of zinc. A perfect separation takes place if phase (2) occupies a long time, and is unaccompanied by metal deposition. If certain impurities, such as arsenic or nickel, be present they must be removed chemically. The paper describes in detail the actual procedure at the works laboratory, and also includes a table showing the kind of accuracy obtained in cases where re-analyses have been made. Since these analyses in bulk have been made it has been found that they act as an infallible guide to the casters, so that remelting has now become very rarely necessary.

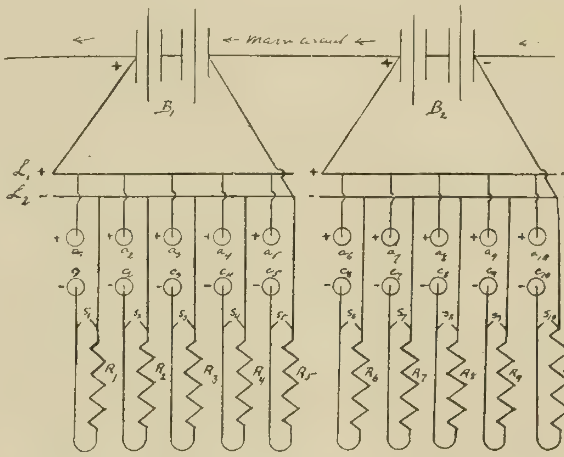
main circuit
B₁
B₂
L₁ +
L₂ −
L₃
L₄
a₁ a₂ a₃ a₄ a₅ a₆ a₇ a₈ a₉ a₁₀
c₁ c₂ c₃ c₄ c₅ c₆ c₇ c₈ c₉ c₁₀
S₁ S₂ S₃ S₄ S₅ S₆ S₇ S₈ S₉ S₁₀
R₁ R₂ R₃ R₄ R₅ R₆ R₇ R₈ R₉ R₁₀

FIG. 2.—ELECTRICAL CONNECTIONS OF CELLS.

In conclusion, the author discusses the electrochemical properties of Muntz's metal. The metal is largely used as a sheathing to protect ships' bottoms from certain mollusca and algæ, and to be successful it should dissolve in sea-water just to a sufficient extent as to render the surface poisonous, the best conditions being the equal dissolution of the copper and zinc. The author shows how these may be calculated approximately by supposing that the electrolytic dissolution rate is proportional to the heat of formation of the ultimate compounds (zinc and cuprous chlorides), and to the conductivities of the metals which dissolve. Assuming that both the chlorides of copper are formed, and taking the mean of the results in the two cases, the best values are found to be 60.811 per cent of copper, and 39.189 per cent of zinc, numbers which agree very closely with the results of practical experience. The author is now engaged in exhaustively investigating the *absolute* dissolution-velocity of pure Muntz's metal at a definite temperature, and he adds here a preliminary description of these experiments, describing in an appendix his most recent experiments on the subject. He finds that a binary alloy like Muntz's metal dissolves slowly at first, the velocity then quickly arrives at a maximum, then it falls suddenly—remaining almost constant for some time—and finally a more or less rapid fall again occurs. During the period of constant velocity the surface must alter as the resistance capacity, if the action is galvanic. The seat of e. m. f. must be in the electrolyte, as the constancy of action indicates a steady e. m. f. of considerable magnitude. It is probable that the result is influenced by the actual mass of metal present, relative to that of the solvent. The external pressure certainly exercises a very considerable effect on the reaction-velocity, by influencing the speed with which hydrogen can leave the surface of the metal.

The author promised further communications on the subject.

Equilibrium Between Sodium Sulphate and Magnesium Sulphate.

A paper on this subject was presented by Dr. R. Beckett Denison.

Experiments conducted from the standpoint of the phase rule are described, the object of which was to determine whether the double salt of sodium and magnesium sulphates, $2MgSO_4.Na_2SO_4$, which has been described as a naturally occurring mineral, is capable of existence in contact with solution; that is, whether it has been formed in nature by the evaporation of saline waters. The corresponding potassium compound is known to occur in Stassfurt as langbeinit, and it was thought that a detailed investigation might result in the isolation of the sodium langbeinit from solution.

A transition temperature of 59° C. was obtained for the system $MgSO_4.7H_2O + MgSO_4.Na_2SO_4.4H_2O$, but it was found that only the double salt loeweit, $Na_4Mg_2(SO_4)_4 5H_2O$, was formed, the usual transition-point (71°) being depressed to 59° C. by saturation with $MgSO_4$.

Dilatometer and tensimeter experiments pointed pretty conclusively to the assumption that the compound sodium langbeinit cannot exist in contact with solution, at least below 100° C., and hence, this substance, if found as a mineral, must be a product of a higher temperature.

Mr. E. **Kilburn Scott** gave a short abstract of his paper on "Refractory Materials for Furnace Linings," which dealt principally with carborundum, siloxicon, and electrically-shrunk magnesite. (Concerning the latter, see our Vol. II, page 454 and 455.)

At the next meeting, to be held on March 6, Mr. F. W. **Harbord** (the metallurgist of the Canadian Commission) will read a paper on "Electric Steel Smelting."

Society of Chemical Industry.

The American Committee of arrangements for the next annual meeting of the Society of Chemical Industry, to be held during the second week of July, 1905, at London, consists of Messrs. Russell W. Moore, Chairman; T. J. Parker, Secretary; Virgil Coblentz, E. G. Love, H. Schweitzer and R. C. Woodcock. The committee has issued the preliminary programme and expresses the hope that as many American members as possible will attend and be present when the President, Dr. William H. Nichols, of the General Chemical Co., of this country, delivers his annual address.

The following programme has been provisionally arranged:

During the first week in London, the annual meeting will be held, and the address of the President, Dr. William H. Nichols, will be delivered. Various excursions and visits in and around London are to be arranged.

During the second week visits will be made to places of industrial and historical interest in England, the guests being in charge of the provincial committee. Three separate excursions are contemplated, the first to places of historical interest (cathedrals, chiefly on the East Coast of England; Ely Lincoln Repose, Fountains Abbey, York, Durham); the second to the factories of the textile industries of Bradford, Birmingham, Nottingham, Burton-on-Trent, etc.; the third to chemical works, dye and print works, cotton mills and potteries in the vicinity of Liverpool and Manchester.

These visits will be so timed that the whole party will finally meet in Glasgow, where the Scottish Section will take care of the visitors and where the meeting will terminate.

The London committee has extended a most cordial invitation to ladies, and a ladies' committee has been formed, including those ladies who were present at the annual meeting in New York.

Electrolytic Separation of Metals.

To the Editor of Electrochemical and Metallurgical Industry:

Sir:—In comparing the metallurgy of different metals, there are always found certain types of reactions that resemble each other. It is often quite possible to formulate a general principle that will describe the class to which these similar reactions belong. The way is first to recognize the similarity by intuition, and then by analysis to separte the essential points. Synthesize these and you have a general law.

In this connection two general principles are seen applying to the separation of metals by their selective electrolytic properties.

One general principle applying to the separation of a negative metal from a more positive one is by "acid electrolysis." For instance, to separate iron from copper in solution it is only necessary to have the concentration of the hydrogen as "ionic hydrogen" strong enough so that no iron can be deposited at the cathode under ordinary current densities.

In other words, the presence of 2 or 3 per cent of mineral acid will keep the "electrode potential" so low that only copper and a small amount of hydrogen is deposited. This is done in the case of iron and copper and nickel and copper in the electrolytic copper plant. The presence of the hydrogen ions keeps the iron or nickel in solution for the above reasons. The hydrogen ions act like a safety valve on a boiler, and prevent the cathode potential rising above the low limit, provided that the circulation of the catholyte is vigorous enough to neutralize the concentration changes set up by the electrolysis. This has been tried with many pairs of metals in analytical electro-analysis and in practical electrometallurgy.

A second general principle governing the electrolytic separation of metals is that of selective electrolytic dissolution of a composite metal anode. This is not always complete and can be followed by a cementation process in an outside vat.

For instance, if the anode consists of 90 per cent M^+, and 10 per cent M^- at low rates of dissolution the cathode will be quite pure. However, if the current density be increased, the less positive metal will be dissolved and precipitated on the cathode in considerable amounts. But if the electrolyte be subjected to such a system of circulation that before much of M^{--} reaches the cathode, it is removed to another system, it is possible to remove M^- from the process by the reaction: $M^+ + M^- x = M^+ x + M^-$. Thus the electrolyte is kept in a continued state of purity.

It is, of course, necessary to have all the conditions right for the rapid and complete metallic precipitation above indicated. This is a question of "physical metallurgy," if a new phrase can be coined on the analogy of the phrase "physical chemistry." Such a reaction effectually stops any deposition of M^- at the cathode.

This set of reactions has been made use of in several processes. It has been the practice to run the copper electrolyte over shot copper. A German electrolytic zinc-silver process (C. F. Schnabel, *Handbook of Metallurgy*, Vol. II, chapter on electrical processes) is another example. I found it quite useful in some work I did a few years ago in the electrolytic refining of iron.* It is of general application and can be of value to many other separations of metals by electrical methods.

Woolsey McA. Johnson.

New York City.

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* U. S. Patent No. 780,191, Jan. 17, 1905.

Commercial Possibilities of Blast Furnace Gas for the Development of Electric Power.

By F. duP. Thomson.

Not until the Lackawanna Steel Company decided upon the use of gas engines for the development of electric power and for the compression of the blast with gas obtained from the blast furnace itself, did any iron manufacturer in this country take the initiative and install even a trial engine constructed for this special class of service. To those European iron masters who have demonstrated the feasibility of the system we are indebted for a long step forward in blast furnace economy, a step which has made possible the development of fully one million horse-power over and above the requirements of the blast furnaces themselves, when the output of this country is taken as twenty million tons of iron per year. Despite discouraging reports as to the results of European installations, made by American investigators, the blast furnace gas engine industry continues to grow.

When the first installation was made at Buffalo great activity prevailed in the iron trade. During the years 1899 to 1903 the volume of business was so great and prices so remunerative as to give no good reason to consider the expense of replacing equipment which answered the purpose, for money employed as working capital yielded greater returns than if invested in equipment which might be operated with greater economy than that in existence. With business on somewhat closer margins and the present necessity of replacing or supplementing plant, consideration of gas engines using blast furnace gas is more serious.

Competition has sprung up also in the manufacture of large gas engines. The performance and detail construction of the engines offered for sale to-day is subjected to much closer scrutiny than usual in the purchase of steam engines for the same duty. At present, there are in this country no less than six firms, to my knowledge, who will undertake to furnish gas engines in sizes varying between 2000 hp. and 3000 hp.

A profitable market for power is essential before we may consider at all the use of surplus blast furnace gas for the development of power. Without such a market, the replacement of steam blowing engines by gas-driven blowing engines, in order to make a larger surplus available, should not, at present prices for equipment, be considered. If, for example, a blast furnace plant is not operated in connection with steel works nor in the vicinity of works which do or may use large quantities of power, no use could be found for the surplus gas which would be made available by the installation of gas blowing engines. In such a case, there would be no good grounds for recommending gas engine blowing equipment, unless the cost were less than that of the equivalent steam blowing equipment, including boilers. If, however, a market is within economical transmission distance, and other conditions are not prohibitive, very satisfactory returns may be expected.

An important and, in most cases, essential condition with which a power plant must comply, is that of uninterrupted service. The power plant which we consider here depends for its supply of gas upon the blast furnace, the heir to many ills. It may be put out of blast because the lining is worn out, or banked because there is a shortage of coke or other raw material, or it may be banked or put out of blast because of trade conditions, or on account of labor troubles, or there may be a stoppage of greater or less duration due to accident or trouble in the working of the furnace, and in its regular operation as well, the result of which is a suspension of its function of gas producer.

To secure, within a reasonable degree of certainty, a continuity of supply, no group of less than three blast furnaces can be considered as available to supply a power plant, which is intended to produce power for the general market. At any group of furnaces at least one will at some time be out of blast for relining, and at least one other furnace may be stopped for some one of the other reasons given above. With a group of three blast furnaces, therefore, the surplus gas of one will always be available.

With groups of more than three, still greater security against interruption would result, if we assume one furnace out of blast and two temporarily stopped. The percentage of surplus gas actually available from any group of N furnaces may be expressed by the formula,

$$\frac{N-3}{N} \times 100$$

in which, if values of "N" from 4 to 11 be substituted, the curve shown in Fig. 1 will be obtained. For the special case of a group of three furnaces, the surplus available becomes 33 per cent. To be even more conservative, two furnaces might be assumed as out of blast for relining at the same time, but with a power plant dependent upon the blast furnace plant, repairs of this kind can be so timed that it will not be necessary to reline two furnaces at once.

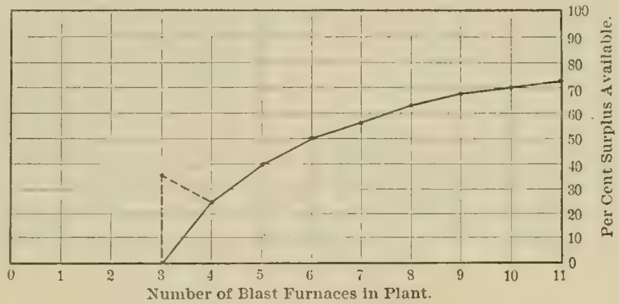


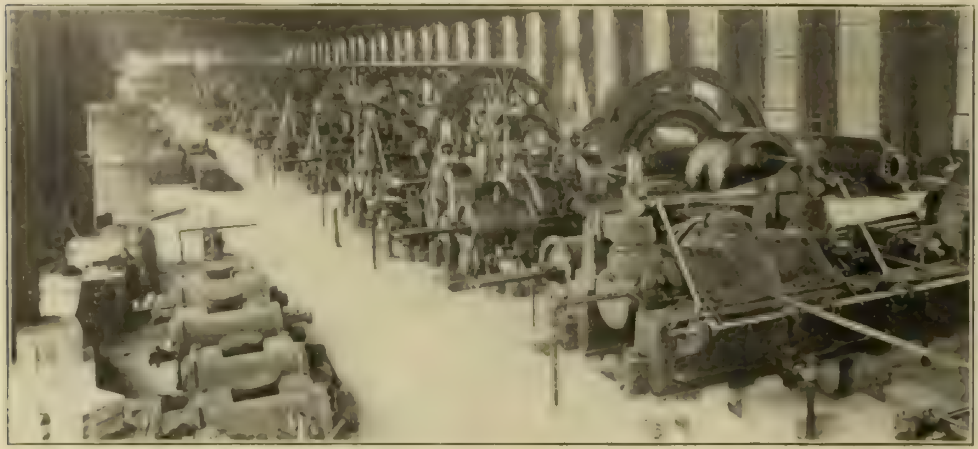
FIG. 1.—SURPLUS GAS AVAILABLE WITH DIFFERENT NUMBERS OF BLAST FURNACES.

The value of a large number of units to obtain a given output of iron is very evident, if the development of a large percentage of the surplus gas into power is to be made possible. Increased cost of construction and operation of a blast furnace plant, made up of a number of small units, as compared with the same items of expense for a plant of the same capacity made up of large units, must be more than balanced by the profits which are to be realized from the operation of the larger power plant. Further than this, the prices obtained for the power must warrant the continuance in blast of all but one furnace of a blast furnace plant, when trade conditions would imperatively dictate a suspension of operation. The profit must at least pay the interest on the labor and material invested in manufacturing periods in which there is no market for the product and at least part of the loss due to a falling market.

With proper foresight in location and intelligent design we can eliminate from water-power plants every probable cause for a suspension. The labor question enters into it only to a very small degree. It looms up larger in the steam and gas producer power plants, owing to their dependence upon a supply of coal, into which the labor element of mining and transportation enter. The blast furnace gas power plant, however, is burdened with the responsibility of maintaining operations in spite of trouble which may occur at ore, stone and coal mines, coke ovens, along the lines of transportation, and finally at the blast furnace plant itself. Nor is this minimized to any great extent by the large stock carried in the furnace yards, for the troubles may occur at the end of the winter when the stock are low and no estimate, with any accuracy, can be made as to how long the trouble may last.

To meet such an emergency a gas producer plant might be installed. Even this would be but a partial remedy, and would involve an additional yearly charge for power of not less than [illegible] per hp. or [illegible] per kw., not including the cost of any coal which may be consumed by such a plant to maintain at least a portion of it in readiness, or that required during such periods that the blast furnaces are unable to supply gas.

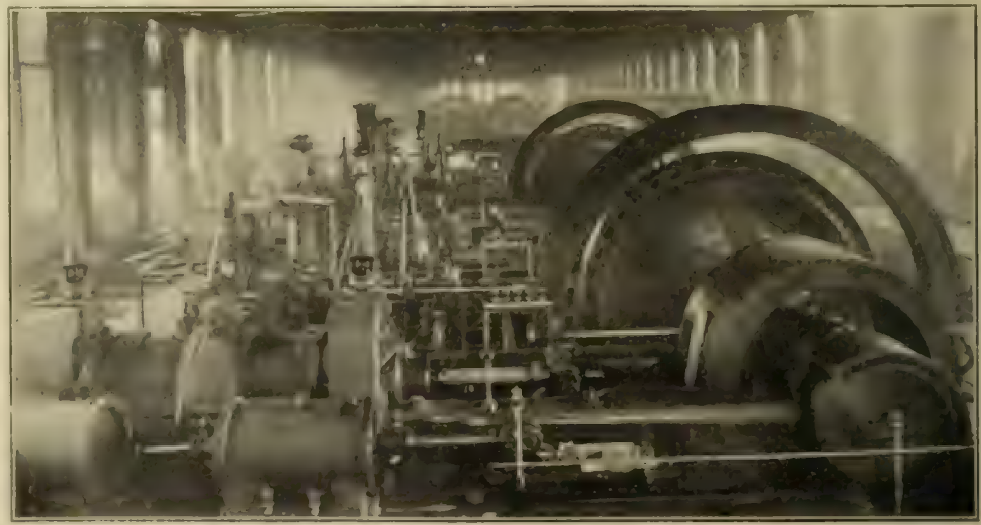
Without wishing to discourage the idea of building commercial power plants in connection with blast furnace plants, I believe it better to point out as clearly as possible all the disadvantages in the first instance, and leave to the decision of the investor whether the possibilities, as I shall attempt to show them, sufficiently outweigh the disadvantages, to encourage him to venture capital in this rather than some other undertaking.



BLAST FURNACE GAS ENGINE POWER PLANT, CAPACITY 9000 HP.

The amount of surplus energy and the proportion available for the development of power, is to be determined. The waste gases of the blast furnace contain about one-half of the total heat originally possessed by the fuel, and is approximately in direct proportion to the coke consumption per ton of iron. It has been variously estimated that from 18 per cent to 33 per cent of this heat is required to heat the blast. Not only have no accurate determinations ever been, to my knowledge, made of the gas consumed in hot blast stoves, but the heat which is to be imparted to the blast varies with local conditions, even when the final temperature is identical.

The distance between engines and stoves, the size of mains, the efficiency of burners and of the heating apparatus itself, materially affect the heat consumption. Differences in blast furnace practice result in different pressures, and the range in tem[illegible] is by no means inconsiderable. A conservative estimate of the percentage of the heat of the waste gases to heat the blast would be 30 per cent, and one which would not, in general practice, be exceeded. For the compression of the blast by steam engines, giving an economy of 20 pounds of steam per hp. per hour when run condensing, under normal conditions, with boilers which convert 60 per cent to 70 per cent of the heat of the gas burnt into heat of steam, requires from 33 per cent to 40 per cent of the heat of the waste gases. For auxiliary purposes, such as pumps, light, power for hoists and ore-handling machinery, approximately 7 per cent to 10 per cent is required. By the substitution of gas blowing engines and gas engines for the production of power for auxiliaries, the figures above may be reduced to read 16.5 per cent to 20 per cent and 3.5 per cent to 5 per cent, which, for clearness, are summarized in Table I.



BLAST FURNACE GAS ENGINES NOS. 6 AND 7 OF 9000 HP. PLANT

Absence of arms of fly-wheels indicates both engines in operation. Engine No. 7, in foreground, is the [illegible] gas engine [illegible] and operated in this country, first started January 2, 1903, and in regular service ever since.

The wide difference between the surplus available under the head of steam engines, is due principally to differences in efficiency of apparatus and only in a limited degree to differences in practice. The difference in surplus available under the head of gas engines, depends almost entirely on differences in furnace practice, for the efficiency of the gas engine equipment is in each case the same. The maximum surplus with steam equipment is, curious to say, identical with the minimum to be expected with gas engine equipment.

The superiority of the gas engine over the steam engine may appear rather small, when the figures given in Table I are compared with those frequently seen as the results of published tests. It must be remembered, however, that the gas blowing engine operates normally at only about two-thirds of the load which it must be designed to carry when the blast pressure reaches 25 pounds to the square inch. Such periods of high pressure may be coincident with a poor quality of gas, making the conditions even worse than indicated above. The gas engine in such service is at a disadvantage compared with a steam engine, designed to operate with

Table I.

Use.	Steam Engines.		Gas Engines.	
	%	%	%	%
Heating the blast	18	to 33	18.0	to 33
Compressing the blast	33	to 40	16.5	to 20
Auxiliaries	7	to 10	3.5	to 5
Surplus available	42	to 17	52.0	to 42
	100	100	100	100

economy at normal load, with capacity, however, to supply air at the highest pressures met with in general practice.

Large gas engines are built to-day, which will, at full load, give a duty of 194,000,000 effective foot pounds, as compared with a duty of 82,000,000 effective foot pounds, from one of the best steam engines per 1,000,000 B. T. U. in the gas consumed in each case. As the economy of the gas blowing engine will approximate only 80 per cent of the figure given above, it may readily be seen that its ultimate economy, in blowing engine practice, may be safely said not to exceed twice that of the steam engine.

The actual amount of power available from the surplus gas developed by gas engines will vary with the coke consumed per ton of iron and the efficiency of engine and load factor. The coke consumption per ton of iron may be within a comparatively short time reduced to an average figure of 1700 pounds, as the result of the method which has recently been developed to remove the moisture from the blast. On the other hand, where the mixture is poor and other conditions unfavorable, the coke consumption may be as high as 2800 pounds per ton of iron. There will be present in the waste gases for the one case 9,000,000 B. T. U., and for the other, 18,000,000 B. T. U. per ton of iron.

The average of present practice in coke consumption is probably very close to 2240 pounds per ton yielding 14,000,000 B. T. U. in waste gases. We may reasonably expect a reduction of this figure to that of 1700 pounds, as stated above, but, on the other hand, improvements in engine and stove economy, which are to be anticipated, may permit the use of the larger figure given in Table I as the percentage of surplus gas available with gas engines. The amount of surplus heat available for power per ton of iron may, therefore, be taken as 52 per cent of 9,000,000 B. T. U. or 4,680,000.

PORTION OF SWITCHBOARD FOR 900-HP. POWER PLANT SUPPLIED WITH BLAST FURNACE GAS ENGINES OF 1000 HP.

From this amount of heat it is possible to develop 468 hp. per ton of iron made per hour, with gas engines, which have a thermal efficiency per effective hp. of 25 per cent, or can produce one effective hp. per 10,000 B. T. U. per hour. Manufacturers of gas engines are willing to guarantee this result when their engines are operated at full load. For a blast furnace making 300 tons of iron per day, we may rely with certainty upon obtaining not less than

$$\frac{300 \text{ tons}}{24 \text{ hours}} \times 468 \text{ hp.} = 5850 \text{ hp.}$$

or let us say 6000 hp.

I shall not attempt to discuss here the details of engine construction nor any of the claims to excellence of the several makes of large gas engines, for these have been quite fully set forth in periodical literature, as well as the catalogues of the builders.

The subject of gas cleaning, however, is of great importance and merits some attention at this point. It is now generally conceded that blast furnace gas must be cleaned before use in the gas engines; if for no other reason than that the cleaning process, at the same time, reduces its temperature and thus increases its density, thereby increasing the power available from a cylinder of given dimensions. Whether cleaned by transmission through great length of pipe at low velocity, or by contact with sprays or surfaces of water, the temperature is lowered. Cooling and cleaning by the dry or transmission method is not satisfactory, and becomes very costly if a temperature below 120° F. is desired. Nor do conditions of velocity, satisfactory for cooling, permit the settling of the dust, and the finest particles, when dry, require practically absolute rest, which is, of course, impossible. Water cooling and washing is now generally employed.

Electrostatic methods of cleaning have been suggested and experimented with, but so far as I am informed no practical results have as yet been obtained. In any event the gas for gas engines must be cooled. Cooling not only increases the density of the gas itself, but removes by condensation the moisture which is brought over with the gases from the ore, limestone and coke amounting to between 3 and 5 per cent by volume, reducing the heating value of the gas to that extent. The temperature of the gas must be lowered as much as the temperature of the water available, and the quantity which it is practicable to use, will permit, so that the gas at this temperature, when saturated, shall not contain more moisture than previous to treatment.

Various means have been employed to produce an intimate mixture of the cooling water and the dust-bearing gas. At first the old coke scrubber of coal-gas works practice was used, consisting of a vertical pipe filled with coke through which water trickled from above, and gas was made to pass from below. In order to overcome the resistance to the passage of the gas through the coke, either a steam jet blower or centrifugal fan was used. It was found that the dust adhered to the blades of the fan, and to avoid frequent stoppages for cleaning, water was introduced at the center of the fan to wash the dust away. Not only did this water keep the fan clean, but being dashed into fine particles by the action of the arm and blades, became intimately mixed with the gas and dust, reducing their temperature. The action of centrifugal force, assisted also to separate the particles of water and moistened dust from the current of gas. The mixture of water and dust was very readily flushed out of the fan. We thus owe to accident the most effective form of gas washer that has yet been devised.

Improvements have been made in details, making it more efficient, but in essentials the apparatus as first used remains to-day unchanged. More or less fine spray or mist of water is delivered by the fan with the gas and some form of separator, either of a type similar to steam separators and exhaust steam heads, or else a dryer in the form of a saw-dust box is placed in the gas main between the fan and the engine to remove it. The presence of mist in the gas is objectionable, because it may accumulate in the gas mains, freeze in winter or at any time it may be carried into the motor cylinder, cause difficulty in ignition, or absorb a certain quantity of latent heat when converted into steam by the combustion of the gas with which it is mixed.

The equipment of fans required and the consequent cost of power may be materially lessened by the introduction of precooling devices, relying on the fans principally for the separation of the water and dust from the gas. These pre-coolers may take the form of horizontal pipes, equipped with water sprays or vertical pipes or towers similarly equipped and arranged so that the gas moves upward against the sprays. The gas in this way not only comes in contact with each successive spray, but by the action of the drops of water, owing to the longer time which they are exposed to the gas in falling through it, reduce the temperature and moisten the dust much more effectively in a vertical than in a horizontal spray pipe.

The action of the sprays, however, is to beat the gas back so that as a matter of fact the fan motors must exert some power to draw the gas through the towers. This fact is evidenced by an inflow of air through any opening at the top of the tower, where, if there were any of the pressure such as exists at the top of the blast furnace, there would be an outflow of gas. The expenditure of power to draw the gas through the tower is less than that required to accelerate the gas and water required for cooling in a fan, should the pre-cooling towers be omitted. The water consumption is approximately 0.06 gallons per hp. per minute, and the fan, motors and pumps take about 1.3 per cent of the rated hp. of the gas engine installation which they serve.

The degree of purity necessary will depend to some extent upon the type of engine in which it is to be used. The two-cycle gas engine equipped with gas pumps provided with piston valves must be supplied with purer gas than four-cycle engines which take their supply through the motor cylinder only. Entire exemption from troubles incident to the presence of dust in the gas may be avoided, if the purity is so high that the gas contains no more than 0.025 grams per cubic meter. A very simple apparatus has been devised, which has made possible accurate and rapid determinations of the quantity of dust present in the gas. It is highly important to determine the quantity and character of the dust, and to study carefully in each case all the local conditions before any attempt is made to design a gas-washing plant.

Any variation in the composition of the gas affects, to some extent, the regulation. Especially is this true when this change in composition consists in a sudden increase in the percentage of hydrogen, due to a leak in some cooling device which permits water to enter the furnace. At present there are no means known by which the power plant may be warned of its approach in the current of gas. By means of the composimeter changes in the calorific value of the gas, other than those due to hydrogen, are at all times indicated, and the power house would in most cases have time to start up a reserve unit when the calorific value of the gas falls to a point below which the full power of the engine cannot be developed.

Without uniformity of gas pressure, regulation sufficiently close to insure successful operation in parallel of alternating-current generators, is extremely difficult, if not impossible. The use of a gas holder affords a simple means of maintaining a constant pressure, and provides some little storage room as well. Changes in the operation of the blast furnace and variations in the consumption of gas by the engines are too frequent and rapid to permit at present of any satisfactory method of regulation at the fan. The cost of the gas holder of sufficient capacity to maintain even for one hour a supply of gas for the plant at full load, when all the blast furnaces are stopped, would not, except in special cases, be warranted, and in general the use of a large gas holder is unnecessary.

In order to give the power proposition every benefit possible, we shall consider only the largest units at present obtainable. I have elsewhere stated that builders are prepared to furnish and guarantee the successful operation of gas engines ranging between 2000 hp. and 3000 hp. per unit.

The cost of plant and cost of power must be based on certain assumed conditions of service and load factor. We will assume, for example, that the character of the service which the power plant is to supply will involve suddenly applied though possibly momentary overloads of 50 per cent, and that the load factor is 50 per cent. As the gas engine has no overload capacity, or stated differently, we may say, that full load is the limit of its overload capacity, the nominal full rated load must be two-thirds of the actual full load capacity, for the case which has been assumed. A gas engine of 2000 effective hp., coupled to a 1000-kw generator, capable of momentary overloading to 50 per cent, would be a suitable combination. This class of service and type of unit will be designated as Class "A."

For service in which there is never any overload, or where the full load of the generator is but 80 per cent of the engine capacity and the generator overload never excedes 25 per cent, [illegible] coupled to a generator of 1500 kw. would be a satisfactory unit, and we may designate this type and service as Class "B."

In either class the space occupied would practically be the same and require for each unit a length of 50 feet in a building 75 feet wide, together with some additional length to house the compressors, pumps and exciters which go to complete the installation. No proposition, except one which utilizes all the surplus gas upon which we can depend, need be considered here. Nor shall we consider blast furnaces producing less than 300 tons per day each; but this must not be taken to pre-

TABLE II.

Cost of Power Plant.

Data.—Gas engines in both classes are 2000 hp. each. 75 r.p.m. One unit in each case in reserve.

Items.	Class "A" 4 1000 kw. Unit, 50% Overload Capacity.	Class "B" 4 1500 kw. Units, No Overload Capacity.
Land	$ 5,000	$ 5,000
Building	67,500	67,500
Engine Foundations	6,000	6,000
Crane—Travelling	8,000	8,000
Piping	16,000	16,000
Compressors and Engines	8,000	8,000
Pumps and Engines	12,000	12,000
Exciters and Engines	15,000	15,000
Switchboard—Wiring—Lamps	16,000	16,000
Generators	100,000	120,000
Gas Engines	280,000	280,000
Exhaust Stack	4,000	4,000
Gas Washing Plant	30,000	30,000
Gas Holder and Piping	10,000	10,000
Office and Fixtures	2,500	2,500
Total Cost	$580,000	$600,000
Cost per kw. of Total Station Capacity	145.00	100.00
Cost per kw. of Station Capacity Operated	193.33	133.33

TABLE III.

Cost of Power at Station Switchboard.

Data.—Gas engines in both classes are 2000 hp. each. 75 r.p.m. One unit in each case is in reserve. Value of blast furnace gas not considered.

	Class "A" 4 1000-kw. Units, 50% Overload Capacity. Load Factor 50% 24 Hour Power		Class "B" 4 1500-kw. Units, No Overload Capacity. Load Factor 80% 24 Hour Power.	
	Per Annum Amount Dollars	Per kw. Hour Cents	Per Annum Amount Dollars	Per kw. Hour Cents
Superintendence and Wages	$20,450	0.156	20,450	0.065
Supplies, including water	18,800	0.143	18,800	0.0597
Repairs and maintenance	11,600	0.0885	12,000	0.0382
Taxes and Insurance	8,700	0.0665	9,000	0.0286
Capital Charges:				
5% on Investment	29,000	0.578	30,000	0.248
8% Sinking Fund	46,800		48,000	
Total Cost	135,350	1.0320	138,250	0.4395
Cost per kw. year of station capacity operated		$45.12		$30.75
Cost per kw. year power generated		90.24		38.40
Assuming profit of 10% on Investment	58,000		60,000	
Total Price	193,350	1.475	198,250	0.630
Price per kw. year power generated		$128.70		$55.15

clude consideration, except in this particular case, of such a project for a group of smaller blast furnaces. The estimate here given is on a power plant in connection with a group of either three or four 300-ton furnaces, in which the surplus gas from one furnace only may be utilized.

The Tables II and III, on page 98, give in detail the estimated cost of installation and cost of power at the switchboard, and are based on actual experience which the writer has had in the construction of such plants. Sufficient allowance has been made to insure against variation in price and efficiency of labor and cost of materials. In Table III the annual amounts have been given, as well as the kw. rate, so that the reader may make his own assumptions as to the total current generated, and find the total cost per kw.-hour. The only variable is "Supplies." Unless the load factor is so low as to permit of stopping one of the three units which have in calculations been assumed as always in service, the value of "Supplies" will not vary appreciably.

The cost of installation compares favorably with that of the best steam plants, and we may expect to better the figures given. We cannot, however, look for any great reduction, and as the cost of operation is, to the extent of 70 per cent, made up of charges based on investment, we may say that the cost of $38.40 per kw.-year at the switchboard is susceptible of little, if any, improvement. By dropping the sinking fund rate to 2 per cent, the cost could only be reduced to $35.00.

As stated in Table III, no allowance has been made for the value of the blast furnace gas. If the surplus were not thus utilized for the development of power it would be lost. If the same amount of heat which is supplied by the blast furnace gas were obtained from coal at $2.00 per net ton or natural gas at 7 cents per 1000 cubic feet, the cost of fuel for the gas engine per kw. per year would be approximately $12.50 in either case. For the steam engine the expense would only in rare instances be less than $25.00.

The value of the surplus gas may also be taken as the sum of the capital charge of 5 per cent and the profits, amounting in Class "A" to $87,000, and in Class "B" to $90,000 per annum for a blast furnace plant, of which the yearly output is 300,000 tons of iron. An increase in profits to the iron manufacturer of approximately 30 cents per ton may thus be realized. If this is true of a plant which can convert but 25 per cent of the surplus gas available, still better results may be expected from those at which the percentage of surplus convertible, as indicated by Fig. 1, is greater. It would not be impossible to realize 75 cents or more per ton, where 8 to 10 furnaces are grouped. No management can well afford to neglect a very thorough investigation of the conditions at his plant and his locality to determine whether they are favorable to the development of power from surplus gas.

Wherever there is a large demand for power and cheap water-power is not available and blast furnaces are, the blast furnace gas engine power plant need fear no rival. We are not far from the day when the power house will be as much a part of every modern blast furnace plant as are the condenser and ammonia houses of the modern by-product coke plant.

RADIUM AND TERRESTRIAL HEAT.—C. Liebenow, in a paper in *Physikalische Zeitschrift,* vol. 5, No. 20, page 625, points out that a relatively very small quantity of radium in the earth would be sufficient to keep the temperature in it constant. The heat given off per second by the surface of the earth is about 10,000,000,000 kilogram-calories. Since 1 gram of radium produces 226 gram calories per hour, the earth cannot contain more than 200,000,000,000 kilograms of radium, as otherwise its temperature would continually rise. If this quantity was uniformly distributed throughout the earth, one cubic meter would contain 0.0002 milligram. Elster and Geitel have found a thousand times this amount in the soils investigated by them. The radium must therefore be distributed nearby the surface, about 0.4 gram per sq. meter.

Electrochemical Reactions in Solid Electrolytes.

BY PROF. F. HABER AND H. TOLLOCZKO.

Our knowledge and mastery of the processes in the electrolysis of solid bodies is very poor. That solid substances conduct currents, especially when they are heated, is well known, and there is no doubt as to the electrolytic nature of this conductivity, for chemical changes at the poles have often been observed qualitatively when a current passes through solid salts and the occurrence of counter electromotive forces (polarization) under these conditions has been proven. Such observations are found in the literature of the subject, partly in earlier times, and partly recently, in connection with the well-known technical results which have been brought to light through the application of solid electrolysis for lighting, in the Nernst lamp[1]. Quantitative study of the electrochemical changes in solid electrolytes, however, has hitherto been successfully carried through only in those cases in which it was possible by some special device to keep the electrolyte unchanged. Warburg achieved this by using ordinary heated glass as electrolyte, between an anode of sodium amalgam and a cathode of mercury. The glass remained unchanged, both under the microscope and to the naked eye, while a quantity of sodium, corresponding to Faraday's law, passed through it.

We have investigated this subject more closely, devoting ourselves especially to the electrolysis of solid barium chloride at about 400° below its melting point; that is to say, about 500° to 600° centigrade.

Barium chloride melts readily a little below 1000° to a clear liquid. For containing vessels we used nickel crucibles of common form (dia. 3.5 cm. at the top), which were connected with the positive pole of a source of current; while for counter-electrodes sticks of graphite, iron wires, or platinum wires, were dipped into the melted mass. Then we let the melt harden and cool off. Later we heated again to 500° or 600°, and this time passed the current through. We worked with current intensities which seldom exceeded 50 milliamperes and required pressures which seldom reached 20 volts, and for the most part amounted to less than 10 volts. The quantity of electricity used was measured with the copper coulometer (we use this expression, following the usage of Th. W. Richards, instead of the older designation "copper voltameter"). The counter electromotive force of polarization could be measured by means of the compensation method, after breaking the current. The heating of the crucible traversed by the current took place either by means of a little gas furnace, or in an electrical muffle. Particulars may be found in a detailed report, in the *Zeitschrift für Anorganische Chemie* (Vol. 41).

At the anode, that is, at the inside wall of the nickel crucible, nickel chloride ($NiCl_2$) was formed, in the anhydrous condition; as the electrolysis continued, it slowly grew into the interior of the solid barium chloride. Fortunately, it formed no dendrites, but penetrated only slightly and uniformly towards the interior of the mass of barium chloride. After the end of the electrolysis, when the block of barium chloride had been removed from the crucible, this layer containing the nickel could be easily scratched off. Then the block consisted of barium chloride, in which the cathodic products were enclosed.

One might at first suppose that barium would have gathered at the cathode. If this had been the case, then the polarization must have amounted to about 2.65 volts. For in this case, all the substances are in the solid condition, so that Thomson's rule may be applied to calculate the electromotive force from the heat reaction. With the help of this rule,

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[1] See e. g. Hittorf. Pogg. Ann., 84 (1851) 1, Bose Druder's Ann.,9 (1902) 164, and summary of the literature in Wiedemann, Die Lehre von der Electricität, II. edi. Braunschweig, 1903, Vol. I., p. 545, and Vol. II., p. 491.

[illegible] it is found from thermochemical data that 2.05 volts [illegible] for the reaction

$$Ni + BaCl_2 = NiCl_2 + Ba \qquad (1)$$

But the polarization turned out to be considerably higher at the moment of breaking the current, and sank rapidly, without stopping at 2.05 volts, till it reached about 1.9 volts, where it remained steady for a long time. This behavior is readily explained when it is remembered that Guntz recently has shown that barium subchloride is formed from barium and barium chloride. The reaction follows the equation:

$$Ba + BaCl_2 = 2BaCl \qquad (2)$$

Barium in the presence of barium chloride is therefore unstable, and changes into the subchloride. It behaves, therefore, similarly to iron in contact with ferric chloride solution, or tin in contact with stannic chloride solution. In such a case theory[2] permits the prediction of certain values of the counter electromotive forces before and after breaking the current. As long as the current is flowing, and barium is present in addition to barium subchloride, the counter electromotive force of the reaction:

$$Ni + 2BaCl = NiCl_2 + 2Ba \ldots\ldots\ldots\ldots\ldots(3)$$

has to be overcome. As soon as the current stops, barium is consumed by the barium chloride according to reaction (2). After this there is only barium subchloride and barium chloride in contact with the electrode and the counter electromotive force corresponds to the equation:

$$Ni + 2BaCl_2 = NiCl_2 + 2BaCl \ldots\ldots\ldots\ldots(4)$$

It is easy to see that the electromotive force for reaction (4) must be less than that for (1), by just as much as the electromotive force for (3) exceeds that for (1). Now, we found, as mentioned above, an electromotive force of about 1.9 volts after breaking the current. We conclude from the considerations just adduced that this electromotive force must correspond to the reaction (4). Then reaction (3) must have the electromotive force 3.40 volts. The values of the polarization which are measured immediately after breaking the current will come the nearer to this value, the slower the reaction (2) takes place. In order to retard (2), we made use of the device of lettting the temperature sink during the electrolysis. In this manner we reached a point where we could observe 3.26 volts of counter electromotive force immediately after breaking the current.

This value is as near to the calculated amount of 3.40 volts as can be expected, in view of the uncertainty in the thermal data on which the calculation is based, and in view of possible small deviations from Thomson's rule.

It must accordingly be concluded that in the electrolysis barium was produced, which soon united with the barium chloride to form barium subchloride. More exact discussion of the numbers given shows that the subchloride must be able to decompose water with great activity, producing barium hydroxide, barium chloride, and hydrogen. It is therefore not in the least strange that the production of the subchloride in aqueous solutions does not succeed any better than the electrolytic separation of aluminium from aqueous solutions of salts of alumina, or the separation of free fluorine from aqueous solutions of fluorides. On the other hand, this offers the possibility of determining the quantity of barium subchloride produced by the current, if quantative measurement is made either of the hydrogen or of the barium hydroxide produced when the mass is dissolved in water. The determination of the hydrogen is of use only when care is taken to prevent the access of oxygen to the subchloride, both during the electrolysis and subsequent operations. The moisture of the atmosphere must also be carefully avoided. The quantity of barium hydroxide, which is found after dissolving the mass in water, must, however, always permit a conclusion to be drawn as to the amount of BaCl produced by the current. Now we have been able to show that the quantity of this barium hydroxide came extraordinarily near to the theoretical value, so that we may say: *The formation of barium subchloride at the cathode took place according to Faraday's law.*

We were able to bring about a different action at the cathode, by adding barium carbonate to the barium chloride during the melting. In this case carbon appeared at the cathode, in quantity corresponding to Faraday's law. This carbon formed dendrites, which grew upon the cathode in the direction of the lines of flow of the current. It was easy to separate it by dissolving the melt and collecting the undissolved black residue. The carbon could be determined quantitatively, both from the loss of weight in heating this residue to redness, and from the carbon dioxide produced in its elementary analysis. The counter electromotive force, when carbon is produced, can be calculated by means of Thomson's formula, from the heat of the reaction:

$$2Ni + BaCO_3 + 2BaCl_2 = 2NiCl_2 + C + 3BaO$$

It comes out about 1.6 volts. We found, in agreement with this, that the polarization after breaking the current does in fact reach a stable value between 1.5 and 1.6 volts.

The action when carbon is produced at the cathode may be conceived as consisting at first of the production of barium, which then reacts according to the equation:

$$2Ba + BaCO_3 = 3BaO + C$$

In order to justify this equation, we have demonstrated that sodium readily separates carbon from barium carbonate when the two substances are heated together. This is an experiment which is quite simple to carry out. We arranged it as follows: Barium chloride and barium carbonate were melted together in a porcelain crucible, and during the cooling the crucible was rocked. In this manner the inside of the crucible was coated with a protecting layer of the carbonate and chloride of barium, which formed a hollow. In this hollow we laid a piece of sodium, covered it with barium carbonate, and heated with a small gas burner, at the same time passing nitrogen over the top, to avoid oxidation. After a few minutes the sodium disappeared, and a considerable quantity of carbon was separated.

If pure barium chloride is melted in the nickel crucible, and electrolysis is carried out, after it has become hard, then also carbon will be found at the cathode, in case the melting took place in a gas furnace. The reason for this is that the combustion gases change melting barium chloride into barium carbonate, with loss of chlorine. This remarkable reaction, to which we have devoted special experiments, takes place according to the equation:

$$BaCl_2 + O + CO_2 = BaCO_3 + Cl_2$$

Melting calcium chloride shows the same reaction, to an even greater degree; and even in the case of common salt we were able to demonstrate a slight formation of sodium carbonate and chlorine gas under similar conditions.

We have been describing a few cases of solid electrolysis, which, so far as we know, are the first of their kind. To supplement the observations made in this connection, we have carried out a few experiments with reference to the electromotive forces of electric cells consisting entirely of hot solid salts and solid metals. A good example of such a battery is:

Lead—Lead chloride. Silver chloride—Silver

This battery can be easily investigated between 150° and 250° centigrade. Its voltage was near to that calculated from Thomson's rule for the reaction:

$$Pb + 2AgCl = PbCl_2 + 2Ag$$

But the temperature coefficient turns out not to be zero, but has a small measurable value. This is related to certain theoretical considerations, regarding which we refer to the detailed publication in German, cited before. Our experiments on this branch of the subject are not yet completed.

We believe that the electrochemical phenomena described in this article show that the electrolysis of solid substances deserves greater interest. We have very few other ways to study the reactions between solid substances. And these reactions deserve special attention, because the mass actions which are inseparable from the reactions in the dissolved, melted, or gaseous states, either disappear in the case of solids, or else assume special and interesting forms. Also, as the example of the electrolytic separation of carbon shows, we may expect some results which have hitherto not been attained in other ways.

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[2] [illegible]

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Equipment of a Laboratory for a Smelting Plant.

In a paper presented at the Lake Superior meeting of the American Institute of Mining Engineers, Mr. **Herbert Haas** describes a laboratory for metallurgical chemistry and technical analysis, which he built late in 1903 for a pyrite smelter at the Afterthought mine, Ingot, Shasta County, California.

A works-chemist, having the shortest time in which to make his determinations, should have a laboratory arranged as conveniently as possible to save time in carrying out his work. It is not uncommon at large smelters to have the night-assayer prepare thirty or more samples for titration in time for the chemist to begin work on them at 7 o'clock on the following morning. During the time that the solutions are being heated on the hot-plate, the chemist is weighing out portions of slag-samples which have been taken from each furnace. As a rule, he determines the various percentages of silica, iron and lime in the slag (the values in lead and silver, and sometimes that of gold being determined by the assayer), but very frequently the slag is analyzed for other constituents as well; and every fourteen days a complete analysis of it is made. The chemist cannot give his entire time to a single analysis until it is completed, for the reason that a dozen or more additional samples come in regularly which have to be analyzed for "insolubles," silica, iron, lime, zinc, sulphur, lead, copper, alumina, manganese, arsenic and antimony.

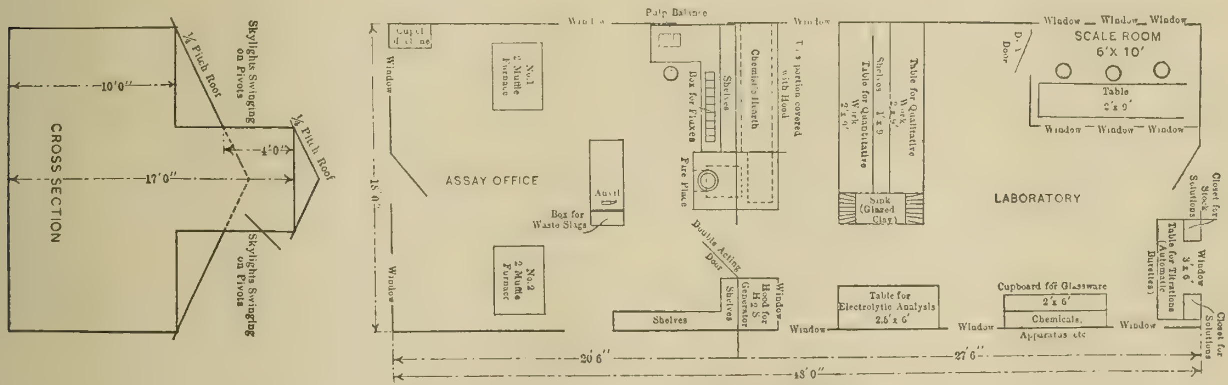


FIG. I.—GENERAL ARRANGEMENT OF LABORATORY AND ASSAY OFFICE FOR A SMELTING PLANT.

Every fourteen days, both flue-dust and speiss are tested for their copper and lead contents, and occasionally a complete analysis of these products is required. The lead bullion is analyzed for its content of iron, copper, arsenic, antimony and bismuth. In addition to the routine work mentioned above, an analysis of coal or coke for the percentage of ash, sulphur, volatile matter and fixed carbon, as well as a determination of its calorific value, is called for, and occasionally an analysis is made of refractory brick and of gas. The chemist, in order to do all this work, must devote his entire time for ten hours every day. The above-named conditions are not hypothetical, but have actually occurred in the author's own experience at a large lead smelter.

The chief requisites for a works laboratory are aboundant light and good ventilation, because, even with the best arrangement of hoods and flues, the air is liable to become contaminated by acid fumes.

The laboratory building is constructed of wood, and is ceiled inside with tongue and groove lumber, leaving a 4-inch air space between the outer and inner walls. The roof terminates in a large ventilator, having windows swinging on pivots over the laboratory portion, while over the assay office the ventilator has fixed shutters of 1 x 6 inch wooden slats, set at an angle of 45°, with 3-inch spaces between adjacent slats. A chemist's hearth having a hood supplies the heat necessary to boil solutions, as well as for other uses. The main working table is placed opposite the chemist's hearth, which arrangement allows the chemist to carry on his analytical work, and, at the same time, to control the heating of the samples in sight on the hearth. The table is equipped on both sides with drawers, closets and shelves; the side nearest the hearth is used for quantitative work only, while on the other is a complete set of the reagents needed for qualitative mineral analysis. A small kerosene stove and a copper kettle, placed on the highest shelf of the table, gives a constant supply of hot distilled water. On opposite sides of the bottom of the kettle are two small bibbs, to which are fastened small rubber tubes, having at each outer end a pointed glass tube, and sufficiently long to reach both ends of the table. A Mohr "pinch" cock regulates the flow of water to a nicety, and the 4-foot head gives sufficient pressure to wash thoroughly and quickly precipitates, filters, etc. This arrangement avoids the discomfort arising from continued inflation of the cheeks which the ordinary wash-bottle necessitates. A sink of glazed clay with drainboards is placed at the end of the table.

A table opposite the main working table is reserved for electrolytic analysis, and is equipped with an accumulator, which is used as the source of current. The current strength and potential respectively measured by an ammeter with a range of 5 amperes, and sub-divisions of 0.1 ampere, and a voltmeter with a range of 10 volts, and sub-divisions of 0.1 volt. The rheostat consists of a band of German silver placed in the circuit and having a sliding contact enables the operator to change the potential. As the electrolytic determinations are limited to copper mattes, and occasionally leaf copper for standardizing purposes, the amperage and voltage best suited to the chemical composition of the matte will be used, and when once ascertained, they will remain practically constant. Luckow's cylinder and spiral are used as electrodes.

A table in front of a window is used for titrations and is provided with burettes, which can be filled automatically. This special type of burette consists of a two-liter glass bottle with a pressure and a delivery tube passing through the cork, the delivery tube connecting direct to the upper end of burette. The presseure necessary to force the solution from the bottle to the burette is supplied by two rubber bulbs having air inlet valves so arranged that a continuous pressure may be produced. The inner (and larger) bulb is covered with a strong cord net in order to remove the danger of bursting from too great a pressure. On the table are two closets, having shelves large enough to hold a two-quart bottle, and serving the purpose of keeping the solutions in the dark; the closet doors bear the symbols of the respective stock solutions. In his laboratory the author has the following-named stock solutions, potassium cyanide, potassium dichromate, potassium permanganate, potassium ferro-cyanide, N/10 potassium hydroxide, N/10 sulphuric acid, copper sulphate, ammonium molybdate. An additional set of these solutions kept in reserve in the dark closet allows a convenient interval of time in which to prepare fresh solutions.

Fig. 2.

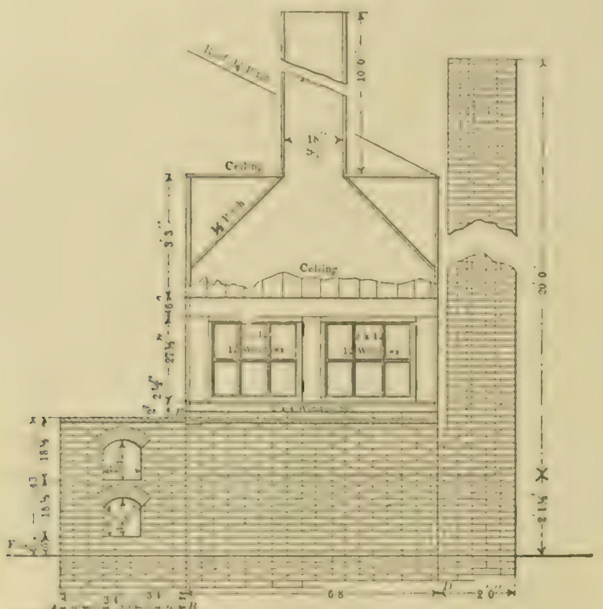


Fig. 3.

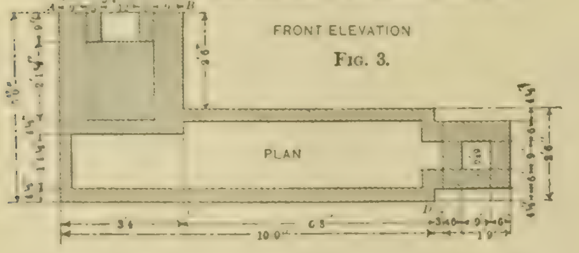
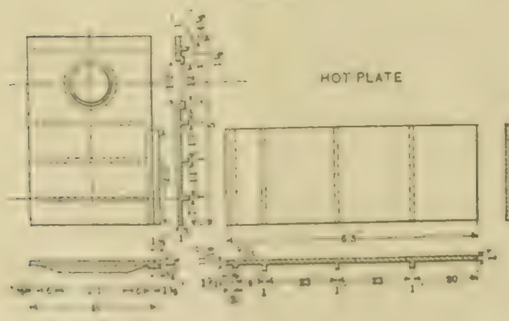


Fig. 4.



Figs. 2, 3, 4.—Details of the Chemist's Hearth.

A cupboard is provided in which to store a complete set of the chemicals needed in this line of work, as well as such apparatus and glassware as are only occasionally used.

The balance-room is tightly ceiled, special care having been taken to make it dust-proof. It has three windows on each long side, and a swinging sash-door entrance at the end, which guards against collisions between assayer and chemist when passing through the door-way in opposite directions.

The bench on which the balances rest is suported by wooden posts set in concrete, and extending through the floor to the ground with a 0.25-inch clearance. The scales set on plate glass, when once adjusted, remain level for a considerable time, any shrinkage of the table having no effect on the position of the plane of the plate glass.

The general arrangement of the laboratory and assay office is shown in Fig. 1.

The chemist's hearth, a description of which may possess special interest to chemists, and metallurgists at places where gas is not obtainable, is illustrated by Fig. 2. This hearth rests on the ground and the flue from it is filled with ashes and earth to within 18 inches of the hot-plate. In the front of the ash-pit, beneath the assay office floor, is a 12 x 12 inch opening, which is closed with a piece of sheet iron luted on with clay. The accumulated ashes in the ash-pit are removed through this opening, thus avoiding their removal through the assay office. The ash-pit door, shown in the drawing, is used solely to regulate the draft. Old rails, preferably, are used as grate bars.

The front elevation of the hearth is shown in Fig. 2, which includes also the elevation of the fireplace front, with the ash-pit door and the feed door. The lines *AB* and *CD*, explain the respective elevations. The walls of the hearth consist of one course of brick, excepting at the stack, which is of a course and a half, and the fire-box, which is of two courses. These walls support the hot-plate, having its upper surface 43 inches above the level of the floor. A detailed dimensioned drawing of the hot-plate is given in Fig. 4. The plate is cast in two places, having a lap so that a tight joint may be obtained, and at given intervals ribs are cast as a safeguard against warping. A circular hole, over which the still is placed, is left in the plate. A portion of the hot-plate 2.5 feet by 6 feet 8 inches, is covered with a hood, which rests on one layer of bricks, except at the hottest parts, where there are two layers in order to protect the wood. The back side of the hood does not rest on bricks, but is separated from the plate by a 2-inch air space extending the entire length of 6 feet 8 inches. Access to the hot-plate is obtained through two windows, each having twelve lights of glass and hinge on butts. The hood is tightly ceiled with tongue and groove lumber, and has an 18 x 18 inch wooden chimney, 10 feet high, to carry off the fumes. The temperature of the inside of the hood and hood chimney is sufficient to draw in fresh air constantly and thus improve the ventilation of the laboratory.

The great advantage of the hot-plate is in its gradual decrease in temperature towards the chimney. The heating of solutions is generally started at the coolest place, and gradually continued toward the hottest part. The heat is diffused over a large area, and is not concentrated at one small spot, as is the case with a Bunsen burner; and the boiling over of solutions is thus easily avoided at the expenditure of the least attention and care; this allows the chemist time in which to attend to other work. A small, uncovered portion, 2.5 feet by 3 feet 4 inches, is reserved for operations which are preferably conducted in the open air. The fire-place extends into the assay office, and is closed off from the laboratory by a ceiled partition, formed by the continuation of the back side of the hood.

A still, placed over the circular hole described above, provides the laboratory with about 14 gallons of distilled water daily. The still now used is called the Cuprigraph Sanitary Still, No. 11, and when once regulated requires very little additional attention. The hearth thus serves two purposes, one to provide the laboratory with distilled water, and the other to give the chemist, at the same time, a very efficient way of heating solutions. The distillation of the water utilizes much of the heat, yet the hearth is necessarily wasteful, the flue being too short for economizing fuel. The weekly fuel consumption, however, is only from one-half to three-quarters of a cord of wood, which, at $4.00 a cord, is equivalent to an operating expense of from 28 to 42 cents a day.

Fig. 5 shows the decrease in temperature with increasing distance from the fire-place. The temperatures were taken with a thermometer registering up to 250° C., and, while the readings are not exact, because the hearth was fired intermittently, they convey a fairly true idea of the temperature at different portions of the plate. Generally, the hot-plate is not used at temperatures as high as those recorded.

The laboratory is also provided with a hood, containing a hydrogen sulphide generator.

The assay office has two muffle furnaces, one kept as a reserve, or used when an accumulation of work has to be done. A general fluxing mixture is kept, which, with slight alterations, can be used to flux ordinary ores.

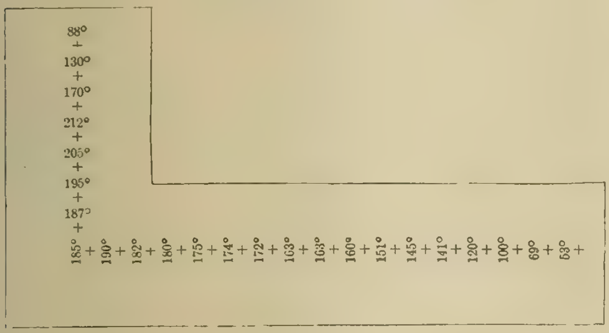


FIG. 5.—TEMPERATURES IN DEGREES C. AT DIFFERENT PARTS OF THE HOT PLATE.

All the samples are received from the sampling mill in a ground condition passed through a 100 or 125 mesh sieve. No grinding or sampling is done in the assay office. The cupels are made by a cupel machine.

The cost of the assay office and laboratory (not including chemical apparatus and supplies) was a little below $525, while the cost of the chemist's hearth was $142.

Metallurgical Calculations—I.

By J. W. Richards, Ph. D.

Professor of Metallurgy in Lehigh University.

Introduction.

The making of calculations respecting the quantitative working of any process, furnace or piece of apparatus used in metallurgical operations is of the greatest importance for estimating the real efficiency of the process, for determining avenues of waste and possible lines of improvement, and for obtaining the best possible comprehension of the real principles of operation involved.

The possibility of making such calculations respecting any process, furnace or apparatus depends on skill in collecting such necessary data as can be obtained by observation or measurement, the insight or intuition to see the further use which can be made of said data when once obtained, and, finally on the possession of a working knowledge of the fundamental chemical, physical and mechanical principles involved in the calculations. The highest desideratum, all in all, however, is a plain analytical, common sense mind, capable of clear, logical thinking. It is the writer's conviction that no study of details, or even observation of plants in actual operation, can supply the insight into metallurgical processes and principles, such as is gained by these calculations, in addition to the high grade of mental training involved.

Scope of the Treatise.

- Discussion of the chemical equation.
- Weights and volumes of gases.
- Correction of gas volumes for temperature and pressure.
- Combustion of commercial fuels.
- Heat of chemical combination; of combustion.
- Theoretical flame temperatures:
  - With pure oxygen.
  - With ordinary air.
  - With diluted air: Farley's system.
  - With hot air and cold gas
  - With hot air and hot gas.
  - Effect of excess air.
- Calculation of furnace efficiencies.
- Chimney draft.
- Transmission of heat through metals, brick, etc.,
- Water gas.
- Producer gas: Efficiency, effect of drying.
- Mixed gas:
  - Use of steam in producers.
  - Increased efficiency.
  - Maximum steam permissible.
- Regenerative gas furnaces:
  - Proportioning of gas and air regenerators.
  - Efficiency of generators.
  - Heat balance sheet.
  - Theoretical temperatures under different conditions.
- Gas engines:
  - Calculation of temperatures in cylinder.
  - Efficiency; balance sheet.
- Cupolas: Amount of blast required.
  - Efficiency of running.
- Blast furnaces:
  - Balance sheet of materials.
  - Calculation of blast received.
  - Efficiency of blowing engines.
  - Power and dimensions of blowing engines.
  - Carbon consumed at tuyeres.
  - Effect of atmospheric changes.
  - Effect of the moisture in the blast.
  - Calculation of the temperature.
  - Effect of hot-blast.
  - Heat balance sheet of the furnace.
  - Power producible from the waste gases.
- Hot-blast stoves: Theory of iron-pipe and fire-brick stoves.
  - Efficiency.
- Bessemer Converters:
  - Blast required and time of operation.
  - Balance sheet of materials.
  - Heating efficiency of various ingredients of bath.
  - Heat balance sheet.
  - Theoretical rise in temperature.
  - Conversion of copper matte.
- Open-hearth Furnaces:
  - Pig and ore process; calculation of charge.
  - Heat evolved or absorbed in bath reactions.
  - Efficiency of furnaces; of furnaces and producers.
  - Heat balance sheet.
- Electric furnaces:
  - Working temperatures.
  - Heat balance sheet.
  - Efficiency.
- Electrolytic furnaces:
  - Absorption of heat in chemical decompositions.
  - Equilibrium of temperature attained.
  - Ampere and energy efficiency.
- Electrolytic refining:
  - Calculation of plant and output.
  - Power requirements; temperature of baths.
  - Ampere and energy efficiency.
- Condensation of metallic vapors:
  - Principles involved.
  - Application to condensation of zinc and mercury.

[As this treatise on metallurgical calculations is not now written, but is being written as it appears in this journal, the above plan may be only followed in general, and additions or variations introduced as seems best, as the work progresses.

The above gives, however, some idea of the scope of the work and the field which will be covered. Any suggestions from readers, as to important fields for calculation not mentioned, will be gratefully received and included in the scope of the treatise, if practicable.]

The Chemical Equation.

The calculation of the quantitative side of many metallurgical processes depends upon the correct understanding of chemical equations. Every chemical equation is capable of giving three most important sets of data concerning the process which it represents; it shows the relative weights of the reacting substances, their relative volumes, when in the gaseous state, and the surplus or deficit of energy involved in the reaction, when the heats of formation of the substances concerned are known.

Atomic Weights.

These are the basis of all quantitative chemical calculations. For metallurgical purposes we may use them in round numbers as:

Hydrogen	H	1
Lithium	Li	7
Beryllium	Be	9
Boron	B	11
Carbon	C	12
Nitrogen	N	14
Oxygen	O	16
Fluorine	F	19
Sodium	Na	23
Magnesium	Mg	24
Aluminium	Al	27
Silicon	Si	28
Phosphorus	P	31
Sulphur	S	32
Chlorine	Cl	35.5
Potassium	K	39
Calcium	Ca	40
Titanium	Ti	48
Vanadium	V	51
Chromium	Cr	52
Manganese	Mn	55
Iron	Fe	56
Nickel	Ni	58.5
Cobalt	Co	59
Copper	Cu	63.6
Zinc	Zn	65
Arsenic	As	75
Selenium	Se	79
Bromine	Br	80
Strontium	Sr	87
Zirconium	Zr	90
Columbium	Cb	94
Molybdenum	Mo	96
Palladium	Pd	106
Silver	Ag	108
Tin	Sn	118
Antimony	Sb	120
Tellurium	Te	125.5
Iodine	I	127
Barium	Ba	137
Tantalum	Ta	183
Tungsten	W	184
Iridium	Ir	193
Platinum	Pt	195
Gold	Au	197
Mercury	Hg	200
Thallium	Tl	204
Lead	Pb	207
Bismuth	Bi	208
Thorium	Th	232
Uranium	U	238

Relative Weights.

Writing any chemical equation between elements or their compounds, the relative weights concerned in the reaction are obtained directly from using these atomic weights, which are, themselves, of course, only relative. *E. g.*, the slagging of iron in Bessemerizing copper matte:

$$2FeS + 3O^2 + 2SiO^2 = 2(FeO.SiO^2) + 2SO^2$$
$$176 + 96 + 120 = 328 + 128$$

These relative weights may be called kilograms or tons, pounds, ounces or grains; whatever units of weight we may be working in. In most metallurgical work we use kilograms or pounds as the convenient weight units.

Relative Volumes of Gases.

Where gases are involved, the relative number of molecules of the gaseous substance concerned in the reaction stands for the relative volume of that gas concerned in the reaction. It is usual and convenient to designate these relative volumes by Roman numerals, placed above the formulæ. The following are some simple examples:

Complete combustion of carbon:

$$C + \overset{\text{I}}{O^2} = \overset{\text{I}}{CO^2}$$

Incomplete combustion of carbon:

$$2C + \overset{\text{I}}{O^2} = \overset{\text{II}}{2CO}$$

Combustion of marsh gas:

$$\overset{\text{I}}{CH_4} + \overset{\text{II}}{2O_2} = \overset{\text{I}}{CO_2} + \overset{\text{II}}{2H_2O}$$

Production of water gas:

$$C + \overset{\text{I}}{H^2O} = \overset{\text{I}}{CO} + \overset{\text{I}}{H^2}$$

In each case above, the volume of a solid or liquid cannot be stated, but the relative volumes of all the gases taking part in a reaction are derived simply from the number of molecules of each gas concerned. These relative volumes may be called so many cubic meters or liters, or cubic feet, or whatever measure is wanted or being used. In most metallurgical calculations it is convenient to use cubic meters or cubic feet.

Exact Weights and Exact Volumes.

If we specify or fix the weights used, as, for instance, so many kilograms of each substance as the numbers representing the relative weights, then we can, by using one constant factor, convert all the relative volumes into the real or absolute volumes corresponding to the weights used. If, for instance, we take the equation of the production of water gas:

$$C + \overset{\text{I}}{H^2O} = \overset{\text{I}}{CO} + \overset{\text{I}}{H^2}$$
$$12 + 18 = 28 + 2$$

With the relative weights written beneath and the relative volumes above, then if we fix the weights as kilograms, the relative volumes can be converted into actual volumes in cubic meters by multiplying by 22.22. A cubic meter of hydrogen gas (under standard conditions) weighs 0.09 kilogram, and thence 2 kilograms will have a volume of $2 \div 0.09 = 22.22$ cubic meters. But the relative volumes show that the CO and H_2O gas are the same in volume as the hydrogen, and it, therefore, follows that each Roman I stands for 22.22 cubic meters of gas, if the weights underneath are called kilograms. The consideration of these relations is very advantageous, because, by means of this factor (22.22) we pass at once from the weight of a gas to its volume; each molecule or molecular weight of a gas, in kilograms (or, briefly, each kilogram-molecule), represents 22.22 cubic meters of that gas.

The conversion from weight to volume is quite as simple using English measures; and, by a strange coincidence, the same factor can be used as in the metric system. The coincidence alluded to is the fact (which the writer, as far as he can discover, was the first to notice) that there happens to be the same numerical relation between an ounce (av.) and a kilogram, as there is between a cubic foot and a cubic meter; in short, there are 35.26 ounces (av.) in a kilogram, and 35.31 cubic feet in a cubic meter. The difference is only one-seventh of one per cent, which can be ignored, and we can therefore say that if the relative weights in an equation are called ounces (av.), each molecule of gas in the equation represents 22.22 cubic feet.

Example.—The production of acetylene from calcium carbide:

$$CaC^2 + H^2O = CaO + \overset{1}{C^2H^2}$$

$$64 \quad + \quad 18 \quad = \quad 56 \quad + \quad 26$$

Interpreting by weights, and calling the relative weights ounces, we can call the 1 molecule of C^2H^2 gas 22.22 cubic feet, so that, theoretically, 64 ounces of pure carbide, acting on 18 ounces of water, produce 56 ounces of lime and 26 ounces of C^2H^2 gas, the volume of which is 22.22 cubic feet.

WEIGHTS AND VOLUMES OF GASES.

The weight of one cubic meter of dry air, under standard conditions (at 0° Centigrade and at a pressure of 760 millimeters of mercury), is 1.293 kilograms. The composition of air is:

	By Weight.	By Volume.
Oxygen	3	21
Nitrogen	10	80
or, in percentages,		
Oxygen	23.8	20.8
Nitrogen	76.2	79.2

While these may not represent the absolutely accurate average composition of dry air, yet the variations are such that the above simple ratios, 3 to 10 and 21 to 80, are close enough for all practical purposes in metallurgy.

The weight of one cubic foot of dry air is 1.293 ounces (av.).

The weight of one cubic meter of hydrogen gas, at standard conditions, is 0.09 kilogram (1 cubic foot, 0.09 ounces). The formula of hydrogen gas is H^2, is molecular weight 2; and since the densities of all gases are found experimentally to be proportional to their molecular weights, it follows that the density of any gas referred to hydrogen is expressed numerically by one-half its molecular weight. But, the weight of a cubic meter of gas is the weight of a cubic meter of hydrogen multiplied by the density of the gas referred to hydrogen; thus, is obtained the weight of a cubic meter of any gas whose formula is known. Examples follow:

Formula.	Molecular Weight.	Density Referred to Hydrogen.	Weight of 1 Cubic Meter.
Hydrogen ....H^2	2	1	0.09 kilos.
Water vapor ......H^2O	18	9	0.81 “
Nitrogen ....N^2	28	14	1.26 “
Oxygen ....O^2	32	16	1.44 “
Carbon monoxide...CO	28	14	1.26 “
Carbon dioxide ....CO^2	44	22	1.98 “
Marsh gas ....CH^4	16	8	0.72 “

Etc., etc.

In the case of water vapor, a particular explanation is necessary. It cannot exist under standard conditions, but condenses to liquid at 100° C., if under 760 millimeters pressure. It does exist at lower temperatures than 100°, but only under partial pressures of fractions of an atmosphere; thus, at a pressure of of 1-50 atmosphere (when it forms 1-50 of a mixture of gases) it can exist uncondensed at ordinary temperatures (15° C. or 60° F.). The above weight for a cubic meter of water vapor (0.81 kilos. per cubic meter at standard conditions) is, therefore, only a hypothetical value, but it is extremely useful, because it enables us to calculate, by the principles to be explained further on, the weight of a cubic meter of water vapor under any conditions of temperature and pressure at which it is possible for it to exist.

CORRECTIONS FOR TEMPERATURE.

The volumes of all permanent gases increase uniformly for uniform increase of temperature, so that, starting with a given volume at 0° C., it is found that their volume increases 1-273 for every degree Centigrade rise in temperature. Thus, at 273° C., the volume is just double the volume at 0°. Stating this fact in another way, we may say that the gas acts as if it would have no volume at — 273° C., and would increase uniformly in volume from this point up to all measurable temperatures, the increment being, for each degree, 1-273 of the volume which the gas has at 0° C. A still briefer statement is that the volume of a gas is proportional to its temperature above — 273° C., or to its absolute temperature—the latter being its temperature in C° + 273.

The converse of these principles is, that the density of a gas; that is, the weight of a unit volume, varies inversely as its absolute temperature.

In Fahrenheit degrees, we can say that a gas expands 1-490 (1-273 × 5-9) for every degree rise above 32° C.; or that the volume is proportional to the absolute temperatures, *i. e.*, to the F.° — 32 + 490 (= F.° + 458).

These principles are in constant use in metallurgical calculations. Thus, one kilogram of coal will need 8 cubic meters of air to burn it, at 0° and 760 millimeters pressure. What volume will that be at 30° C. and the same pressure? Since 30° C. is 30 × 273 = 303° absolute, the two temperatures will be 273 and 303, and the

$$\text{Volume at } 30^\circ \text{ C.} = \text{volume at } 0^\circ \text{ C.} \times \frac{303}{276}.$$

It is always to be recommended to make such calculations in the above form; that is, to first put down the known volume, and then to multiply it by a fraction, the numerator and denominator of which are the two absolute temperatures. A moment's reflection will show which way the fraction must be written; if the new volume must be greater than the old, the value of the fraction must be greater than unity, the higher temperature must be in the numerator; if the fraction were inverted, we know that the result would be less than the starting volume instead of greater, which would be wrong.

Taking an example in Fahrenheit degrees: What is the volume under standard conditions of 175 cubic feet of gas measured at 90° F. and standard pressure (29.93 inches of mercury)? Since 32° F. is 490° absolute and 90° F. is 548° absolute, and the new volume must be *less* than the starting volume, we have,

$$\text{Volume at } 32^\circ \text{ F.} = 175 \times \frac{490}{548} = 156.5 \text{ cubic feet.}$$

CORRECTIONS FOR PRESSURE.

The principle is that the volumes of a gas are inversely as the pressure upon it, so that doubling the pressure halves the volume, etc. Since the practical problems almost always present the pressures as two numbers, all that is necessary is to multiply the original volume by a fraction whose numerator and denominator are the two pressures concerned, and arranged with the numerator the larger or the smaller of the two numbers, according as to whether the final volume should be greater or less than the starting one. Putting the solution in this manner avoids the primary school method of making a proportion, which is so apt to be expressed upside down, and absolutely avoids error with the minimum exercise of brain power.

Examples: What is the volume of 100 cubic meters of any gas, if the pressure is changed to 700 millimeters?

$$\text{Answer: } 100 \times \frac{760}{700} = 108.6 \text{ cubic meters.}$$

What is the volume at standard pressure of 150 cubic feet of gas measured at 28.50 inches of mercury?

$$\text{Answer: } 150 \times \frac{28.50}{29.93} = 142.8 \text{ cubic feet.}$$

CORRECTIONS FOR TEMPERATURE AND PRESSURE.

These can be both allowed for, by simply correcting first for one, and then for the other. Actually, the simplest statement is to put down the original volume, then to multiply it by one fraction, which corrects for temperature, and again by another fraction correcting for pressure, thinking out carefully for each fraction the proper way of expressing it, *i. e.*, whether it should increase or decrease the volume.

Examples.

What does 100 cubic meters of air at standard conditions become at 50° C. and 780 millimeters pressure?

Solution: $100 \times \frac{50 + 273}{273} \times \frac{760}{780} = 115.3$ cubic meters.

What is the weight of one cubic meter of hydrogen at 1000° C. and 250 millimeters pressure, its weight at standard conditions being 0.09 kilograms?

Solution: $0.09 \times \frac{273}{1000 + 273} \times \frac{250}{760} = 0.00637$ kilograms.

What weight of oxygen is in 1500 cubic feet of dry air at 100° F. and at 28.50 inches of mercury? (Refer to weight of air at standard conditions, and percentage composition.)

Solution: $1.293 \times \frac{3}{13} \times \frac{490}{558} \times \frac{28.50}{29.93} \times 1500 = 374$ ounces.

What is the weight of 50 cubic meters of water vapor at a temperature of 30° C. and a pressure of 31.6 millimeters?

Solution: $0.81 \times \frac{273}{303} \times \frac{31.6}{760} \times 50 = 1.517$ kilograms.

or $50 \times \frac{273}{303} \times \frac{31.6}{760} \times 0.81 = 1.517$ kilograms.

The first expression calculates the weight of a cubic meter of water vapor at the assumed conditions, and multiplies by 50; the second calculates the hypothetical volume of the 50 cubic meters if reduced to standard conditions, and multiplies by the hypothetical weight of a cubic meter at those conditions.

Problems Illustrating Preceding Principles.

Problem 1.

A bituminous coal contains on analysis:

Carbon	73.60
Hydrogen	5.30
Nitrogen	1.70
Sulphur	0.75
Oxygen	10.00
Moisture	0.60
Ash	8.05
	100.00

It is powdered and blown into a cement kiln by a blast of air.

Required: 1. The volume of dry air, at 80° F. and 29 inches barometric pressure theoretically required for the perfect combustion of one pound of the coal.

2. The volume of the products of combustion, using no excess of air, at 550° F. and 29 inches barometer, and their percentage composition.

Solution: The reactions of the combustion are:

$$\underset{12}{C} + \underset{32}{O^2} = \underset{44}{CO^2}$$

$$\underset{4}{2H^2} + \underset{32}{O^2} = \underset{36}{2H^2O}$$

$$\underset{32}{S} + \underset{32}{O^2} = \underset{64}{SO^2}$$

Requirement (1):

The oxygen required for burning one pound of coal is:

Oxygen for carbon.... $= 0.7360 \times 32/12 = 1.963$ pounds.
Oxygen for hydrogen.. $= 0.0530 \times 32/4 = 0.424$ "
Oxygen for sulphur.... $= 0.0075 \times 32/32 = 0.0075$ "

Total required .... 2.3945 "
Oxygen in coal .... 0.1000 "

Oxygen to be supplied .... 2.2945 "
Nitrogen accompanying .... 7.6483 "

Air necessary .... 9.9428 "
= 159.08 ounces (av.).

Volume of air necessary (standard conditions)

$$= \frac{159.08}{1.293} = 123.03 \text{ cubic feet.}$$

Volume of air necessary at 80° F. and 29 inches barometer =

$$123.03 \times \frac{80 + 458}{490} \times \frac{29.93}{29} = 139.4 \text{ cubic feet.} \quad (1)$$

Requirement (2):

		Pounds.
The weight of CO^2 formed is....	$0.7360 + 1.963$	$= 2.729$
The weight of H^2O formed is....	$0.0530 + 0.424$	$= 0.477$
The weight of moisture is.......		0.060
The weight of SO^2 formed is....	$0.0075 + 0.0075$	$= 0.150$

The weight of nitrogen altogether is $7.6483 + 0.0170 = 7.6653$ pounds. Converting these weights into ounces, and dividing each by the weight of a cubic foot of each gas in ounces, we have the volume of these theoretical products at standard conditions:

Volume $CO^2 = 2.729 \times 16 \div 1.98 =$
$43.664 \div 1.98 = 22.05$ cubic feet.
Volume $H^2O = 0.537 \times 16 \div 0.81 =$
$8.592 \div 0.81 = 10.61$ " "
Volume $SO^2 = 0.150 \times 16 \div 2.88 =$
$2.400 \div 2.88 = 0.83$ " "
Volume $N^2 = 7.665 \times 16 \div 1.26 =$
$122.645 \div 1.26 = 97.34$ " "

Total volume at standard conditions.... = 130.83 " "

Volume at 550° F. and 29 inches barometer =

$$130.83 \times \frac{550 + 458}{490} \times \frac{29.93}{29} = 277.8 \text{ " "}$$

The percentage composition by volume follows from the above volume as:

CO^2	16.9 per cent.
H^2O	8.1 "
SO^2	0.6 "
N^2	74.4 "
	100.00

Problem 2.

Natural gas in the Pittsburg district contains:

Marsh gas	CH^4	60.70 per cent.
Hydrogen	H^2	29.03 "
Ethane	C^2H^6	7.92 "
Olefiant gas	C^2H^4	0.98 "
Oxygen	O^2	0.78 "
Carbonic oxide	CO	0.58 "

Required:

(1) The volume of air necessary to burn it.

(2) The volume of the products of combustion.

Reactions:

$$CH^4 + 2O^2 = CO^2 + 2H^2O$$
$$2H^2 + O^2 = 2H^2O$$
$$2C^2H^6 + 7O^2 = 4CO^2 + 6H^2O$$
$$C^2H^4 + 3O^2 = 2CO^2 + 2H^2O$$
$$2CO + O^2 = 2CO^2$$

Solution:

Oxygen required for CH^4.. $= 0.6070 \times 2 = 1.2140$ parts.
Oxygen required for H^2... $= 0.2903 \times \frac{1}{2} = 0.1451$ "
Oxygen required for C^2H^6.. $= 0.0792 \times 7/2 = 0.2772$ "
Oxygen required for C^2H^4.. $= 0.0098 \times 3 = 0.0294$ "
Oxygen required for CO... $= 0.0058 \times \frac{1}{2} = 0.0029$ "

1.6686 "
Deduct oxygen already present ....0.0078 "

Leaves oxygen to be supplied ....1.6608 "

Corresponding to air .... $\frac{1.6608}{0.208} = 7.985$ (1)

Volumes of products of combustion:

	CO^2	H^2O	N
From CH^4......	0.6070	1.2140	
From H^2		0.2903	
From C^2H^6	0.1584	0.2376	
From C^2H^4	0.0196	0.0196	
From CO	0.0058		
From air			6.3242
Total products ..	**0.7908**	**1.7615**	**6.3242** (2)

The above solution is entirely in relative volumes, which may be all considered cubic feet or cubic meters, and are true for equal conditions of temperature and pressure.

Problem 3.

A Bessemer converter contains 10 metric tons of pig iron of the following composition:

Carbon	3.00 per cent.
Manganese	0.50 "
Silicon	1.50 "
Iron	95.00 "

On being blown, one-third the carbon burns to CO^2, the rest to CO; 5 per cent of iron is oxidized, and no free oxygen escapes from the converter. Blast is assumed to be dry.

Requirements:

(1) What weight of oxygen is needed during the blow.

(2) How many cubic meters of air, at standard conditions, will be needed.

(3) What will be the average composition of the gases.

Requirements:

$$\begin{array}{rcccc} C & + & O^2 & = & CO^2 \\ 12 & & 32 & & 44 \\ 2C & + & O^2 & = & 2CO \\ 24 & & 32 & & 56 \\ 2Mn & + & O^2 & = & 2MnO \\ 110 & & 32 & & 142 \\ Si & + & O^2 & = & SiO^2 \\ 28 & & 32 & & 60 \\ 2Fe & + & O^2 & = & 2FeO \\ 112 & & 32 & & 144 \end{array}$$

Oxygen needed:

C to CO^2....100 kilos $\times$ 32/12 = 266.7 kilos.
C to CO.....200 " $\times$ 32/24 = 266.7 "
Mn to MnO. 50 " $\times$ 32/110 = 14.5 "
Si to SiO^2..150 " $\times$ 32/28 = 171.4 "
Fe to FeO..500 " $\times$ 32/112 = 285.6 "

Total .... 1004.9 " (1)
Nitrogen accompanying this .... = 3349.7 "

Air needed .... 4354.6 "

Volume of air ...... $= \frac{4354.6}{1.293} = 3367.8$ cubic meters. (2)

Volume of products of combustion:

$CO^2 = 100 + 266.7 = 366.7$ kilos $= \frac{366.7}{1.98} = 185.2$ cu. m.

CO $= 200 + 266.7 = 466.7$ kilos $= \frac{466.7}{1.26} = 370.4$ cu. m.

N $= \frac{3349.7}{1.26} = 2657.9$ "

Total volume .... 3213.5 "

Percentage composition by volume:

CO^2	**5.8** per cent.
CO	**11.5** "
N	**82.7** "

[The next instalment of the "Metallurgical Calculations" will deal with the thermochemistry of metallurgical processes.]

A System of Classification for an Electrochemical Bibliography.

By Adolph L. Voege.

If a scientific investigation is undertaken in a university to-day, the experimenter is advised to devote two-thirds of his time to the literature of his subject, much of which must be spent in locating that literature. The work is completed, published, and the experimenter later discovers an earlier article unknown to him which offers suggestions that would have advanced his work much beyond where he left it. A manufacturer sees a by-product wasted or making but small returns. He wishes to put it to better use, to know in what trades its elements or their compounds are employed, and to learn the processes of their manufacture; but no thorough method of doing this now exists. An inventor must know completely the work already done in his field or his thought and time are largely wasted and misdirected. He hazards that to-day. Who does not fail to appreciate all of the numerous applications of a given principle of operation? Who does not recognize the difficulty of tracing the complete published work of any man?

These are needs at present, but it is conceivable that they could be overcome by a living index, properly manipulated, of all of the literature of a subject.

To keep pace with any branch of science, no book system is applicable, even assuming it to be otherwise perfect; in a half-dozen years the searcher is confronted with as many volumes, their indices and references. A book is old and incomplete when put on the market. It is inexpansible-bound.

Every specialty is growing. It requires independent indexing, where the title of each of its articles stands by itself. This means, for sufficient elasticity, nothing less than a card index.

The two main demands of an index are arrangements by authors and subjects. The author index can be alphabetical; but its use presupposes familiarity with an author, and generally with his work, a knowledge that most investigators do not have, and which greatly limits its value. It is more often of importance to learn all that has been done in a certain line, of its mere existence, when the experimenter's name is not known, and is of little interest. This makes the great demand for the second classification, the subject index. A system is required for this where articles on common subject matter are not liable to be separated by the use of synonomous terms, the estranging tendencies of the daily press, or the language in which the articles are written, but where they will be grouped by inherent characteristics and principles, and a list of references be secured, complete at a single point.

This system for economical operation must be such that the manipulation of it by unskilled labor is feasible.

The first requirements are fulfilled by the relative subject system, in which the subjects are grouped and classed from consideration of related interests.

The requirements of speed and cheapness are fulfilled by subdividing the classes into approximately equal groups and assigning to each subject of each group a unique symbol. These symobls might be characters, letters or numbers. Characters in general lack the essential of simple consecutiveness. Letters fail, in groups, through slow appeal to the eye, through lack of rhythm (especially in consonant groupings) to the ear, and through the labor of tracing their correct consecutive order in their comparatively large body, the alphabet, to the memory. Arabic numbers overcome these objections, and applied as decimals, their base, 10, reduces consecutiveness to the most well-known form.

The relative subject decimal system, therefore, has been found superior to others, and it is proposed to employ it for the bibliography of electrochemistry, which is being undertaken. This system will be applied to the formation of a printed scheme wherein each class, branch and sub-branch of the science after much study, is carefully given a place among those most closely related to, or affected by it. A number is then assigned to each branch which identifies it always. This system possesses boundless expansibility, for we have but to add a decimal-place to any sub-group to obtain ten new subject locations under that sub-group. The first consideration in prefixing these numbers will be conformity with existing bibliographical work classification. Then, with the assistance of specialists, more detailed division will be secured. These theoretical groupings will be made to yield to positions of greatest usefulness through testing the provisional system with several thousand titles collected from varied old and recent sources. The final grouping arrived at will be that used for the bibliography. It will be arranged into a printed scheme and the numbers that have then been assigned must never be altered.

The system of Melvil Dewey, librarian of the New York State Library, as elaborated by the Institute de Bibliographie, of Brussels, and the Institute de Bibliographie, of Paris, is based on the above principles, and will form the foundation of the electrochemical classification. It has arranged human knowledge into nine great classes: Philosophy, Religion, Sociology, Philology, Science pure, Science applied, Art, Literature and History, utilizing a tenth great class for general topics applying to all of these, such as societies, schools, periodicals and so forth. To these ten classes one thousand main divisions have been given; those for pure science ranging through 500, and for applied science through 600. Further, pure science is grouped into mathematics, astronomy, physics, chemistry with mineralogy, geology, paleontology, biology, botany and zoology, with another division, reserved for topics general to all of pure science. Physics then extends from 530 through 539, having sections for mechanics, hydraulics, pneumatics, acoustics, optics, thermics, electricity, magnetism and the ever-present section for generalities. Electricity thus becomes 537. By adding one decimal figure to this ten new subdivisions are obtained. These are given to theories, statics, galvanic, atmospheric and dynamic electricities, to electrodynamics, applications, tables and problems. Galvanic electricity would be located then by 537·3, and may be sub-divided into propagation, calorific phenomena, electrolysis, electrolytic polarization, cells, various current actions with the other divisions still held vacant to provide for the growth of the science. These can be again sub-divided, and later these sub-divisions receive further expansion. The expansibility of the system is unlimited

Electrochemistry must enter the divisions of several groups of physics, 530, and of chemistry, 540; of applied science chiefly under electrical engineering, 621.3; chemical technology of chemicals, 661; and metallurgy, 669. Two criticisms of this separation will be made. One will be that electrochemistry should be, as a science, concise; a unit. The second, that many subjects have not been correctly located, will be heard from the specialist who tends to the belief that many subjects are of most special interest to his own branch. They may be answered by extending a borrowed simile: "A railroad must choose the most direct course to serve the most people, and its running schedule cannot be intelligently criticised by the patrons of any one of its sections. Whether 9:45 or 9:46 is the time of train departure for any station should be of small moment; that it carries one to his destination rapidly should suffice." The assignment of train departures is left to the despatcher's bureau, represented by the bibliographer, who knows the different positions of moving traffic and cross-roads, and seeing all sides of a relationship, prevents collisions.

To illustrate the arrangement of the headings of the final scheme, the following page from the provisional scheme for metallurgy, 669, is presented:

669·5	Zinc.
·51	Origin. Properties. General processes. Statistics.
·51	Minerals.
·521	Zinc sulphide. Blende.
·522	Zinc carbonate. Calamine, etc.
·523	Zinc silicate. Willemite, etc.
·524	Zinc oxide. Franklinite.
·525	Zinc aluminate.
·53	Extraction by dry methods.
·531	Silesian process. Calcination.
·532	Belgian process.
·533	English process. Distillation from crucibles.
·534	Combined processes.
·54	Extraction by electricity.
·541	Extraction by electric separation and distillation. Processes of Siemens and Halske, Cowles, etc.
·542	Extraction by electrolysis of the chloride. (fused). Process of Lorenz.
·543	Extraction with soluble anodes. Processes of Luckow, Hermann, Friedenshutte, etc.
·544	Extraction by electrolysis of sulphate solutions. Processes of Lestrange, Linderman, etc.
·545	Extraction by electrolysis of chloride solutions. Processes of Heinzerling, Hopfner, Forster, etc.
·546	Extraction by electrolysis of ammoniacal or alkaline solutions. Processes of Burghardt, Mond, Diffenbach, etc.
·55	Working of zinc.
56	Refining of zinc.

Treating the index in the spirit of the satisfied commutor suggested above, an attempt will be made to show, not how it does things, but rather what it will do. It has developed into three distinct parts; a printed alphabetical list of subjects and synonyms, with the assigned decimal of each affixed; a printed tabulated scheme with the subjects relatively grouped by their prefixed decimals, and a drawer or more of arranged cards of printed references. These are the tools.

It is desired to find what has been done, and where, and by whom, on damage to street mains by stray electric currents. Two methods of obtaining the answer offer. Conduit electrolysis can be looked up in the alphabetical index and its number found to be 621.315.25; then the cards of this number can be sought in the tray aided by guide-cards; and following those of 621.315 and preceding those of 621.316, grouped together, all of the cards of 621.315.25 are found. On each is a printed reference to one article that has appeared on our subject, consisting of the decimal of the subject in one corner, the author's name in the opposite one, and below, the year of publication of the article, with its title, and its English translation, the periodical, volume and page-limits, mention of illustrations, etc., together with a short explanatory note and a group of decimals, serving in concise manner as a digest of its contents. Had the searcher looked up "corrosion" or "electrolysis" or other leading word in the alphabetical index, under all of these would have been found a sub-head referring conclusively to the subject and possessing the same number as "conduit electrolysis" already determined. In the second method, the related subject scheme would have been used. Under the great head 621.3 "industrial electricity" would have been found a group, 621.31 "production and transformation of electric energy," with a sub-group 621.315 "conductors," of which one division is 621.315.2 "underground conductors," and here the subject would have been located under the sub-division 621.315.25, "electrolysis due to underground conductors."

This illustration is an electrochemical subject, yet it is isolated among the problems of linesman's work, because it means more to the linesman than to the electrochemist. It has been assigned the number of greatest utility.

However, there is a conviction of great flexibility which makes the adaptability of the system to special needs limitless. It is that of expressing relationships. As an instance of its use, an anatomist has interest in X-rays, still all X-ray work cannot be relegated to anatomy; however, a card can be numbered 611 :537.531 where the numbers for anatomy and X-rays are united by the colon, the symbol for relationship. These cards, being of a general nature, would appear in the card-drawer after those of 611, and before those of 611.1. By reversing the position of the numbers with respect to the colon, we form the card for the X-ray group, 537.531 :611, meaning X-rays applied in anatomy.

Again, were the problem to find what work had been done in preparing "membranes appropriate for osmotic pressure determinations," "membranes" or "diaphragms" or "dialyzing apparatus" will be found in the alphabetical index numbered or having a branch numbered 532.720.71. Turning to the card-drawers, the guide-cards, taller than the reference cards, lead the searcher by the numbers printed on them, in a few seconds to the cards of this number with their references to what has been done on the subject. Had the scheme tables been used, under 530 "physics" and its group 532 "liquids" would have been found a division, 532.7 "osmose," with a sub-division, 532.72 "osmotic pressure": and under the general head of this sub-division, 532.720 would be found 532.720.7 "apparatus" with a section 532.720.71 assigned to "diaphragms and membranes."

As a final illustration of the application of the system, it is desired to determine "the electrolytic sodium manufacturers of England" (so far as their plants have received mention in the literature). The numbers for "electrolytic sodium" and for "England" are determined. Sought through the scheme, under 660, "chemical technology" is a main group, 661 "manufacture of chemicals" with a great sub-group, 661.3 "manufacture of alkali" having a division, 661.32 "manufacture of sodium salts" with a sub-division, 661.323 "sodium," one branch of which is 661.323.2, "preparation of sodium by electrolysis." Following up the geographical portion of the scheme, distinguished by the parenthesis ( ), we find (4) "Europe," (42) "England." Turning to the card-drawer containing the cards for 661.323.2, electrolytic manufacture of sodium, following the relationship, colon, cards for this number are the geographical, or parenthesis, cards, and under these 661,323.2 (42) are those on "English plants."

By the use of type of different heights and heaviness of face, for the several decimal groups, the formidable appearance of such numbers is lessened. The last illustration would appear thus: 661.323.2 (42).

As illustrative of the electrochemical literature easily overlooked, among the one thousand electrochemical patents of Switzerland, important ones have been found in such unsuggestive sections as the classes devoted to varnish manufacture, liquor and beer ageing, canning industries, and glass smelting. In over 50 per cent of the classes electrochemical patents exist.

The application of this system is to be undertaken as a branch of the Concilium Bibliographicum of Zurich, Switzerland. A paper explaining the scope of the work this institute hopes to accomplish, and the spirit in which it is undertaken and the general plan to be followed by the enterprise, was presented at the April meeting of the American Electrochemical Society, and published nearly in full on page 313 of the August issue, 1904, of this journal.

Notes on Electrochemistry and Electrometallurgy in Great Britain.

(From Our Special Correspondent.)

STANDARDIZATION IN RELATION TO ELECTROCHEMISTRY.

At first sight there does not seem to be much room for standardization. Electric furnaces are not only numerous, but the application of a single maker's type of furnace frequently requires modification for various products, and one can only assume that none of the sub-committees of the British Engineering Standards Committee will give much attention to the subject. An interim report, however, of the sub-committee on generators, motors and transformers, issued last summer, is not without interest. Unfortunately, a great deal of electrochemical plant does not readily lend itself to standardization. The makers of small dynamos for low voltages and heavy currents are not numerous, and while each of such firms has doubtless its own standard sizes, these undoubtedly differ between firm and firm. When, however, we consider dynamos of higher voltages, the value of the work of this sub-committee to this specific industry becomes at once apparent. Those who propose to run several electrolyzing tanks in series across terminals connected to low-pressure distributing mains, should bear in mind that the standard low pressures recommended are 110, 220, 440 and 500 volts.

In one manner, an indirect one, it is true, the recommendations as to motors should certainly be borne in mind by the manufacturers of special low-voltage generators. It may be suggested that their particular shop standards of design should be synchronized with the speeds laid down for direct and alternating-current motors. It is only in very extensive works that the putting down of a special prime mover, steam or electric, will really pay. The average isolated small plant which one sees, in works where electrochemical work is carried on with its gas engines, countershafting and belting, is an anachronism in the days of cheap electric power distribution in nearly every municipality in Great Britain. The electric motor directly coupled to the shaft of the special generator is surely preferable to antiquated devices for wasting power in the form of friction. It seems highly probable that the interest and depreciation on the motor would be met by the saving of transmission losses. Electric current at 2d per unit is undoubtedly superior to gas at 2s 6d per 1000 cubic feet, and the items of wear and tear, lubrication and cooling water for the gas engine would disappear from the works accounts.

The recommendations of this sub-committee, which are too long for quotation, may be obtained by American firms who hope to supply such apparatus to Great Britain by application to the Engineering Standards Committee at 28 Victoria Street, London, W. C. Another feature of the report in question lies in the striking differentiation made between generators and motors which work under a fluctuating load, and those which work under a steady load. An electrochemist has no use for an armature which becomes unduly heated when working at its rated load. The subcommittee is investigating the matter by tests at the National Physical Laboratory at Teddington, and at the works of certain makers, with a view to formulating a temperature limit.

Faraday Society Transactions.

The Faraday Society has just issued Vol. I, part I, of the Transactions, which, from now on, will be published quarterly. The first number contains papers by H. J. S. Sand, B. Pool, H. D. Law and F. M. Perkin, L. Kahlenberg, F. J. Brislee, A. Minet, F. M. Perkin and W. C. Prebble, and F. Gelstharp, all of which were abstracted in Electrochemical and Metallurgical Industry. There is also a note by L. Gaster on the use and treatment of siloxicon.

Metallurgical Papers of the Month.

The Institute of Mining and Metallurgy had at their meeting on January 19, four papers down for reading. One, by Mr. H. E. West, dealing with "Early Dry-Crushing Plants in Western Australia and the Introduction of the Filter Press," was historical, if that adjective can be applied to events happening within the last decade. Among the reasons that led to the adoption of the dry-crushing process in Kalgoorlie, or at least on the Hannan's Brownhill mine, was probably first and foremost the scarcity of water, which cost 10s. per 1000 gallons at the government wells in 1896, for salt water. Fresh water in the previous year had fallen from 6d. to 2d. per gallon at Coolgardie, where government condensers were installed; but the conditions at Kalgoorlie were not so accommodating, and the first mill was erected long before any water was met with in the Brownhill Extended. A dry-crushing process, therefore, on the face of it, was presumed to require less water than wet-crushing, and this premise, whether correct or otherwise, was decidedly a factor of selection. The main advantages of this principle of reduction proved to be, however, in actual practice, not so much conservation of water as a more efficient state of the crushed pulp for gold extraction, in a condition amenable to direct cyaniding after the elimination of a certain percentage of fines; also, an efficient air separation of the fines in a dry state, for independent treatment; which ultimately resolved itself into filter pressing, a process also adopted in wet crushing and thus eventually becoming common to both systems. Filter pressing, moreover, gave a considerably higher extraction than percolation, it being safe to assume 90 per cent to 92 per cent of the gold contents, and of silver there was nothing to speak of.

The second paper, by Mr. Sydney Fawns, consisted of some "Notes on the Mount Bisthoff Tin Mine, Tasmania." According to the author cheapness and efficiency of operation have not yet been fully attained, for the cost of firewood is very high, and as all the boilers are old, and more or less out of repair, the horse-power must cost anything between 4d. and 6d. per unit; this can be greatly reduced by the use of electric power. There is a splendid supply of water-power below the dressing sheds near the mine running to waste, and, with a proper installation, the power would not cost more than a fraction of 1d. per unit. Hauling by electricity, instead of by the locomotive at present in use, would have the additional advantage that the motor could be worked in the tunnel underneath the present workings. The cost of mining crushing and dressing a ton of stanniferous material containing 1.322 per cent of tin amounts to 6s.7.933d. The company owns its own smelting works, in which the furnaces are of the reverberatory type. The charge for each furnace is 50 cwt. of ore and 10 cwt. of small coal. Eight hours are allowed for complete reduction. The metal is tapped into a float or brick-lined vessel, the slag on partial cooling being skimmed off and the metal refined in a large kettle by sinking billets of green wood under the surface, the steam arising from the wood releasing the dross, which is then skimmed off the surface. The metal, when refined, assays 99.80 per cent, and is shipped to England in the form of ingots weighing 75 pounds each. The slags vary in richness according to the ore treated and the working of the furnace, the average assay value of the slag being 5.3 per cent. The slags are broken up and mixed with small coal and lime and resmelted; the resulting metal being impure from the presence of a large quantity of iron, this metal is again resmelted with the next charge of ore.

Mr. G. A. Stonier, formerly chief inspector of mines in India, read the third paper, which dealt with "Mica Mining in Nellore (Southern India)." The method of preparation for market is crude enough in all conscience. The mica is carried from the mines to the dressing sheds in saucer-shaped baskets 1 foot 6 inches in diameter and 6 inches deep, made from the stalks of the date palm leaf or from jungle creepers at a cost of 1d. each. The headman selects and personally splits up the good plates with hoop iron or a cheap knife with 3-inch blade and 4-inch handle (purchased in the local bazaar for 2d. each) to obtain flat pieces free from flaws and cracks, not more than ⅛-inch in thickness and without cross-graining. They are passed on to women and children, who, with a knife and sheet tin or zinc templets, mark on them as large a rectangle as possible. The cutting along these ruled lines is done by men with ordinary European garden-shears, one end being firmly embedded in a block of wood buried in the ground, so that the cutting edge lies in a vertical plain. The poorer qualities of mica are treated in the same way, and the rectangles are tied into bundles according to the different qualities (1) clean; (2) stained; (3) spotted, and where possible according to sizes which do not differ by more than an inch in length. Compared with the Koderma mines in Bengal, which, in 1902, yielded 923 tons of mica, employing 7363 persons, the Nellore mines are small, their output in 1902 being only 143 tons, in the mining of which 2097 persons were employed.

The last paper, by Mr. E. H. Garthwaite, merely described a shaft-signalling device of a mechanical nature.

The Faraday Meeting.

For the meeting on January 30th three papers were announced, but only two were circulated to the members. Mr. John G. A. Rhodin's paper on "Mass Analyses of Muntz's Metal by Electrolysis" was first read. It was a pity that Mr. Rhodin was manifestly out of health, and no one will regret more keenly than the able author that he was not able to do full justice to his interesting paper. To begin with, the paper was read practically in its entirety, so that the discussion was not started until very late in the evening. The discussion was somewhat disconnected, and degenerated into a species of cross-examination, the editing of which will not be an easy task for Mr. Spiers.

Dr. Beckett Denison's paper on "The Equilibrium between Sodium and Magnezium Sulphates" was taken as read, and members requested to send their communications thereon to the *Proceedings*. Lastly, Mr. E. Kilburn Scott, in place of reading the paper, announced, said a few words on furnace linings, and the meeting adjourned.

The Duration of British Coal Supplies.

Twice within the memory of some of those who are not yet fifty years of age, the British public has suffered from scares respecting an immediate exhaustion of coal supplies. Two Royal Commissions have sat thereon, the second being mainly necessitated by the fact that the predictions of the first commission as to the future annual output were speedily falsified. A large electro-chemical industry can never flourish in England, because of our deficiencies of water-power. Probably there are no water-falls in the United Kingdom of 100 hp. that have not already been annexed by one industry or another. On that account a supply of cheap fuel is of even greater importance to those interested in British electrochemical or electrometallurgical processes than it is, say, to the Bradford woolen industry. The report of the last commission is of a generally reassuring nature, the commissioners estimating the coal available within 4000 feet of the surface as amounting to about 101,000,000,000 tons. Of substitutes for coal, nature, which has so plentifully endowed us with steam coal, has not provided us with petroleum or natural gas. On that account considerable space is given in the report to possible economies in the getting and consuming of coal. About 25 per cent to 30 per cent of the present annual internal consumption of 168,000,000 tons should be saved by judicious economies. In many cases the internal combustion engine is recommended for use. Also, it is recognized, that many seams which cannot now be worked at a profit, will in the future be rendered profitable by washing, sorting, coking, briquetting the coal, or converting it into gas. This should all tend to limit the cost of fuel for the next generation or so.

Expiring Patents.

The following British electrochemical patents of 1897 expire in 1905:

4,757	T. L. Wilson, Electrical Reduction Aluminium.
4,860	H. Howard, Electrical Heating and Welding.
13,124	H. Howard, Electrical Heating and Welding.
13,125	H. Howard, Carbon Holders and Screens for Electroheating and weldings.
19,775	A. Breuer, Diaphragms for Electrolytic Decomposing Apparatus.

So far as the writer is aware their expiration is not likely to greatly influence electrochemical development over here.

Market Quotations.

Chemicals have again been steady, no important changes have taken place, except in regard to shellac, which has now fallen to 156s. per cwt., a fall during the months of 44s.

Fine Para rubber is still at 5s. 3d. per pound, and guttapercha unchanged at 8s. od. per pound. Ceylon rubber has not been so prominent during the month. Some Ceylon planters of my acquaintance expect the price to recede to about 4s. od. per pound in the course of eighteen months or so.

Platinum is unchanged at 81s. 6d. per ounce. The only hope for a fall in the price lies in the ending of the Russo-Japanese war, which will permit of the development of other platinum mines. Copper has remained fairly steady, weakening slightly towards the end of the month. Cleveland pig iron has fallen to 48s. od. Block tin is still falling, being now quoted at £131 to £132 per ton, as against £134 to £135 at the beginning of the month. Ingot and sheet lead have each fallen 5s. od. per ton, being now quoted at £12.17.6 and £14.2.6, respectively.

London, February 4.

ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS.

By George P. Scholl, Ph. D.

Electric Furnaces and Furnace Products.

Process of Separating Metals from their Ores.—J. M. A Gerard, Paris, France. Patent 780,651, Jan. 24, 1905. Application filed Jan. 27, 1902.

The process relates especially to the production of steel directly from the ore, and is carried out in the furnace Fig. 1.

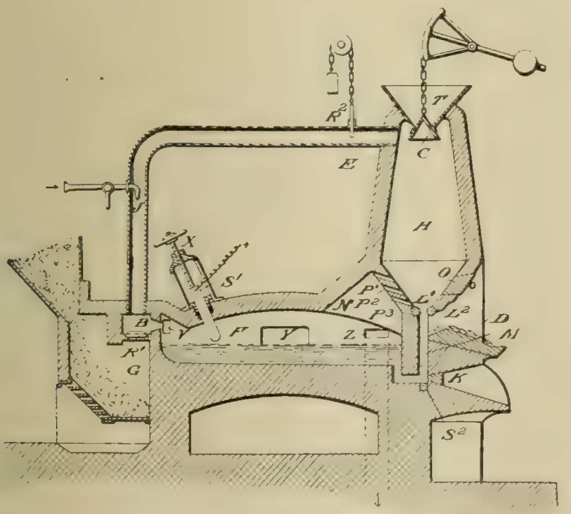


FIG. 1.—ELECTRIC STEEL FURNACE.

The structure comprises a shaft furnace as well as a reverberatory, and in the former the ores are reduced to iron sponge, while in the latter the sponge, which has been melted in the lower part of the reduction furnace, is refined. A gas producer is also a part of the apparatus. The iron ore, mixed with a suitable quantity of reducing agent and flux, if the latter is required, is charged into the hopper *T*, from which it passes into the furnace *H* by means of a bell *C*. At the place where the furnace bosh joins the hearth, a pair of electrodes L' L^2 are located, which serve the purpose of melting the reduced material and the gangue. A discharge channel *D*, located at the top of the conduit *M* serves for drawing off the slag. The molten metal falls into this conduit *M*, which connects the reducing part of the apparatus with the reverberatory part. An opening *O* in the bosh of the apparatus serves for the discharge of the furnace at the end of the operation. The reverberatory part *F* is provided with an opening *V* for the admission of air for burning the producer gas, and an opening *Z* which leads to the regenerative chambers, not shown. In addition the part *F* is provided with the positive electrode S' of an electric circuit, which can be regulated by the handwheel *X*. The negative electrode of that circuit is located on the bottom of the conduit *M*. The producer *G* communicates by means of a register R', which can be closed if the communication is to be shut off, and a passage *B* with the reverberatory furnace. A chamber *N* in the top of the furnace communicates with the reducing furnace *H* through openings P' P^2 P^3. A draw-off pipe *E* for the gases, which can also be closed if desired by means of the register R^2, leads from the upper part of the furnace to the passage *B*. A steam injector *J* permits regulation of the gas current. The process to be carried out in the furnace, as described, for the production of steel is stated to take place in the following manner: The ore mixed with carbonaceous and fluxing materials is initially heated in the reducing furnace by the hot gases from the producer. When the proper temperature for reaching is reached, these gases are shut off and a circulation of gases from the reduction furnace begins. The gases are drawn off through pipe *E*, aided in their circulation by the injector *J*, and enter the refining furnace *F*, where they heat and keep molten the material which has accumulated in the furnace. They then pass into the reducing furnace to heat the charge. This cycle of operations is stated to continue until the ore is reduced to sponge, which, as the ore is reduced, passes between the electrodes L' L^2 and is there melted by an electric current which should be of low voltage and high amperage. The slag passes off by the passage *D*, while the fluid metal sinks down in the conduit *M* and enters the reverberatory furnace *F*. When a suitable quantity of material has accumulated in the latter, the operation in the reducing furnace is stopped and the gases from the top as well as the steam injector are cut off. The refining operations now begin under the action of the electric current and previously heated air which is introduced through passage *V*. The positive electrode *S'* is brought into contact with the molten metal by regulating the extent to which it is introduced into the mass. A current of high voltage and low amperage is passed into the bath, which current is stated to cause an ebullition and projection of globules of metal above the surface of the bath, the effect of which is said to increase the surface presented to the oxidizing gases in the furnace and to stir the metal. The current necessary for this purpose is stated to be more or less variable, depending upon the mass to be treated and the number of electrodes employed. From 75 volts to at least double that amount are stated to give satisfactory results, while a current density of about ten amperes per sq. cm. cross-section of electrode is generally sufficient. The gaseous products during the period of refining are taken off through opening *Z*.

Electric Furnace.—D. B. S. Galbraith, Aukland, New Zealand. Patent 779,844, Jan. 10, 1905. Application filed Oct. 27, 1903.

The furnace is mainly intended for the treatment of iron sands, presumably such as occur in New Zealand. The furnace consists essentially of a rectangular structure, with a rather deep rectangular hearth for the reception of the liquid metal and the slag. It is, however, intended to break up the material of the charge, while it passes through the middle or fusing zone of the furnace, into a series of thin streams. For this purpose V-shaped troughs with perforated bottoms are arranged across the furnace, in a direction parallel to its longer axis. Below them is placed a row of non-conducting bars of crescent shape, upon which the material falls in its descent. Carbon conductors rest upon the ends and are partly embedded in these non-conducting bars. The material resting upon the latter thus completes the circuit between the electrodes, and is, in consequence, heated to redness. In this manner the particles of the charge were subjected to repeated fusion, inasmuch as several rows of these non-conductors or intercepters are placed in staggered fashion below each other, on which the material treated in the uppermost row falls in its descent. During this time, a current of reducing gas from an independent source acts on the material. The intercepters are made of beauxite or any other suitable refractory material and may be made to rotate.

Electrolytic Production of Metals and Compounds.

Electrode.—E. F. Price, Niagara Falls, assignor to Union Carbide Co. Patent 779,733, Jan. 10, 1905. Application filed June 28, 1904.

The patent relates to electrodes composed mainly of calcium carbide, such as are used in J. J. Griffin's process of producing chemical compounds by electrolyzing solutions with a carbide electrode. This process was described in ELECTROCHEMICAL INDUSTRY, Vol. II, page 206. As it is difficult, however, to obtain pieces of calcium carbide of sufficient size, regular shape and uniform composition, to serve as practical electrodes, and moreover the reaction between the carbide and the [illegible] is apt to disintegrate and destroy the electrodes, the present invention provides for manufacturing a composite electrode made of calcium carbide and a binder. The preferred binding agent is a coked hydrocarbon or a carbonaceous material, such as bituminous coal. In order to produce the electrodes, calcium carbide is crushed and mixed with the binder, *f. i.* 5 to 25 per cent by weight, of pulverized bituminous coal and a small amount of water free tar or pitch. The mixture is then molded into the desired form by means of a press or by forcing it through a die and cutting the resulting mass into the desired length. The pieces thus produced are baked at a temperature sufficiently high to decompose the hydrocarbons and to drive off the volatile matter, thus converting the bituminous material into coke. The coke not only binds the mass well together, but it also increases the conductivity, and is stated to be especially useful when the electrodes are to be used in aqueous solutions, as it retards the action of the water on the carbide, although its porosity permits the water to finally react on all the carbide. The porosity of the electrode may still further be increased, by saturating it with a liquid hydrocarbon or carbohydrate and again baking it and thereby depositing more carbon in the pores.

Electrolysis.—G. Rambaldini, Miniera di Bocchegiano, Italy. Patent 779,735, Jan. 10, 1905. Application filed Jan. 28, 1902.

The inventor proposes to conduct electrolysis in a vessel divided into two compartments by a non-porous, non-conducting partition, each compartment being filled with a different liquid. A horizontally arranged flat electrode is located in each compartment close to the upper edge of the liquid. The partition, however, does not reach to the upper edge of the containing vessel, and space is thus provided for a third solution of lesser specific gravity, which is arranged in a layer above two lower solutions. It is stated to be generally advantageous to employ as third liquid a dilute solution of the acid corresponding to the salt which is to be electrolyzed. Thus for the Siemens process for the electrolytic precipitation of copper from solutions containing sulphate of copper, the third liquid may be constituted by a dilute solution of sulphuric acid, while for the Hoepfner process, where the copper is present as cuprous chloride, the third liquid may be a dilute solution of hydrochloric acid. An apparatus for the electrolysis of waters of cementation is described as an example of the practical application of the method. It is a long vessel, separated by a number of partitions, as described above, into a series of compartments. Those of uneven number are filled with the waters of cementation, which are a mixture of sulphate of copper with other metallic sulphates, among which the sulphates of iron generally greatly predominate. The ferric sulphate has first been converted into ferrous sulphate and the waters have been concentrated as much as is possible with regard to the normal temperature for their treatment. The cells of even number are filled with a solution of sulphate of copper of 20 to 25° Bé. Above these solutions, water acidulated with sulphuric acid to a density of 10 to 15° Bé is carefully poured almost up to the edge of the containing vessel, in such a manner as to establish a sharp dividing line between the two layers. Means are provided for circulating the liquids in the compartments, while maintaining the upper liquid at rest. Some centimeters below the upper surface of the liquids in the compartments horizontally arranged sheets of copper or copper-covered materials are located, which serve as electrodes, suspended from above by electrical conductors, which should be insulated from the liquid. The electrode dropping into the water of cementation is connected to the positive pole and that dipping into the sulphate of copper with the negative pole.

Method of Treating Alkaline Solutions of Chromate of Soda.—W. T. Gibbs, Buckingham, Canada. Patent 779,705, Jan. 10, 1905. Application filed Feb. 10, 1904.

The process is intended especially for the production of bichromate of soda from the leach liquor, which is produced in the present method of manufacturing bichromate of soda. The commercial method of producing the leach liquor, and which is preferably used in this connection, consists in heating a mixture of chrome ore, lime and carbonate of soda until practically all the oxide of chromium is converted into chromic acid, which combines with the soda, forming sodium chromate. The roasted mass is then removed from the furnace, cooled and leached with water, when the chromate of soda and the excess of carbonate of soda present go into solution, while the lime and the gangue are left as an insoluble residue. The fresh liquor will contain caustic soda, some of which is, however, gradually converted into carbonate by exposure to the atmosphere. This alkaline solution of chromate of soda is, according to the proposed process, first subjected to a treatment with carbonic acid gas, preferably by pumping the gas through a column of the liquid as in the manufacture of sodium bicarbonate. The alkali present is thus converted into sodium bicarbonate and precipitated. The solution is then introduced into an apparatus divided into two compartments by a porous diaphragm. The anode compartment contains an anode of platinum or of another suitable material, while the cathode is formed by metallic iron or any other material not acted upon by alkali. During electrolysis sodium bicarbonate is formed in the anode compartment, while caustic alkali is set free on the cathode side. The cathode liquor from the cell is similar to the original leach liquor, but contains more free alkali and less sodium chromate. This liquor is concentrated by evaporation to bring up the chromate in solution to such strength that bicarbonate of soda is insoluble in the liquor, and is then treated with carbonic acid gas in the same manner as above so as to precipitate the alkali as sodium bicarbonate. The neutral solution of sodium chromate thus obtained returns to the process, preferably mixed with leach liquor prepared as above. The cathode liquors thus successively formed are successively treated with carbonic acid gas and returned to the cell with successive portions of the leach liquor. The anode liquors from the cell contain nothing but bichromate of soda, and are removed when they are concentrated enough and merely evaporated to the point at which they solidify on cooling, thus giving a cake of solid bichromate of soda.

Electrochemical Separation of Metals.—W. Mc. A. Johnson, Hartford, Conn. Patent 780,191, Jan. 17, 1905. Application filed June 23, 1902.

The process relates principally to the electrolytic refining of iron, which is carried out in a rectangular tank provided with a series of anodes and cathodes arranged in parallel. The electrodes do not reach entirely to the bottom of the apparatus, a considerable space being provided underneath them for the reception of the sludge formed during electrolysis. Slabs of slate are interposed between the lower ends of the electrodes, so as to prevent the formation of trees at those parts. The anodes consist of the iron which is to be refined, while the cathodes are thin sheets of iron or steel formed by rolling or by previous electrodeposition. The electrolyte may consist of a solution of ferrous ammonium chloride, slightly acidified if desired with acetic or some other weak organic acid, while a solution of ferrous ammonium sulphate may be used when the presence of a small percentage of sulphur in the deposit is permissible. The process of deposition is stated to be economically conducted at a temperature of 60° C, which accelerates the action. A current density of 10 to 20 amperes per square foot is desirable, while the voltage between the electrodes may be about 1½ volts. During the process of deposition, when the electrolyte is found to contain too large a percentage of copper, etc., either derived from the anodes or from an impure electrolyte, it is run into a cementation tank containing scrap-iron for the purpose of removing the copper. The electrolyte is thus kept in a condition which will assume the deposition of metal of the desired degree of purity, and to insure the deposition taking place in a uniform manner it is desirable to run the electrolyte continuously through the cementation tanks. It is stated that in this way the phosphorus, arsenic, antimony and carbon combined in the iron being refined will fall down as sludge, mostly as phosphides, arsenides, etc., of iron. Any precious metals will also be found in the sludge, while the manganese, nickel or cobalt present in the iron will be deposited with the iron as an alloy upon the cathode.

Method of Coating Metals.—Hugh Rodman, Cleveland. Patent 781,230, Jan. 31. 1905. Application filed Dec. 3, 1904.

The invention relates to a method of securing upon a metal, such as iron or copper, an adherent coating of a fusible metal, such a lead, tin or zinc. The coating is effected in a bath of the fusible metal in which a suitable proportion of a highly electropositive metal, such as potassium or sodium is maintained during the coating operation. Preferably also the metal, which is to be coated, is preliminarily subjected to the action of the electropositive metal end is thus cleaned and prepared for the coating before it enters the coating bath. The operation as described is carried out in a cast-iron vessel which contains a layer of the fusible metal, resting on the bottom, and a supernatant molten electrolyte, preferably the hydroside of an alkali metal. An anode of iron dips into the electrolyte and is fixed close to the surface of the molten metal. A wire of iron, copper or other metal which it is desired to have coated, is directed by suitably placed rollers through the bath of molten caustic alkali and the fused metal, it being moved by any desirable means. The cast-iron vessel in which the molten bath is contained, constitutes the cathode of this apparatus, and the electrodeposition of the alkali metal will therefore take place both upon the surface of the molten metal and upon the wire to be coated. The alkali metal rapidly diffuses in the molten metal, and thus serves to secure a closely adherent and uniform deposit of the coating metal. That portion of the alkali metal which is deposited upon the wire effectively cleans it by reducing any coating of oxide on the surface, on account of which operation the necessity of a preliminary extensive pickling or cleaning of the wire, previous to the coating, is avoided. The path of the wire through the electrolyte may be increased, if desired, in order to insure a thorough cleaning. An alternative construction of the apparatus shows a pot with two compartments, in one of which electrolysis is carried out as above, while the wire to be coated travels through the second compartment, which communicates at the bottom with the first one, so that the molten metal can freely pass over. The wire is thus not exposed to the cleaning action of the alkali metal previous to the coating operation. A third alternative construction shows also a two-compartment vessel, in one part only of which electrolysis is conducted, while the molten metal can freely communicate with the other compartment. The wire, however, in this instance, passes through the electrolyte in one compartment, before it is being coated, and passes out in the other compartment without coming there into contact with the electrolyte again after coating. The process is not limited to the coating of wire only, nor need it be a continuous one, the essential requirement being that a due proportion of the electropositive metal be maintained in the molten metallic bath during the operation of coating.

Electroplating Tank.—J. B. Brothwell and G. F. Penley, Torrington, Conn. Patent 781,327, Jan. 31, 1905. Application filed Jan. 11, 1904.

The tank is intended specially for the plating of small articles, and the essential part of the apparatus consists of a tumbling barrel in which the articles to be plated are rolled about while the plating is going on. The rotating tumbling barrel proper is mounted upon a revolving shaft in such a manner that it can be adjusted in horizontal or more or less inclined positions, as may be required during the operation. It is counterweighted so that it can easily be brought to the desired position. A flexible conductor in the form of a chain, which is connected to the wire from the negative pole of a source of current, drags through the solution and makes contact with the articles to be plated. The tumbling barrel contains more or less liquid according to the angle at which it is tilted, and it can be quickly emptied by lowering it. The articles can therefore easily be inspected and dumped out at any time, preferably by holding a sieve under the mouth of the vessel so as to catch the plated articles, and a tight vessel under it, for catching the electrolyte. This method of plating is stated to have the advantage that the articles are tumbled and finished, and that a more durable and lasting plating is obtained. The apparatus is also stated to have the desirable feature of exposing the articles more or less during the plating operation, which permits of their inspection, and it also does away with the laborious practice of stringing the articles on wires. The mechanical construction of the rotating parts is described in detail.

Apparatus for Nickel Plating.—J. W. Aylsworth, East Orange, N. J. Assignor to Edison Storage Battery Co. Patent 781,807, Feb. 7, 1905. Application filed Sep. 15, 1903.

The apparatus is primarily intended for carrying out the process described in Mr. Edison's patent of July 28, 1903, which was described in ELECTROCHEMICAL INDUSTRY Vol. I, p. 409, relating to the welding of a nickel film deposited on an iron or steel surface to the latter by heating the plated objects in a non-oxidizing atmosphere, for the purpose of making the articles thus treated practically homogeneous and relieving the tension existing in the plated deposit. Tension is made in the apparatus to carry out the above process in a continuous manner. Fig. 2 shows the apparatus in longitudinal cross-section, as applied to the plating and welding of a long, very thin end, finely perforated strip of steel, which is subsequently cut up into sections of the required length, and formed into pocket sections for receiving the active materials in the make-up of storage batteries of the Edison type. The thin strip is carried on a reel 1, from which it passes through a tubular testing chamber 2, which has a glass section 3 for insulating the top and bottom sections of the heating chamber from each other, and also serves to observe the temperature of the heated strip. From there the strip travels through pipe 4 into the plating tank 5, which is filled with a solution of nickel ammonium sulphate. The nickel anodes, preferably in the form of rectangular bars 33, are suspended from conductors 32 along both sides of the tank, by means of hooks 34. The strip forms the cathode, the source of energy for electroplating being shown at 30. The strip passes over conducting pulleys 6 at the top, all connected to the dynamo 30 by conductor 7, and over insulating pulleys 8 at the bottom, all pulleys being supported from beam 9. The strip then passes over pulleys 10 in washing 11, which contains water. It then travels through the heating chamber 12, which corresponds to the heating chamber 2 and has also a glass section 13. The strip then passes over a feed wheel 15 of suitable construction, actuated by mechanical means driven by motor 21 and is then wound on a reel 22. The strip is heated electrically by means of conductors 24 and dynamo 25 before plating, and in the circuit formed by another dynamo 25 and conductors, after plating. A non-oxidizing atmosphere in the heating chamber 12 and a reducing atmosphere in heating chamber 2 is provided by connecting hydrogen tank 26 with them, provided with a regulating valve 27. The heating of the strip in the chamber 2, and in the presence of hydrogen, effects a reduction of any oxide on its surface, so that it enters the plating both thoroughly deoxidized and cleaned. The strip passes through the plating bath at such a rate as to receive the required thickness of coating. After being heated to a welding heat in chamber 12, it is permitted to cool until its temperature is reduced below the oxidizing point. The apparatus is, however, not only claimed for as applied to nickel plating only, but also to electrolytic plating of any other metal, the adhesion or the character of which are improved by being heated to a welding temperature. The feature of continuous cleaning and electroplating may also be used for electroplating various metals, even if the heating feature is dispensed with.

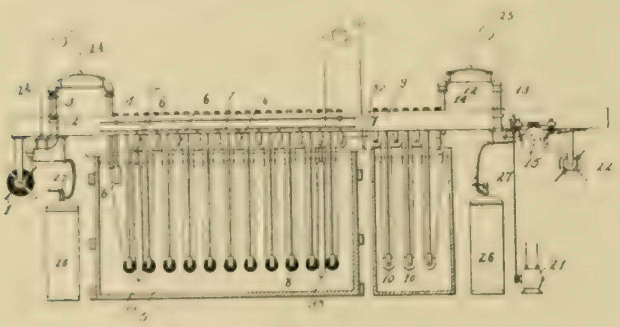


FIG. 2.—NICKEL PLATING APPARATUS.

Process of Reducing Pure Copper.—L. M. Lafontaine, Paris, France. Patent 782,145, Feb. 7, 1905. Application filed Sept. 12, 1903.

The inventor prepares a bath composed of a saturated solution of sulphate of copper free from iron or other sulphates, to which has been previously added eight or ten centiliters(?) of pure sulphuric acid for each liter of the solution, any preprecipitate being filtered off. About one kilogram of lampblack is added to each 500 liters of the solution, it being claimed that a more regular and dense deposit is obtained by this means. A soluble anode is placed in the bath, which anode for copper ores or pyrites is formed by constructing a compartment between two perforated partitions with closed bottom and top. The compartment is made of wood and is adapted to receive from 500 to 1000 kilograms of ore. A plate of copper ½ to 1 centimeter thick, and covering a surface equal to ⅜ of the width and height of the vat as put into the center of the compartment, the ore or pyrites being packed around as compactly as possible. In the case of the treatment of mattes, the latter are prepared by casting them into plates. Concentrates or dust are agglomerated by preparing a bath of concentrated quicklime, and when the lime is slacked introducing a quantity of concentrate in the proportion of 70 per cent of concentrate to 30 per cent of quicklime, the mass being stirred so as to form a paste or mortar. The object is to enable the dust produced in the pyrites and the ores to be utilized without roasting. It is stated that concentrates thus agglomerated with lime can be transformed into sulphate of copper until they are completely exhausted, without danger of disaggregation of the agglomerated anode. The mass is then molded in rectangular molds five or six centimeters thick, and corresponding in surface to the size of the vat. The cathode is always formed of a thin plate of pure copper one-half to one millimeter thick, and having a surface corresponding to the anode. Better results are claimed to be obtained when the electrodes are connected in tension.

STORAGE BATTERIES.

Negative Pole Plate.—P. Seeliger, Hagen, Germany. Assignor to Electric Storage Battery Co., Philadelphia. Patent 781,745, Feb. 7, 1905. Application filed June 18, 1903.

If substances such as finely divided carbon, kaolin or chinaclay are added to the active material for storage battery purposes, the shrinkage of the active material during continued use may be prevented. If sufficient of the inert material be added, an expansion of the active material may take place, resulting in increase of volume. The inventor has discovered that the loose structure thus produced in the mass is especially suitable for use in connection with active material for negative pole plates. The expansion of the material may continue until the original volume has been doubled or more, and by this action the loss of capacity of the negative plates during their life is prevented. The active material thus prepared expands from the plate, gradually loses contact, and may in a short time fall off and leave the grid bare, and in order to prevent this, the spaces of the grids which contain the active material are covered by perforated retaining plates. The receptacles or spaces of the grid are only partly filled with active material, so as to leave ample room for further expansion. Fig. 3 represents alternate constructions of plates embodying this principle, showing the perforated plates *a*, the grid *b* and the active material *c*, which can grow into the empty spaces *d*. A further series of alternative constructions embodying the same principle in its application to one plate or two plates superposed upon each other are described in the specification.

Storage Battery Electrode.—E. A. Sperry, Cleveland. Assignor to National Battery Co., Buffalo. Patent 781,795, Feb. 7, 1905. Application filed Jan. 27, 1904.

The electrode consists of a framework of vertical and horizontal bars, providing rectangular openings. The frame is preferably cast of antimonial lead in upper and lower half sections, which are burned together or connected by solder. Each of the rectangular openings contains a separate electrode section which is fastened to the frame at its upper edge only, thus leaving spaces all around between its side edges and the frame. Each section is wrapped over with strips of pyroxyline fabric, so as to retain the active material. The sections are retained in the frame opening by lugs which extend from the frame, and are so located that the bending of each electrode section produces a minimum deviation. The fabric cover of each section not only prevents adjacent electrodes from coming into contact, but serves as a cushion for the sections as they become expanded by use. This cushioning action is stated to be important in portable batteries as permitting the use of a lighter frame and one containing a higher percentage of antimony, and which is therefore more brittle, though it resists oxidation better.

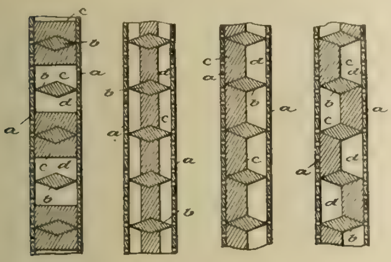


FIG. 3.—STORAGE BATTERY PLATE CONSTRUCTION.

Storage Battery.—J. Melzer, Cleveland. Patent 780,365, Jan. 17, 1905. Application filed April 2, 1904.

The invention aims at providing an apparatus which can be charged from a source of direct or alternating current. The cell is divided into two independent compartments by means of a plate of lead, which also constitutes the positive electrode. Both compartments are filled with dilute sulphuric acid. To one compartment a preferably bifurcated lead electrode is immersed, while a flat electrode of aluminium is similarly immersed into the compartment. When the cell is to be charged by direct current the aluminium electrode is cut out of the circuit and charging is effected by means of the two lead electrodes. When alternating current is used for charging, the central lead plate is disconnected, one terminal of the current being connected with the aluminium electrode in one compartment and the other with the lead electrode in the other compartment, the central plate serving as bipolar intermediary electrode during the operation.

GALVANIC ELEMENTS.

Battery.—Uri D. Foster, Meriden, Conn. Patent 781,817, Feb. 7, 1905. Application filed Sept. 30, 1904.

The invention relates especially to "medical" batteries, special means having been provided to secure the electrolyte from splashing out when the battery is carried around. For this purpose the electrodes are of special construction, with so-called "anti-splash" flanges.

MISCELLANEOUS.

Device for Preventing the Formation of Scale in Boilers.—A. Stewart, Chicago. Patent 779,326, Jan. 3, 1905. Application filed Feb. 12, 1904.

The device consists of a central tubular rod of metal on which are sitting at suitable intervals alternating discs of copper and zinc, separated from each other by metal collars. The zinc and the copper discs are stated, when thus arranged, to form the electrodes of a battery, of which the boiler water forms the electrolyte and the electric current thus generated is supposed to precipitate the calcuous matter in the boiler water and to prevent the formation of scale. The device is enclosed in a crate of wood, so as to keep it out of contact with the boiler shell.

Method of fastening abrasive material to metal bodies.—E. G. Case, Niagara Falls. Patent 779,639, Jan. 10, 1905. Application filed April 29, 1903.

The inventor aims at providing an abrasive material, the surface of which has materially increased wearing qualities and permanency. It consists in the mechanical application of an abrasive substance such as carborundum, or an equivalent material to the surface of a metal body, and at the same time subjecting the latter to an electroplating process. Thus as the abrasive material is applied to the body, a metal coating is deposited over and around its surfaces, which fixes the abrasive material permanently to the support.

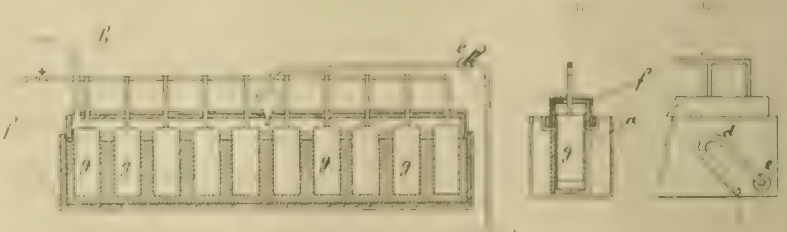
SYNOPSIS OF PERIODICAL LITERATURE.

A Summary of Articles Appearing in American and Foreign Periodicals.

INDUSTRIAL APPLICATIONS OF ELECTROCHEMISTRY.

Electrolytic Iron.—The *Zeit. f. Elektrochemie* of January 20, contains an article by S. Maximowitsch, of the laboratory of the Russian Bank Note Engraving Bureau in St. Petersburg. The object of his experiments was to get electrolytic iron which is both solid and flexible. Since the extreme hardness and brittleness of electrolytic iron is due to an occlusion of hydrogen, a suitable electrolyte should contain the hydrogen ions in as small a concentration as possible. He found ferrous bicarbonate solutions to be such an electrolyte. The starting salt is commercial $FeSO_4$, chemical purity being not required; to get a high conductivity, he applies $MgSO_4$; in order to change a part of $FeSO_4$ into $Fe(CO_3H)_2$, he used $NaHCO_3$. The Na salts do not increase the conductivity of the solution of the bath in the same degree as NH_4 salts, but in a bath which is of very high conductivity, the distribution of the lines of current, and, hence, the thickness of the deposit, lacks greatly in uniformity. The bath was made as follows. A vessel of six liters was filled with a solution of 20 per cent $FeSO_4 + 7aq$ and 5 per cent $MgSO_4 + 7aq$. The two electrodes had a size 20 by 15 cm., a wrought-iron plate was used as anode, while a copper plate was used as cathode. To facilitate the taking off of the iron deposit, the copper plate was slightly silver plated and iodized After the addition of 25 grams $NaHCO_3$ a dirty froth of yellow color formed on the surface which changed after three days into a brilliant brown film. The solution which was first muggy, got gradually clearer, while a voluminous deposit was precipitated. Twice a week $NaHCO_3$ was added in quantities of 20 to 25 grams; the total quantity of the bicarbonate used in four weeks was 186 grams. This would be sufficient to change 300 grams $FeSO_4 + 7aq$, or 25 per cent of the total quantity of the iron salt into carbonate. When protected by the ferric hydrate film against the action of the [illegible] the [illegible] solution remained absolutely clear, but [illegible] decomposed by the action of air and with great rapidity by boiling. The quantity of the dissolved $Fe(CO_3H)_2$ in a sample was 0.23 per cent. The deposit, as well as the film on the surface prevent the further decomposition of the bath, and should not be removed. The dirty color of the bath has no bad influence, but the electrodes should not reach into the sediment. Immediately after the first addition of $NaHCO_3$ the circuit was closed; to control the quality of the iron deposit, it was interrupted at intervals of two or three days. "The current density varied between 0.2 and 0.3 amperes." The first deposit was light gray and brittle. The deposit became more and more solid and flexible, until, finally, after four weeks, the bath was in its best condition. The iron deposit then obtained was specially suitable for plates for engravings. It reaches a tensile strength of 5180 kg. per sq. cm., and is at the same time so flexible that it can be bent to a sharp angle without showing a fracture. "The best current density is 0.3 amperes, maximum current density up to 0.6 amperes." The ampere-hour efficiency is 97 to 99 per cent. A bath made up as described will not give at once a flexible iron deposit; it is necessary to pass a current of moderate strength through it for a long while. It seems that a certain quantity of anodically dissolved iron in the bath is of great importance for the condition of the deposit. In one case a bath which had given good results ceased working properly after some time; the iron became brittle and full of holes. He then introduced pure carbonic acid into the bath; the result was very good, and the bath was restored nearly to its original good condition. In this way it is possible to maintain the bath for a longer time in proper working condition.

Chlorination versus Cyanidation.—The discussion started by W. E. Greenawalt's articles and noticed on pages 35 and 79 of this volume, is continued in the correspondence columns of the *Engineering and Mining Jour.* In the issue of January 12 P. Argall gives figures to show that the total cost of chlorination is much higher than that of cyanidation. In the issue of January 19 W. E. Greenawalt and J. W. Gibbs reply to a former letter of P. Argall. W. E. Greenawalt emphasizes that the possibilities of electrochlorination have not, by any means, been exhausted. He predicts that "within two years every chlorination plant, treating Cripple Creek ore, will be equipped with an electrolytic plant to generate its own chlorine; already, in at least one more of the large mills, they are preparing to generate their own chlorine by electrolysis." Concerning the small experimental chlorination plant at Colorado Springs, he (as well as J. W. Gibbs) says that "as an experimental plant, it was a qualified success, it did absolutely everything, from a technical view, that was claimed for it." The re-treatment of chlorination tailing, by cyanide, has failed wherever it has been attempted; on the other hand, the property of saving additional values, by an extra treatment, is not peculiar to cyanide. Concerning the plant in Boulder County, mentioned by Davis, in which electrochlorination was replaced by the cyanide process, it is said that this change was made after treating 75 tons of ore by a new management; "the reason why the chlorine did not extract the gold, was evidently in the roasting," since a new roasting furnace had just been installed. Although the change was made over a year ago, only 238 tons of ore were treated by cyanide; the mill was shut down soon after, and not a pound of ore has been treated since. "It is yet to be demonstrated that the changing of the process was not an unwarranted expenditure of the company's money." To Davis' criticism that Greena[illegible] [illegible]rlooks the difference between molecular and nascent [illegible] the latter replies that "in the chlorination of ores there is no difference; a few bottle tests will show this; even if nascent chlorine had a greater affinity for gold than molecular chlorine, it would have a greater affinity for the base elements also, and the sum total, in any process where the time is appreciable, would be the same." W. H. Davis replies in the issue of Feb. 9. He states that cyanidation, as practiced throughout the mining world to-day, leaves little room for improvement. No objection is raised by him to standard chlorination; "it is a time-honored process, and in some cases, no doubt, gives better results than any other process; but the same is also true of the cyanide process; conditions favorable to good extraction in either chlorination or cyanidation are much the same, namely, finely divided and free value; when these conditions are not present, neither process will extract a high percentage. Cyanidation has been applied to ores of low value for the reason that the process is less expensive to operate. But, it does not follow that cyanidation is restricted to ores of low grade; Cripple Creek ores prove this." He gives a few correcting notes on the Boulder County plant mentioned by Greenawalt. This plant was rigged, for a temporary test run, and it treated as such 238 tons; it was then decided to arrange the mill for cyanidation, but the company soon found itself in litigation, which caused the delay. This constitutes no proof of failure of the cyanide process. The writer strongly objects to Greenawalt's statement that in the chlorination of ores, there is no difference between molecular and nascent chlorine. He makes the following argument. When chlorine is set free from chloride compounds, it appears as nascent chlorine ("state A"); under normal conditions, it will combine with itself as Cl_2—if nothing more basic is present—this is "state B". This change from A to B liberates E units of energy, which is absorbed as heat by the solvent, or lost by radiation. Before this chlorine can combine with basic metals, it must again return to state A, which requires the expenditure of E energy units This, under working conditions of a chlorination plant, must come from the reaction between the chlorine and some metal in the ore—gold or otherwise. If the basic metal has a high solution pressure, like sodium or potassium, E would be readily supplied; but if the metal has a low solution pressure, like gold, E would become a commercial factor.



FIGS. 1, 2, 3.—DIAPHRAGM CELL.

Diaphragm Cell for Sodium Chloride Electrolysis.—In the *Zeit. f. Elektrochemie*, January 6, L. Rostosky, in a report on his recent visit to this country, describes the McDonald cell as used in the Standard Chlorination Works in Colorado Springs, Colo. (For a previous description of this cell see our Vol. I., page 387.) The writer was struck by the great simplicity of the arrangement of the cell room. In Colorado Springs there are 75 cells in three parallel rows. Each cell (Figs. 1 to 3) consists of a cast-iron tank, 0.3 m. high and broad, and 1.57 m. long. Two longitudinal partition walls of perforated sheet steel divide each cell into three compartments; these walls are covered on their inner side with asbestos paper, lined with cement, for the sake of greater strength. After about eight months the diaphragms must be renewed, since they gradually become clogged. The middle compartment contains the anodes *g* (Fig. 2), there being 10 anodes in each cell. The graphite anodes (Fig. 4a) are 30 cm. long and have a cross-section of 10 by 10 cm. After ten months they look like Fig. 4b, and are then removed from the bath and reversed (Fig. 4c) and are used for three or four more months. The cell is rendered gas-tight by slate plates *f*, 1 cm. thick. The chlorine gas passes off through the lead pipe *b*. The supply of sodium chloride solution was formerly automatic, but since this was not successful, it is now regulated by hand by means of the cock *c*. The temperature of the electrolyte is said to be 85° to 90°. The concentration of the lye, flowing out off the cathode compartment, increases from 7 per cent to 18 per cent caustic soda, and decreases to 6 per cent sodium chloride, so that finally the ratio of caustic soda to sodium chloride is 3 to 1. The caustic soda solution is removed by the bent tube *d*, which may be turned around *e*. When the diaphragm is new and perfectly permeable, this tube is adjusted to a high position, and the level in the anodic compartment is approximately equal to that in the cathodic compartment. Later on, the tube is turned gradually to a lower and lower position, so that the height of the electrolyte in the cathodic compartment is only a few centimeters, and the poorer permeability of the diaphragm is counteracted by the mechanical pressure of the anode liquid. At the same time it seems that in this way the hydroxyl ions are prevented from passing into the anode compartment.

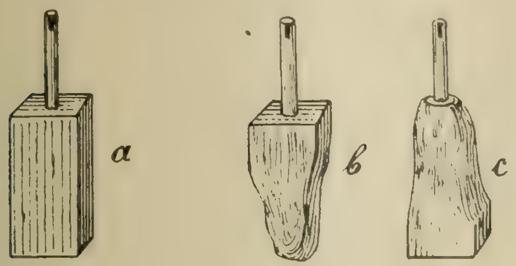


FIG. 4.—GRAPHITE ANODES.

The Electric Furnace in the Iron and Steel Industries.—The well-known Canadian Commission report is the object of some criticisms by J. O. Arnold in (Lond.) *Nature*, January 12. Concerning the experiments made with the Keller electric furnace at Livet on the reduction of pig iron from ore, it is said that "the commission saw smelted several tons of pig iron, as a rule remarkably high in managanese (1.5 per cent to 4 per cent), and hence of limited commercial interest" * * * "From a British point of view Kjellin's induction process deserves the most serious attention in view of (under certain conditions) its probable competition with the crucible steel process. Analytically, mechanically, and micrographically this steel leaves nothing to be desired, but unfortunately chemical and tensile tests, and the indications of the microscope, have a limited value in determining the working capabilities of tool steel." The author urges that evidence as to the equality of electric steel to crucible tool steel on a commercial scale "can be conclusively obtained only by at least two comparative years of shop practice, employing all kinds of tools, and recording the average wear and waste of the steels as evidenced by the ratio between the work turned out and the annual cost of the tool steels purchased." * * * "Prolonged test on Kjellin steel of all carbons, compared with similar crucible steels, have been inaugurated at the University College of Sheffield, and the erection of a Kjellin furnace capable of making one ton of steel per day is under consideration." The writer speaks of some conclusions of the commission as "hasty and ill-digested," although he "cordially" congratulates Héroult, Keller and Kjellin "on the scientific ability displayed in the development of their several methods, all of which, within their legitimate spheres, are undoubtedly of great metallurgical value."

Electric Resistance Furnace.—The January issue of *Elektrochemische Zeitschrift* contains the conclusion of the article by J. Bronn on electric smelting of glass. The author first gives some brief notes on various patents which have been taken out in connection with this subject, and then describes in greater detail various electric resistance furnace constructions (crucible furnace, muffle furnace), which are heated by means of the granular resistance material Kryptol.

Electro Zinc Plant.—Electrogalvanizing continues to make steady progress, the process being largely used for two purposes, viz., as a protection for steel tubes for boilers during the time of assembling and erection, and also for the purpose of detecting flaws in the tubes in the early stage of construction, as it is found that the tubes, after pickling, if flashed with a thin coating of zinc, readily show any inherent flaw. One of the latest plants is described in the Lond. *Elec. Rev.*, January 6, the process being that of Cowper-Coles; it is installed at the works of the Aktiebolaget Aandviken, Finland. The plant consists of a galvanizing tank of 20 feet long, which is connected to two regenerating tanks, in which zinc dust, mixed with coke, is placed for replenishing the zinc in the electrolyte, as it is electrodeposited on the iron plates. A lead-lined centrifugal pump, driven off the dynamo shaft, is used for circulating the electrolyte. The anodes employed in the Cowper-Coles regenerating process are lead, attached to compound anode and cathode bars.

Calcium Cyanamide as Fertilizer.—In a recent address before the Farmers' Club in Berlin, a reprint of which we received from the author, Prof. Frank gives some notes on the production of calcium cyanamide and its use as fertilizer (see also page 79 of our last issue). In the manufacture on a commercial scale one electric hp.-year "fixes 250 kg. of nitrogen in form of about 20 per cent calcium cyanamide;" the watt-hour efficiency is about 33 per cent, and it is expected that it may be improved. The quantity of cyanamide required as fertilizer on one hectar (1 hectar = ca. 2½ acres), is 150 to 300 kg., corresponding to 30 to 60 kg. nitrogen. In order to avoid irritations of the workingmen by dust, the cyanamide is mixed with the double quantity of dry soil. The fertilizer is to be applied about a week or two before sowing the seed, and must be mixed with the soil 3 to 5 inches below the surface.

Production and Utilization of Ozone.—The long article by J. B. C. Kershaw is concluded in the Lond. *Elec. Rev.*, January 6. The author describes briefly the experimental plant of Siemens & Halske at Martinikenfelde, near Berlin, and the commercial plants at Wiesbaden and Paderborn for the sterilization of drinking water by ozone (see our Vol. I, p. 179). The author reaches the following conclusions. The use of ozone for water purification and sterilization has emerged from the experimental stage of its development, and under certain conditions this method of purification can be operated with successful results, both commercially and otherwise. The conditions required to make such an installation a success, however, are exceptional, and it seems unlikely that ozone, or, more correctly, ozonized air, will be employed on a very extensive scale for sterilizing the water supply of great cities. It is both cheaper and safer in such cases to go far afield for water, and to obtain a pure supply, than to take the polluted river water that happens to be near at hand, and to purify it by treatment with ozone in special purifying plants. But in small towns and villages, where the initiation of large waterworks schemes is impossible, owing to the financial outlay involved, the ozone treatment for purifying and sterilizing an impure water supply near at hand would seem to have a future field of usefulness. The Schierstein and Paderborn works are examples of such installations. Siemens & Halske estimate that a plant capable of dealing with 126,000 cubic feet of water per day could be erected for an outlay of $32,000, this cost covering all the pumping and filtering plant of the waterworks—and installations half this size would, no doubt, prove sufficiently large for many small town and village communities. The possession of an ozone sterilization plant, as a stand-by in times of severe epidemic of typhoid, enteric and similar fevers, would also be an aid to the preservation of health in large city communities, since at such times the ozone plant could be put into use, and the water supply to the affected district of the city could be thoroughly sterilized. The purification of water by ozone would also appear to have a future in works where drinks and beverages of all kinds are manufactured for public use, and possibly also on board ships and in military camps, where the necessary supply of steam or electricity for generating the current can be obtained. While the application of ozone to water purification is unlikely, therefore, to revolutionize the present system of obtaining water for drinking and other purposes in our great cities, it will, undoubtedly help in certain cases and under certain conditions to solve the problem of a pure water supply, and as the yield of ozone is increased by improvement in the form and working of ozonizers, this field of usefulness is likely to widen and extend. Since the theoretical yield of ozone per e. h. p. hour is stated to be 1000 grams, and the miximum yet obtained is 184 grams, or an efficiency of only 18.4 per cent, there is still ample room and opportunity for improvements in ozonizers; and the progress in the ozone industry is likely to depend very largely upon the success of inventors in turning to useful work some portion of the 82 per cent of electrical energy still unaccounted for in the usual form of ozonizer tube and apparatus.

THEORETICAL AND EXPERIMENTAL.

Vanadium, Niobium, Tantalum.—In our last issue we gave a resumé of W. von Bolton's work with tantalum which has resulted in the construction of the tantalum incandescent lamp by the Siemens & Halske Co. Von Bolton now gives a summary of his researches in the *Zeit f. Elektrochemie*, January 20. When investigating the conductivity of solid electrolytes. Nernst succeeded in maintaining magnesia rods of uniform cross-section for a long time at white incandescent heat, by passing the electric current through them. Electrolysis takes place whereby the magnesium oxide is decomposed into magnesium and oxygen. But both combine immediately again to magnesium oxide. If such a Nernst lamp filament is placed in a vacuum, the oxide is strongly pulverized, the oxygen being given off and recombining with the magnesium at snort intervals until the filament burns through at some place. It is the same with the other oxides which may be used in the Nernst lamp. Von Bolton found that vanadium, niobium and tantalum oxides behave in a quite different way. He first experimented with vanadium; he made small rods of the brown vanadium oxide, mixed with paraffin, and heated them, embedded in granular carbon, to a temperature of about 1700° C., whereby the paraffin evaporates, while compact rods of trioxide remain in the original shape. Rods were thus made, 20 mm. long, with a diameter of 0.8 mm., and placed in a vacuum, and while evacuation was continued, these rods were heated electrically with a current of 1.8 amperes at 42 volts; they then became white hot and gave off much gas which was collected in a separate vessel and proved to be oxygen. The rods had assumed a metallic gray color. The walls of the vacuum vessel had been darkened with a brown powder, due to the evaporation of the oxide. The reduced vanadium filaments were then brought into a new vacuum and their melting point was determined photometrically by Lummer's method. The filament melted at moderate white heat and consumed in the average 3.5 watts per candle, giving 0.42 candle per square millimeter surface, which corresponds to a melting point of vanadium at 1680° C.

Similar experiments were made with niobium. Its melting point was found by a corresponding method to be 1950° C. Finally tantalum was experimented with, which was found to be the most suitable for an incandescent lamp, since the [illegible] point is between 2250° and 2300° C. The author first [illegible] some historical notes on the preparation of tantalum and [illegible] especially the experiments of Berzelius and Rose. [illegible] produce tantalum by heating the potassium and tantalum fluoride (K_2TaF_7) with potassium, and extracting the potassium fluoride with water, but this method does not yield pure metallic tantalum, but contains only about 5 per cent of the pure metal. Moissan described in 1902 the production of tantalum in the electric furnace by reducing the oxide with carbon; but his metal contained about 0.5 per cent carbon and was therefore either a carbide (0.5 per cent C. being a pretty large quantity in view of the high, atomic weight of tantalum) or it was an alloy of the metal with carbide. Von Bolton describes briefly two methods of producing pure metallic tantalum.

The first method is electrolytic reduction of one of the oxides in the incandescent state in a vacuum. He formed filaments of brown tantalum tetroxide and placed them in a vacuum. By passing the electric current through, they were caused to glow slightly, whereby much air was driven out of the porous filaments. Evacuation was then continued and the temperature of the glowing filament was increased until, at beginning white heat, several points of the filament showed strong incandescence. These points gradually extended to longer and longer lines, until the filament was incandescent uniformally along its whole length. During this reaction much gas was given off, which was identified as pure oxygen. The brown color of the original oxide had changed into pure metallic gray, and the wire had become flexible after several hours of strongest white incandescence, like a wire of copper.

The second method, which is the more practical one, is to produce tantalum by the method of Berzelius and Rose mentioned above, and then to purify and refine the metal in the vacuum under the action of the electric arc. This method is based on the fact that the oxides of tantalum melt more easily than the tantalum itself, and on the other fact, that they are much more easily pulverized in a vacuum than the metal, so that the metal can be completely separated from the oxides.

The specific heat of tantalum is 0.0365, the atomic heat 6.64, so that the law of Dulong and Petit is valid. The specific weight is 16.5. The author gives some further numerical data on various chemical and physical properties of tantalum. The tantalum lamp has a useful life of 400 to 600 hours, during which it consumes 1.5 watts per cp. Useful life means the time at the end of which the lamp has lost 20 per cent cp. In the first hundred hours, the cp. increases whereby the specific energy consumption decreases to about 1.3 watts per cp., it then increases, and at the end of its useful life the lamp consumes 1.8 to 2 watts per cp., but the lamp continues to burn up to 1000 and 1500 hours. Since the ordinary carbon filament lamp consumes 3.5 watts per cp., the tantalum lamp consumes less than half the energy for the same useful life. The author finally gives some data on reactions between the tantalum and various elements, and on the properties of alloys of tantalum. The investigation has consumed several years and the co-operation of nine investigators is acknowledged by the author. The importance which Siemens & Halske attribute to this investigation is indicated by the fact that they have taken out or applied for about 200 patents in Germany and other countries with about 1000 claims.

Alternating Current Electrolysis.—The paper of Brochet and Petit, recently abstracted in these columns, is the subject of two articles in the *Zeit. f. Elektrochemie*, January 6, in which M. Le Blanc and R. Ruer offer various criticisms. The former promises a complete summary of his own researches for the furture, while the latter objects to the theory of Brochet and Petit, concerning the solution of platinum by alternating current, and describes some experiments to refute their theory and to conform his own proposition that the solution of platinum in sulphuric acid by alternating current in the presence of an oxidizing agent is due to the fact that "the oxidizing agent weakens the cathodic component of the alternating current and thus changes the symmetrical alternating current into a non-symmetrical one with a preponderant anodic component."

METALLURGY.

IRON AND STEEL.

Dry Air in Blast Furnaces.—A Pourcel has discussed the results of Mr. Gayley's experiment, in *Revue de Metallurgie* for January. He repeats the mistake made by Le Chatelier, in supposing that the composition of the slag was charged, made less basic, and in consequence more fusible and requiring less heat for its fusion, and thus the heat requirement of the furnace was diminished, when working with dry air. That there is no foundation for this view is evident from the fact that Mr. Gayley did not change the proportion of limestone to ore during his experiments; he did, however, use less coke with dry blast, and that would bring in less ash, and therefore diminish the total quantity of slag per ton of pig iron made, while its basicity would be increased by the absence of some coke ash. Altogether, there was a slight saving in heat requirement, due to the smaller amount of slag made per unit of pig iron. Pourcel further discusses the heat saving as produced by increasing the temperature of the crucible and decreasing that of the upper part of the furnace, but he does not give any very convincing reasons why the furnace should act that way with dry blast. He concludes that the idea of drying the blast is not likely to make progress in Europe until a less complex and less onerous method of drying it than that used by Mr. Gayley has been invented. Many metallurgists disagree with this conclusion: *Nous verrous.*

The Utilization of Blast Furnace Gases.—Charles de Mocomble has given us in *Revue de Metallurgie*, for January, a most admirable treatise, occupying sixty quarto pages, on this interesting and important subject. He devotes ten pages to the general considerations of the operation of a blast furnace, in which the waste gas is stated to vary from 18 to 36 per cent in carbonic oxide, and in calorific power between 800 and 1150 calories per cubic meter; it is also assumed that on an average the calorific power is 900 calories; that 3.5 cubic meters of blast are needed for each kilogram of fuel charged, at a temperature of 700° C., and that 4.5 cubic meters of waste gas are formed, leaving the furnace at a temperature of 150° C.

Chapter II. consists of fifteen pages devoted to the utilization of the gases to heat the blast, by means of hot-blast stoves. A table shows the content of dust in the gases to vary between 1 or 2 grams and 50 grams per cubic meter (all above 20 grams were when producing spiegeleisen), and to average about 5 grams. The gas being free from dust, it is burned to heat the stoves; and the writer calculates how much gas is needed to heat the blast used up to 800° C., assuming that the gas is used at 150° C., the theoretical amount of air is used for combustion, and that the stove has an efficiency of 95 per cent, *i. e.*, that the stove losses only 5 per cent of its heat by radiation, conduction, leaky valves, etc. There are two mistakes in this calculation; the efficiency of stoves averages more nearly 75 per cent than 95 per cent, and the mean specific heat of air per kilogram between 15° and 800° is 0.251 instead of 0.237. The writer, in fact, finds that the actual efficiency, as shown by some tables which he has compiled from actual practice, is between 56 and 85 per cent, and he therefore disregards his own assumptions and proceeds with 60 per cent as an actual value for the heating efficiency of the stoves, which requires 0.4 cubic meter of gas to be burned in the stoves per cubic meter of blast heated to 800° C. (instead of the 0.29 cubic meter which his own assumptions and calculations led to). This would require 1.4 cubic meters of gas to heat the 3.5 cubic meters of blast needed for each kilogram of fuel burned, leaving 4.5, 1.4 or 3.1 cubic meters of gas to be otherwise used, equal to 69 per cent of the total gas produced. This figure is about right; a modern furnace uses only about 35 per cent of its gas in the hot-blast stoves, and has 65 per cent, for other purposes. Mr. de Mocomble assumes for his further discussions that practically 35.5 per cent of the gas is needed for the stoves, and 64.5 per cent is otherwise available.

In chapter III, ten pages are devoted to the use of the excess gas in boilers. A table of tests made by different observers between 1891 and 1904 shows a utilization of from 33 up to 75 per cent of the calorific power of the gas burned. The best boilers can scarcely maintain an average of 70 per cent under actual working conditions, and the type of boiler used around blast furnaces is usually so far from the best, that the writer feels compelled to assume a utilization of 45 per cent as representing average blast-furnace practice. This figure leads to the production of 0.56 kilogram of steam at 4 or 5 atmospheres pressure for every cubic meter of gas burnt under the boilers. While the best steam engines will supply one horse-power hour for 4 kilograms of steam at this pressure, the writer assumes that the average blast-furnace engine will require 10 kilograms, which leads to the necessity of burning 18 cubic meters of gas per hour for each effective horse-power produced. It is then calculated that the furnishing of the power for compressing the blast will require, per kilogram of coke burned, 1.85 cubic meters of gas (assuming that the piston displacement is double the air actually received by the furnace and that the pressure of the blast is 0.4 atmosphere, or 6 pounds per square inch). This requires 41 per cent of all the gas made by the furnace, leaving 64.5 — 41 = 23.5 per cent otherwise available. The writer has not, however, allowed for the power necessary for raising the charges, pumping water, running a pig-casting machine, lighting plant, etc. In many works these items use up all the remaining surplus.

In the fourth and last chapter, of twelve pages, the use of the 64.5 per cent surplus gas in gas engines is discussed. Tables showing results already obtained prove that from 2.5 to 5 cubic meters of gas have been used per hour for producing one horse-power; that the requirement of large engines is now not over 2.5 cubic meters, giving a mechanical efficiency of 25 per cent on the heat value of the gas used. For the purposes of his calculations, the writer assumes a consumption of 3 cubic meters per hour per indicated horse-power, and 4 cubic meters per hour per effective horse-power of the blowing engine, corresponding to a net efficiency of 17 per cent. Since the boilers and steam engine required 18 cubic meters, the gas engine is 4.5 times as effective. Since there is available 3.1 cubic meters of gas per kilo of coke burned, there is producible 3.1 ÷ 4 = 0.775 horse-power for every kilo of fuel burned per hour in the furnace. Of this power, 9 per cent is required for the blowing engines (0.070 horse-power for every kilo of fuel burned per hour) leaving available for all other purposes, in round numbers, 0.7 horse-power for every kilo of fuel charged per hour. Since, in a furnace producing 100 tons of pig iron daily, there is used about 4000 kilos of fuel per hour, there is a 2800 horse-power developable by gas engines over and above the power requirements of the furnace. [The subject above noticed is one of the most rapidly-advancing branches of the metallurgy of iron, and bids fair to realize all predictions and expectations. The abstractor pointed out the possibilities in this direction by a series of calculations printed in the Journal of the Franklin Institute for December, 1900.]

On account of limitations of space, a number of abstracts had to be reserved for our next issue.

Artificial Rubies.—A recent consular report states that artificial rubies have been produced in France by "reducing small, natural rubies into a very fine powder, which is melted in an electric furnace, cooled rapidly, and crystallized. The product obtained, from what was of little worth, on account of minuteness, possesses a comparatively high value. The main difficulty encountered is to prevent cavities and fissures in the crystals. The new process cannot be employed with emeralds and sapphires, as they become discolored by the action of the heat."

RECENT METALLURGICAL PATENTS.

Roasting Furnaces.

R. I. [illegible] and P. Thill (781,834, February 7) object to the [illegible] number of excessive hearths in an ordinary roasting furnace, on account of the excessive quantity of flue-dust produced. They therefore provide a helical continuous hearth (Fig. 1); to effect the continuous progression of the finely divided ore, stirrer arms are arranged in each of the spaces between the turns of the helix. The inventors provide for a rotary reciprocation of the arms and arrange for their following the pitch of the helix in their downward movement for a little more than half a revolution (if the arms are angularly displaced 90° apart). At the end of their stroke they are first raised vertically a sufficient distance to clear the body of ore on the hearth, and are then rotated backward, being depressed at the beginning of the stroke, until they again engage the ore on the hearth. To diminish the amount of power required, the inventors counterbalance the feeding devices, so that as one of the furnace-operating mechanisms is lowered the counter-weight is raised; the mechanism is supported on hydraulic cylinders.

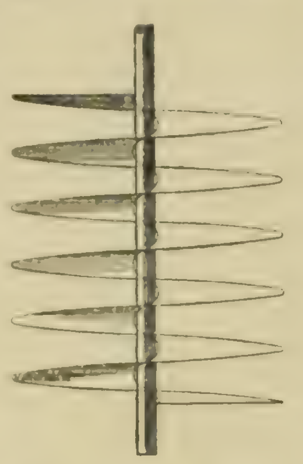


FIG. 1.—HELICAL HEARTH OF ROASTING FURNACE.

F. Heberlein and W. Hommel (781,824, February 7) patent a horizontal rotary muffle furnace of circular construction, the principal object being to prevent the furnace gases mingling with those evolved from the ore. Immediately below the ore-chamber, the floor of which rotates, is a heating chamber which rotates with the floor, while a stationary heating chamber is provided immediately above the ore chamber. The heating gases circulate first through the lower, and then through the upper heating chambers, these chambers being divided into compartments, so as to direct the passage of the gases.

G. O. Petersson (781,904, February 7) patents a roasting furnace, shown both in longitudinal section and in cross-section in Fig. 2. It consists of a shaft *a*, contracted at the bottom and traversed by a number of arches, *b*, *c*, *d*. The space beneath the bottom arch at one end communicates with the outer air through passages *e*, arranged in one of the abutment walls of the arch, while the other end is connected with a gas-fire chamber and a chamber *f*, arranged beside the latter in the other abutment-wall. The gas is supplied to the fire-chamber through a pipe *g*. The space beneath the arches *c*, situated above *b*, communicates at one end with *f* and at the other end with *h*, arranged in the opposite abutment-wall and communicating at its upper part with one end of the space beneath the arches *d*. This space at the other end communicates with chamber *i*, which connects with the chimney *k*. The air used for the combustion of the gas will thus be heated by the material to be roasted in the lower part of the furnace as the air passes to the chamber *f*, and the products of combustion subsequently flow back and forth through the material to be roasted on their way to the chimney.

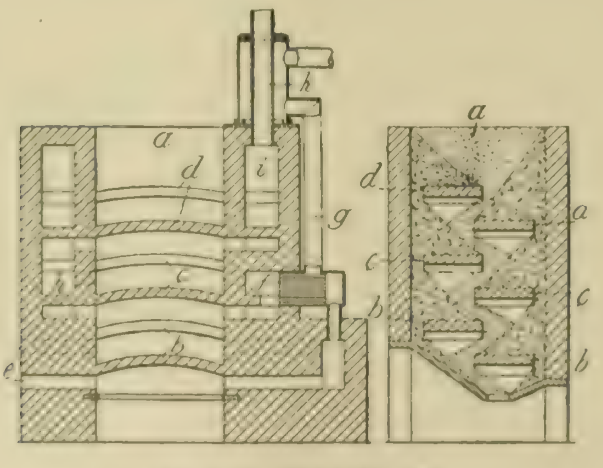


FIG. 2.—ROASTING FURNACE.

Copper.

Certain native copper-bearing rock carries the copper so finely divided that processes of mechanical concentrations are unsuitable. F. R. Carpenter (781,807, February 7) claims that such ores may be cheaply treated by a smelting process, if a carrier or collector for the copper is used. He therefore adds some sulphide material, such as iron, pyrite, pyrrhotite, etc., which collects the finely disseminated native copper. Clean slags are made with the use of limestone. The copper conglomerate, pyrite and limestone are charged with sufficient coke into a blast-furnace; there is formed a very light slag and a very high-grade matte, the former free from copper and the latter containing the copper in form of sulphide, since the copper which was originally free has united with the sulphur, displacing the iron.

Vanadium.

There exists in many localities ores of vanadium, mainly sandstones, having the general composition 0.5 to 5 per cent vanadium, 74 per cent silica, alumina, etc. Notwithstanding the high temperature required for the reduction of vanadium compounds and the readiness with which it reoxidizes when reduced, F. R. Carpenter (781,808, February 7) claims that such ore "may be smelted even in an ordinary blast furnace, provided the fuel and blast are arranged so as to produce a high temperature and a strong reducing action, and a carrying metal is present with which the vanadium may alloy." As such a carrying metal, iron may be used, which is introduced into the charge in the form of iron oxide. In order to slag off the large amount of silica present in the ore, a basic flux, as dolomite, is added. The vanadium is obtained in form of ferro-vanadium, usually containing also silicon.

Alloys.

T. Prescott (781,300, January 31) patents an alloy of a light and strong character, suitable for pattern-plates or patterns for foundry work, machine-bearings, etc. It consists of 53.50 to 74.75 parts of zinc, 25 to 43.5 parts of aluminium, 0.25 to 2 of iron, 0.25 to 1 of silicon.

L. H. E. Lacroix (782,401, February 14) patents an alloy intended to serve as a substitute for lead for most of the purposes for which the latter is used, while it has the advantages of greater strength for the same thickness. "Thus, for instance, if the new alloy is used for roofing in place of lead, the beams and other parts of the structure can be made lighter. The alloy is more easy to treat than lead and can be rolled into exceedingly thin sheets and drawn into very fine wire." The alloy consists of 1000 parts lead, 15 antimony, 1 sodium.

Zinc.

C. E. Dewey (781,133, January 31) patents the following process of treating zinc-sulphide ores containing iron. The ore is first roasted in such a manner that the zinc is converted into zinc sulphate and zinc oxide and the iron present converted into ferric oxide as far as practicable. The roasted ore is charged into a V-shaped tank with water, and is there held in suspension by the introduction, at the bottom of the tank, of a mixture of air and sulphurous gas from the roasting furnace; or a mixture of steam, air and sulphurous gas may be introduced. Thereby "the zinc oxide present is dissolved and converted into zinc sulphite, and through a reaction with the ferric oxide present in the ores, the zinc sulphite is converted into zinc sulphate, in which form the zinc is removed from the residue of the ore by filtering, decantation, or in any other suitable manner." The ore is then washed in order to remove the balance of the zinc sulphate, and enough of the latter is removed with the wash-water to form the latter into a weak solution of zinc sulphate, and the subsequent charge of roasted ore is introduced into this weak sulphate solution instead of pure water, as in the case of the first charge.

Cyanide Process.

A. H. Brown (781,711, February 7) proposes to reverse the order of operation in ordinary cyanide practice, by beginning with cyanidation and following this up by concentration. He objects to the ordinary practice for the reason that in case of most gold and silver-bearing ores the use of water in crushing by stamps or rolls or in connection with the concentrating process occasions much loss of values due to sliming, since the recovery of the values from the slimes requires an extensive system of settling tanks beyond the limits of ordinary plants. In Brown's process it is first subjected to the action of a cyanide solution whereby the finer values are dissolved, and second to concentration whereby the coarser values are recovered. The whole operation may be briefly described as consisting of the following five steps: (1) pulverization of the ore in the presence of cyanide solution; (2) hydraulic classification by the introduction of cyanide solution at the bottom of an overflow-tank to produce an ascending current; (3) leaching the ore by the use of cyanide whereby the finer values of the ore are dissolved; (4) precipitation of the dissolved metallic values from the solution; (5) treatment of the residue of ore by concentration.

Barrel-Filter.

H. C. Holthoff (782,031, February 7) patents a barrel with a barrel arranged centrally or co-axially, so that the balance of the barrel is maintained and less power is required to rotate it. Fig. 3 is a cross-section; *a* is the barrel, in the center of which is arranged the cylindrical filter frames of lead or porcelain, which is composed of inner or outer shells *i* and *i*, having in the outer side of the inner shell and in the inner side of the outer shell, cavities which coincide with each other and from cells *k*, the opposite outer and inner walls of which are perforated with small holes *l*. The component shells of the filter frame are made in segmental sections, bound together and held in place upon the lead-covered rods *g* by steel or iron rings *m*, encased in lead. The cavities or cells *k* are filled with coarse sand, crushed silicious stone or other suitable granular material.

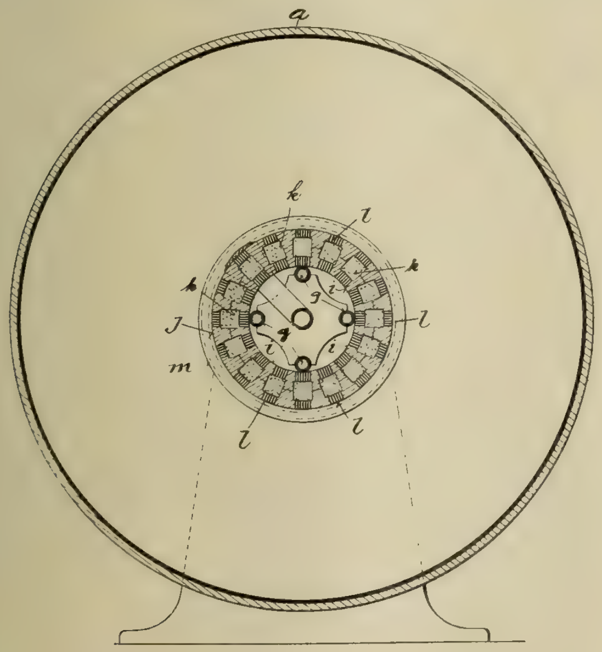


FIG. 3.—FILTER.

On account of limitations of space, notes on a number of patents which were granted during the past month had to be reserved for our next issue.

BOOK REVIEWS.

Techno-Chemical Analysis. By Dr. G. Lunge; authorized translation by Alfred I. Cohn. 12mo., viii. + 136 pages, 16 figures; cloth, $1.00. New York: John Wiley & Sons, 1905.

This compact and yet comprehensive little book is written by a past-master in technical chemistry, and contains information and advice of value to anyone having to determine the quality of raw materials or products of chemical processes, or to control the steps of a chemical manufacture. The scope is best disclosed by an enumeration of the principal headings: Technical Gas-Analysis, Gas-Volumetry, Fuels and Heating, Examination of Water, Sulphurous and Sulphuric Acids, Nitric and Hydrochloric Acids, Salt, Soda, Sulphate Cake, Carbonates and Bi-Carbonates, The Chlorine Industry, Potassium Salts, Clay and Cement, Artificial Fertilizers, Gas and Ammonia Manufacture, Calcium Carbide, Coal-Tar Industry, Mineral Oils, Oils and Fats, Soaps, Glycerine, Sugar, Alcohol, Vinegar, Wine, Beer, Tanning and Dyeing.

Within the scope of the subjects named and the limits of size of the book, the matter is most satisfactorily handled. The information is not elementary, yet very clearly presented. The work can be satisfactorily used by any one who knows the rudiments of chemical manipulation, and at the same time will be found useful by the expert analyst.

Notes on Assaying and Metallurgical Laboratory Experiments. By Richard W. Lodge: 8vo., viii. + 287 pages, illustrated; cloth, $3.00. New York: John Wiley & Sons, 1904.

Nearly two-thirds of this work is a very clear and superior treatise on assaying. Complete explanations are given concerning the assay for silver, gold, lead, bullion, copper, tin and the platinum metals. The methods used by the author in the Massachusetts Institute of Technology are described first, and are followed by other reliable methods. The method of presentation is entirely up to date, and contains all those little touches which evidence that the work is perfectly familiar to the author, and he puts it in such manner as to make it easy to the reader or student.

The latter third of the work describes the metallurgical laboratory test, made by fourth-year students in the author's laboratory. The processes chosen are Plattner's chlorination process, the cyanide process, bromination and milling of gold ores, retorting amalgam, chloridizing of silver ores and pan amalgamation. The directions given are complete, with almost German-like attention to detail, and form a very helpful guide in the restricted part of metallurgical laboratory work which they cover. The book, as a whole, is particularly a work for students, and gold and silver metallurgists.

Kalender fuer Electrochemiker, sowie Technische Chemiker und Physiker. 1905. By Dr. A. Neuburger. Berlin: M. Krayn. Price, 4 marks.

The Kalender appears this year in its ninth edition. The book undoubtedly contains a great deal of useful information in a concise form. During the last years it has been the [illegible] constantly improve the contents, and the book bears evidence of this at many places. That among the mass of miscellaneous data thus collected, there should be more or less inaccuracies is not surprising. One part of the book, which is evidently in need of revision and of being brought up to date, is that dealing with technical electrolysis and electrothermics. The price for electric energy, for instance, at Niagara Falls, is given as 3.2 centimes per kw.-hour for a consumption of at least 80,000 kw.-hours per month. This, for 8760 kw.-hours or a kw.-year figures out at $56, a price far above that actually charged. The surprising statement is also made that the Conley process is carried out in this country "on a large scale," and that "pig iron, cast iron and steel are manufactured by it directly" from the ore. The chapter should be revised by a practical iron metallurgist, as far as the electrometallurgy of iron and steel is concerned, so as to avoid statements as the following,, which is translated literally: "One can manufacture in one and the same apparatus, at the same time, all possible kinds of cast iron, iron, and steel," and others of the same character.

The Polishing and Plating of Metals.—By Herbert J. Hawkins. Chicago: Hazlitt & Walker. Price, $2.00.

The author of this new book on electroplating modestly states that he is not seeking to enlighten the skilled men of the profession or to air his own opinions, but to help the younger element of the trade. The book is a typical trade publication, and is of value in giving the experience of a practical plater conversant with modern shop practice. A commendable feature is the omission of historical matter and the large number of impractical formulæ for solutions which usually burden books on electroplating.

Mr. Hawkins does not pretend to treat the subject from the scientific standpoint, and for justification says that "the science of electroplating is so new and changing so rapidly that writers on the subject are comparatively few, and those we have are generally students rather than practical men." That the book is not scientific, the reader must agree. It is admitted by the author that "a general knowledge of chemistry pertaining to plating should be very beneficial to the practical plater and a thorough knowledge of dynamos and electricity should also be of value," but he says that such knowledge is easily "obtainable at small cost in this age of night schools and correspondence schools."

The first one of the seventeen chapters points out defects of previous works, discusses different kinds of platers and states the author's views as to a practical manual. The second chapter presents "some points on hygiene for platers" and from the author's personal experience with boils, sores and other afflictions arising from cyanide and "potassium poisoning" a warning is given against drinking from vessels which have contained cyanide solutions and unnecessary dipping of the hands in such plating baths. We are also told that there is no valid ground for the belief that a plater should be a user of tobacco or alcohol. In spite of the dangers that lurk about the profession of plating he finds it most attractive, comparing his love for it with a sailor's love for the sea, and concluding with the opinion that "once a plater, always a plater."

Those parts of the book which deal with the polishing room, its machinery and management, acid dips and pickles, their composition and use, the arrangement and management of the plating room and methods of setting up and operating gold, silver, nickel, copper and brass plating solutions, is thoroughly practical and should prove of value to the student of electrochemistry, as well as to apprentices in the plating business. It is noticeable, however, that Mr. Hawkins pays but scant attention to the use of the sand-blast for the treatment of work preparatory to plating, though this process is [illegible] employed for cleaning, as well as for finish[illegible].

The chapters dealing with the construction and management of dynamos and electrical measuring instruments should be revised to free them from various misstatements. Some of the author's views relative to the laws of electricity and magnetism are startling revelations to the "student," however helpful they may be to the "practical man." The ampere is defined as that quantity of current which will pass through one ohm in one second at one volt pressure. Following a somewhat vague description of ampere-meters and voltmeters it is claimed that "the ampere-meter is simply an extremely sensitive mili-voltmeter with a different scale." Mr. Hawkins states that "there is much more to the subject of voltmeters and ampere-meters" than he gives, but he surely should have made reference to the shunt which makes it possible for an extremely sensitive milli-voltmeter to indicate hundreds of amperes.

The discussion of the effects of current density on the character of metal deposit concludes with a table on page 175, giving the current densities required to plate different metals upon thin sheets. For thick sheets or heavy work which *can conduct heat rapidly*, it is said that the current densities given may be multiplied by from six to ten. The explanation for this, which is given on page 172, includes a statement that a dynamo bought to do work according to tables of suitable current densities, will be wholly inadequate for anything but thin sheet metal. "It is easy to see," says Mr. Hawkins, "how this mistake arose. The limit of quantity of current is at that point at which the *metal commences to discolor from the heat evolved* when the metal is deposited out of the solution; a thin plate conducts heat slowly beyond a given quantity. On the other hand, a large heavy piece contains metal enough to conduct heat away as fast as it is formed, and ten or twelve times as much current may be used profitably on such work." The author has apparently applied a too literal meaning to the designation "burned coating," which electroplaters give to a bad deposit.

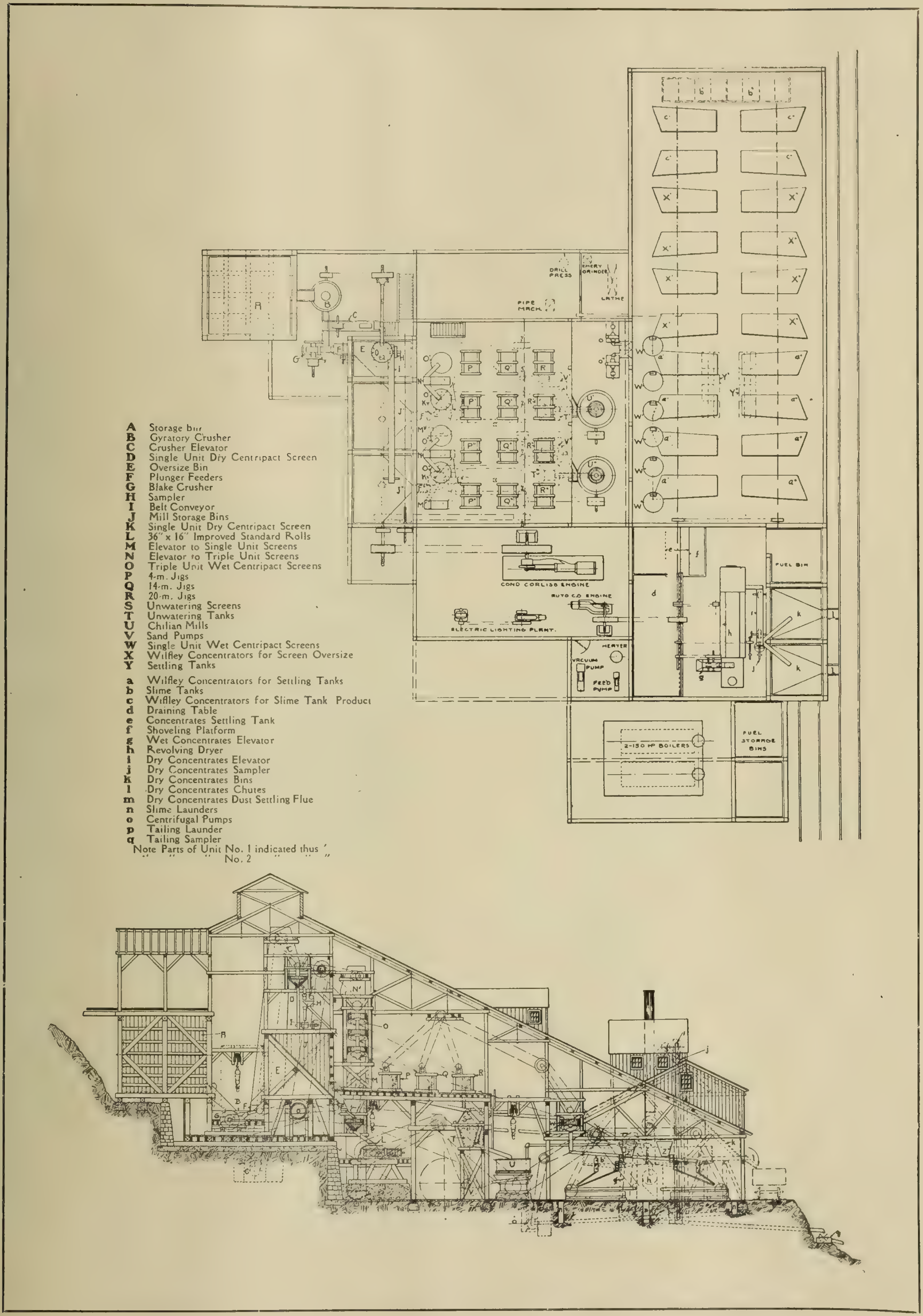
Again, on page 175 is a statement implying a contradiction of Faraday's laws, where it is claimed that " in computing the surface of work in the tank all surface which the solution touches must be counted, whether it plates or not, as current will pass through it even if it does not plate." It is difficult for the student to understand why a current density of ten amperes per square foot flowing to the interior surface of a hollow object will not deposit a coating, while it does so on the outside. Statements such as these can scarcely be considered as tributes to the intelligence of practical platers, and can only lead to confusion on the part of the apprentice.

While the author may, perhaps, have been justified in offering an empirical rather than a scientific treatise on electroplating, he should have used greater care in his treatment of scientific principles where he found it necessary to deal with them.

A Colorado Concentrating Plant.

One of the most modern and efficient concentrating plants in this country—interesting alike in the compactness of the general design and in the extensive use of a novel screening device—is the new mill of the Green Mountain Mining & Milling Co., of Silverton, Colorado. The plant was designed by the Traylor Engineering Co., of New York City.

The mill handles an ore concentrating from 3 to 5 tons into one, the ore carrying gold, silver, lead and copper. In the design of the plant the aim was to make such arrangements as to classify the ore product before and after crushing, to have the feed to each machine under perfect control, as well as being automatic and capable of being operated with the least possible number of men. The plant is a double unit, coarse and fine concentrating mill, each unit of which can be operated entirely independently of the other. Fig. 1 gives the plan and Fig. 2 an end section of the mill, while



A	Storage bin
B	Gyratory Crusher
C	Crusher Elevator
D	Single Unit Dry Centripact Screen
E	Oversize Bin
F	Plunger Feeders
G	Blake Crusher
H	Sampler
I	Belt Conveyor
J	Mill Storage Bins
K	Single Unit Dry Centripact Screen
L	36″ x 16″ Improved Standard Rolls
M	Elevator to Single Unit Screens
N	Elevator to Triple Unit Screens
O	Triple Unit Wet Centripact Screens
P	4-m. Jigs
Q	14-m. Jigs
R	20-m. Jigs
S	Unwatering Screens
T	Unwatering Tanks
U	Chilian Mills
V	Sand Pumps
W	Single Unit Wet Centripact Screens
X	Wilfley Concentrators for Screen Oversize
Y	Settling Tanks
a	Wilfley Concentrators for Settling Tanks
b	Slime Tanks
c	Wifley Concentrators for Slime Tank Product
d	Draining Table
e	Concentrates Settling Tank
f	Shoveling Platform
g	Wet Concentrates Elevator
h	Revolving Dryer
i	Dry Concentrates Elevator
j	Dry Concentrates Sampler
k	Dry Concentrates Bins
l	Dry Concentrates Chutes
m	Dry Concentrates Dust Settling Flue
n	Slime Launders
o	Centrifugal Pumps
p	Tailing Launder
q	Tailing Sampler

Note Parts of Unit No. 1 indicated thus ′
" " " No. 2 " " ″

FIGS. 1 AND 2.—PLAN AND END SECTION OF CONCENTRATING PLANT.

Fig. 3 is a flow sheet which will enable the reader to follow readily the travel of the ore from the storage bin at the head of the mill to the concentrating tables. The course of the ore through the mill is in detail as follows:

The ore is drawn from the storage bin, under control, to a No. 4 gyratory crusher. From the crusher the ore goes to an elevator and is conveyed to one single unit 1-inch mesh dry Centripact screen, the construction of which will be described below. The oversize from this screen passes down into a 50 ton storage bin, feeding back a Blake crusher through an automatic plunger feeder. From the crusher the ore is delivered to the same elevator and is returned to the same screen for further classification.

All product passing through the 1-inch meshes of the screen goes through an automatic sampler which at regular intervals takes portions of the product, thus enabling the mill management to determine at all times the assay value of the ore passing through the mill. After the sample has been taken, the remainder of the product is delivered upon a 16-inch conveyor belt located over bins, as shown in both plan and section, Fig. 1 and 2. All product in these bins, which serve as the general mill storage bins, is thus 1-inch and finer.

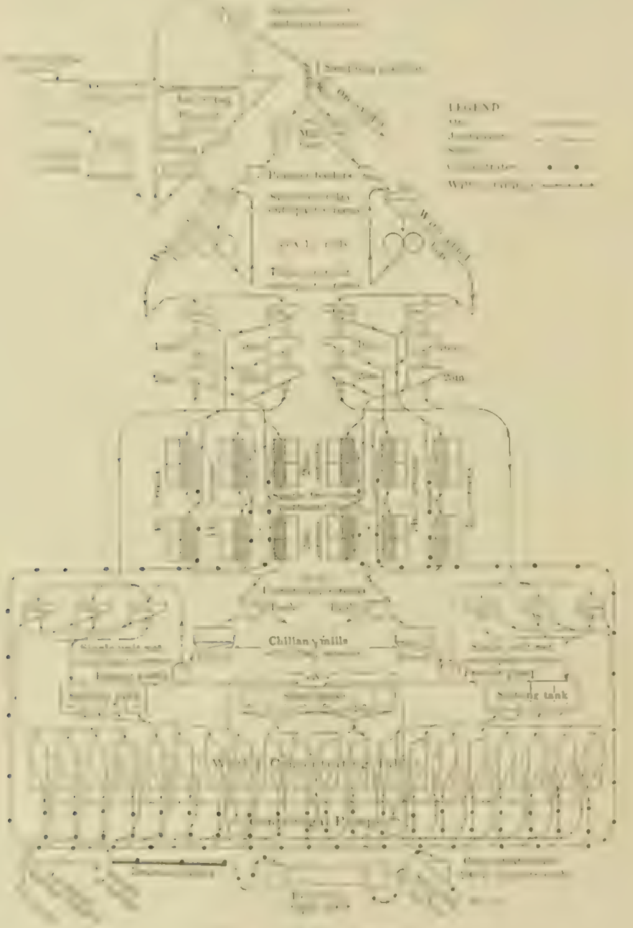


FIG. 3.—FLOW SHEET.

From this point on, the plant is in duplicate, and one unit only need be described. From the storage bins, the ore is fed through an automatic feeder to a single-unit dry Centripact screen of 3-mesh. The oversize from this screen goes by gravity to a set of heavy-duty roughing rolls, and the product fine enough to pass through is delivered directly to the boot of the general mill elevator leading to the triple-unit Centripact screens, water being meanwhile added. These screens, located above the jig floor, are in duplicate, thus insuring continuous operation.

The product passing through the rolls is delivered to the boot of a second elevator which re-elevates it back to the same single-unite Centripact screen for further classification, and so on. Thus, all product delivered to the triple-unit screens is 3-mesh or finer. The screens on the triple-unit Centripact are respectively of 6-mesh, 16-mesh and 25-mesh. The oversize from the first of these screens is delivered to two 2-compartment jigs fitted with sieves of 4-mesh. The undersize from the first screen is delivered to the second screen, the oversize from which passes to two 2-compartment jigs fitted with sieves of 14-mesh. The undersize from the second screen passes on to the third screen, the oversize from which is delivered to two 2-compartment jigs fitted with sieves of 20-mesh. The product passing through the third screen, all of it being 25-mesh and finer, is delivered to one or another of three 50-mesh, single-unit wet screens.

All jig tailings are conducted in a launder to an unwatering screen located over an unwatering tank, thus separating and controlling the surplus water and preventing the passage of this surplus water through the Chilian mill. The product passing through the unwatering screen settles to the bottom of the tank and is drawn through a draw-off gate with such water as required and enters the Chilian mill simultaneously with the oversize from the unwatering screen. The surplus of overflow water from this tank, carrying more or less value in fines, is delivered directly to a general slime tank. In the Chilian mill, the jig tailings are recrushed down to 30-mesh, the mill discharging toward the rolls to a Frenier pump by which the product is elevated to the three single-unit wet Centripact screens referred to above. Thus, the Chilian mill product is mixed with the undersize from the third screen of the triple-unit set above referred to. The three single-unit screens are equipped with brass wire cloth of 50-mesh, as already noted. The oversize, which contains no slime whatever, is delivered directly to four Wilfley concentrating tables. The undersize, 50-mesh and finer, is delivered to unwatering and settling tanks from which all of the settled product, including such slimes as may be associated with same, is delivered to an entirely separate set of tables. The overflow from these tanks is delivered to the general slime tanks.

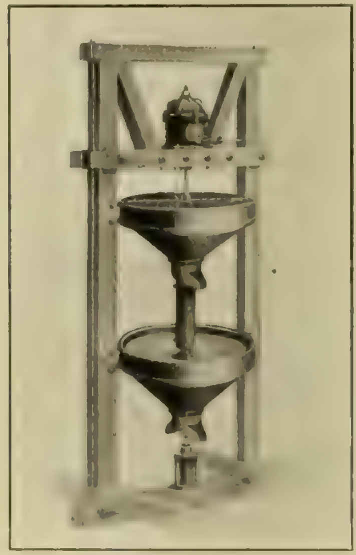


FIG. 4.—DOUBLE-UNIT CENTRIPACT SCREEN FOR DRY WORK.

On the side of each table a portable sliding launder is provided, permitting of cutting out any portion of the tailings, preferably the water crossing directly at the head of the table carrying the fine slime products. This slime product is re-elevated by centrifugal pumps to the general slime tanks where all of the surplus water, including the slimes from the general tables, are settled and drawn on to separate Wilfley slime tables for further concentration. All jig and table concentrates are drawn by gravity to drainage tables and stored for drainage before being delivered to dryer. After allowing all concentrates to drain for twelve hours, more or less, they are fed into a cylindrical dryer, and from this dryer elevated to an automatic sampler where accurate samples are taken. The concentrates are finally stored in bins whence they may be drawn automatically into railroad cars. A study of the plan, Fig. 2, will show the symmetrical arrangement of all machinery included in this plant. The separate unit shafts which permit the driving of either unit separately should also be noted, as well as the fact that all machinery is so located that there are no belts, pulleys, or shafts in the way to prevent perfect access throughout the mill.

The chief interest in details of construction rests in the extensive use of the Centripact screen, invented by Mr. S. W. Traylor, head of the Traylor Engineering Co. This screen promises to inaugurate a very important new development in screening practice, meeting every requirement of a perfect sizer. The advantages claimed for the screen are briefly as follows:

The Traylor Centripact screen screens any material, either wet or dry, up to 100-mesh. It screens wet without any added water. It screens more accurately than does even the hand screen. The meshes of the screen are kept from clogging by its normal motion—no pounding from the outside. It is easily and cheaply installed, requires little power, and runs practically without attention. The life of the screen proper is prolonged indefinitely by the method of distributing the material over the same.

The screen proper is approximately saucer-shaped and is mounted on a vertical shaft which rotates at a speed of about 30 revolutions per minute. To the shaft and screen are also given successive vertical impacts of a frequency of about 600 per minute. The particles on the screen are thus subjected to two forces—centrifugal force, tending to drive them toward the periphery of the screen, and an upward impact, tending to make them leap from its surface. From this combination of centrifugal and impact movements—which is the fundamental feature of the Centripact screen—the name is derived.

The material is fed in from above through a launder to a feed plate at the center of the screen. From there it is distributed uniformly over the screen. Since the vertical impacts on the particles are the same over the whole surface, while the centrifugal force is the greater the further the particles are away from the center, it is evident that in order to get uniform distribution over the screen the screen-surface should not be a plane. The screen curves downward from the center and the natural movement of the material by gravity thus aids here the centrifugal force in distribution. The downward curvature gradually diminishes and the screen flattens out and finally rises in such degree near its periphery as to effectually counteract the greater centrifugal force. The material travels over the screen cloth by leaps and bounds so that wear by attrition is practically nil.

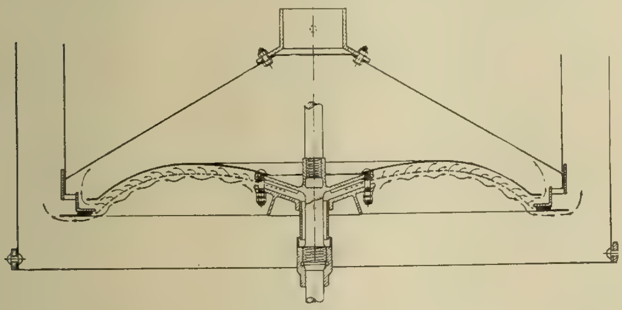


FIG. 5.—SECTION OF CENTRIPACT SCREEN UNIT FOR WET WORK.

In the construction of the driving head and the arrangement of the various parts therein, the leading idea was to make the construction as simple and strong as possible.

The Centripact screen is made by the Traylor Manufacturing & Construction Co., of Newark, N. J., the Traylor Engineering Co., of New York City, being the sole representatives. The screen is built in two distinct patterns, for wet screening and for dry screening, respectively. Fig. 4 shows a double-unit Centripact screen for dry work complete. The most noticeable difference between the wet and dry screening types is in the construction of the housings, a difference made necessary by the fact that a far greater slope is required to secure a free movement of dry ore than is required for pulp. The most essential variation in the two patterns, is that, for wet screening, a sheet steel pan, conforming in shape to the screen, is supported beneath the screen cloth about three-quarters of an inch from the same, as shown in Fig. 5. The water passing through the screen is caught in the pan and flows outward over it. Owing to the action of centrifugal force, the surface of the water assumes a form corresponding to that of the screen and pan. There is thus a layer of water immediately under the screen and, with each impact, the water is projected upward against the screen, serving to clear the meshes and at the same time thoroughly irrigate the material being screened, rendering it of the proper consistency for efficient sizing by the screen.

In cases where the pulp is of a nature to corrode a screen of iron wire, screen cloth of brass or copper wire is used. When necessary, all of the parts of the Centripact which come into contact with the pulp can be protected by a wood lining or by copper sheathing.

It will be seen that in contradistinction to the hydraulic classifier which classifies according to settling power, the Centripact screen, without adding any water to the pulp, classifies it by size and by size alone.

Wood Distillation.

The old charcoal process in which the wood was charred in heaps or in kilns and all products of distillation were allowed to go to waste, is still carried on to some little extent in remote places in this country. However, in the last thirty or forty years—especially since Dr. H. M. Pierce began his pioneer work in 1876—this industry has been completely revolutionized and the recovery of the by-products—wood alcohol, acetic acid, acetate of lime—is now of utmost importance.

Mr. F. H. Meyer, of Hannover-Hainholz, a German manufacturer of apparatus for wood distillation, represented in this country by Messrs. Frederick Bertuch & Co., of New York City, has recently issued an interesting pamphlet in which he gives a full review of the modern methods of wood distillation. He distinguishes between those processes mainly used in Europe and in this country and pays attention to the different results obtained with different kinds of wood. He emphasizes that in present practice, the whole process consists of a great number of steps, with repeated evaporations, thus necessitating a comparatively high consumption of heat, work and a great many apparatus.

Mr. Meyer then gives a thermochemical review of the whole process, stating the amount of heat in calories which become available during the different operations. The result is most interesting, since it shows that in present practice more calories are available at the end of the process than are required for the treatment of the crude wood alcohol. These calories should, of course, not be wasted, and Mr. Meyer proposes to utilize the sources of heat which become available during operation for the treatment itself.

While the details of his process are not given, the gist of it is that by utilizing the "latent heat" of the wood alcohol and tar vapors escaping from the retorts, with the aid of special apparatus inserted between retorts and condensers, he obtains directly from the charring process (*i. e.*, without a second distillation) a wood alcohol, free from tar (and turpentine), which, when neutralized with lime, and evaporated, gives directly a gray acetate of lime of great purity. The result is a saving of heat and apparatus, as well as a reduction of wages. If the process works as well on a large scale as is hoped by the author, it will be another example of a reduction of cost of operation, based on an analysis of the operation on the basis of thermochemical calculation. The author recomurges the more extended use of the large retorts generally employed in this country, though he apparently favors change in construction and details.

The author states that since the publication of this pamphlet, a large-sized plant has been erected and operated in Germany, the results obtained fully justifying the efficiency of the process.

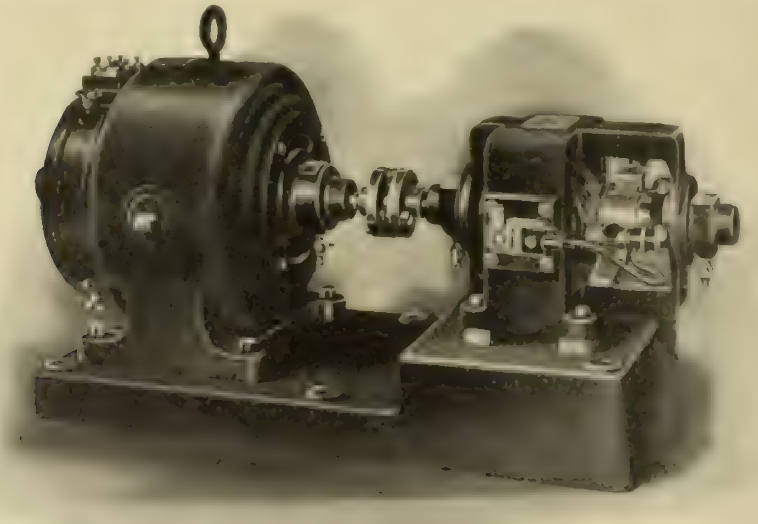
Motor-Generator Sets in Electroplating and Electrochemical Works.

By C. G. Backus.

There has been many great changes in transmission of power during the last few years. These changes have been an improvement on the old order of things, and most of these improvements have been called forth by a continual demand for increased efficiency, better regulation and economy. To attain one or all these results, all lines of manufacturers are now building their machinery to be self-contained and independent.

The factories of a generation ago, with their endless lines of main shafts, pulleys and flying belts are being replaced by modern machinery direct-connected to individual motors, which receive their power from a central dynamo or power plant by means of a system of invisible wires, instead of the cumbersome and unsightly, and often more dangerous and expensive shafting and belt system.

The various electroplating, electrotyping and electrochemical processes all require a low voltage generator, to obtain the best results with a minimum expense of power, and this style of dynamo presents such difficulties that it has only been built successfully by a few manufacturers. The Zucker and Levett & Loeb Co., of New York, one of the oldest manufacturers of supplies for electroplating and polishing material in the country has made, for years, the low-voltage dynamo the subject of careful study and development. At various times in the last thirty years they have brought out new types of generators of this class, each successive type being an improvement on its predecessor. They have long recognized the tendency of the age, and have been building for a number of years generators, self-contained and direct-connected to a motor, in sizes ranging from 25 amperes to 6000 amperes. A direct-connected motor-generator set of this type is the only practical direct-current transformer for low voltage work.



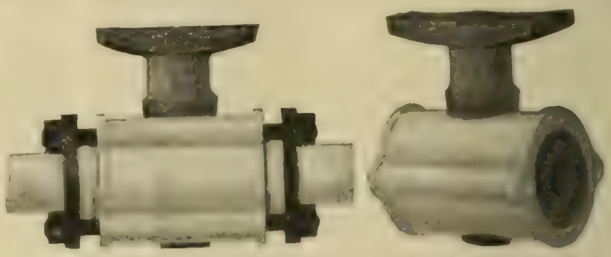
MOTOR-GENERATOR SET.

With the belt-driven generator it was not possible to vary the voltage, or rather to cut it down, without cutting down the field strength, or by throwing resistance into the main circuit of the dynamo. The first way destroys or impairs the balance between field and armature, thus impairing the efficiency of the machine. The second plan was a waste of energy, the power generated being absorbed and given off as heat. With the direct-connected motor-driven generator it is different. The loss of power from the tight belts is eliminated. With a speed controller for the motor, the voltage of the generator can be varied fully 40 per cent, without calling into use the two older methods above mentioned, *i. e.*, changing the field strength or inverting resistance.

The ideal regulation is obtained by having the fields of the generator separately excited from a high-voltage circuit, as by this method, in addition to the regulation above referred to, a certain further variation can be obtained by weakening or strengthening the field by means of a field rheostat operating on a line not controlled by the speed of the generator itself; and for generators of 500 amperes or more the generator can be arranged to use the three-wire system, by which two or three different voltages can be obtained from the machine at the same time.

Armored Stoneware Cocks and Spigots

It has often been emphasized by the most successful chemical and electrochemical engineers that they experienced their greatest troubles in the solution of the numerous small problems in details of mechanical construction. A process may work extremely well in its experimental stage, but when its operation is started on a large scale, innumerable troubles are experienced from such little things as leaking and wearing out of cocks and spigots. The accompanying illustration shows an interesting and novel construction of armored stoneware cocks and spigots, made by the firm of Karl Ruppel in Hoechst-am-Main, Germany, in conjunction with the German Stoneware Factory, of Friedrichsfeld, Baden, this latter concern being represented in this country by F. Bertuch & Co., New York.



ARMORED STONEWARE COCK.

Lead cocks are often liable to wear out quickly, while cocks of pure earthenware are difficult to construct for a high pressure. The construction shown in the illustration combines earthenware and lead construction, and is of extreme compactness and of great strength. It may be used with very high pressures instead of expensive hard-lead valves. The cock itself is made of stoneware armored with lead, the flanges of the connecting pipe being joined to the cock, by means of iron screws. By a special process, a homogeneous union is attained between the earthenware and the lead. The earthenware is important on account of its resistivity against chemical influences, while the lead with the screw construction yields the necessary strength. The installation is most simple; by working the screws on one side, both ends are simultaneously and uniformly tightened. The right-hand illustration shows the stone lining inside the lead armor, without the iron screws put in; while the left-hand illustration shows the cock complete, attached to lead pipe.

Industrial Notes.

Messrs. Hammacher, Delius & Co., of New York, are the American representatives of the Swedish company which makes, at Gysinge, steels of crucible steel quality in the electric induction furnace of Kjellin, which was repeatedly described in our columns.

We have received from Messrs. Eimer and Amend, of New York, a nicely illustrated pamphlet on electric laboratory furnaces and heating appliances in general, in which the new granulated resistance material Kryptol is used. This material is filled into the heating apparatus or spread out on clay or enamel plates; depending on the thickness of the layer of Kryptol, and on the current employed, the temperature can be regulated as desired, so that it is possible to produce at will different temperatures at various points of the heating apparatus. The great ease of handling this granular material—which will not burn through like a metallic heating wire, and which is not attacked by acid vapors—renders it applicable to a great many purposes in the laboratory without any difficulty. Among the Kryptol heating apparatus mentioned in the pamphlet are baths for heating evaporating dishes, flasks, beakers, etc.; hot plates, drying ovens, muffle furnaces, crucible furnaces and tube furnaces.

The National Battery Co., of Buffalo, N. Y., has issued a fully-illustrated pamphlet on automobile batteries. The plates are of the pasted type, the positive grid being of the "staggered" form, while the negative grid is known "as the much litigated and successfully defended (National) patent form of corrugated, punched and clinched type of grid, which, having the material on two sides of the plate, welded together, firmly holds the active material to the grid and gives a large conducting area of metallic lead, the great advantage being light weight and stiffness, combined with great capacity per pound."

The Buffalo Dental Mfg. Co., of Buffalo, N. Y., has issued a fully illustrated catalogue of 56 pages on Laboratory Appliances for colleges, schools, chemists, assayers, manufacturing jewelers, etc. (Catalogue B, list No. 33.) The first part of the catalogue deals with various types of laboratory furnaces: gas furnaces, requiring blast; gasoline gas furnaces; kerosene blast furnaces; gas furnaces, without blast, etc. Then follow descriptions of various types of blow pipes, burners, and laboratory appliances in general.

The bauxite brick, mentioned on page 43, of our January issue, are made by the Laclede Fire Brick Mfg. Co., of St. Louis. This company has just installed in its plant a complete laboratory, which is in charge of Mr. A. J. Aubrey. The company, in manufacturing fire brick, cupola blocks, tiles, sewer and culvert pipe, etc., analyzes all the raw materials entering into the manufacture. Knowing the proportion of all the different elements in the materials used, they are in position to guarantee the serviceability of its products.

Crocker-Wheeler Annual Convention.—The annual convention of officers and branch managers of Crocker-Wheeler Co., took place at the main office and works, Ampere, N. J. January 26, 27 and 28. Managers and representatives were in attendance from all parts of the country, and were unanimous in predicting a prosperous season during the coming year in the field of alternating-current generators and direct-current generators and motors. On the evening of the 27th a banquet was held at the Cafe Martin in New York, at which the president of the company, Dr. Schuyler Skaats Wheeler, presided.

New Smelter in Mexico.—It is reported that a large new smelter at the important mining camp of the Velardeña Mining and Smelting Co. (now regarded as a branch of the American Smelting and Refining Company), located between the Mexican International and Mexican Central roads in the State of Durango, will be speedily built. Its capacity will be 1000 tons, and perhaps more; it will be equipped in the most modern style for copper matte, as well as silver-lead smelting. The old smelter of the company will be abandoned when the new one is completed. It will be a custom smelter.

Steam Turbine Plant in Klondike.—The Westinghouse companies have just entered an order for the equipment of a power house for the electrical operation of gold-dredging boats on the Alaskan rivers. The plan is an entirely new one and involves many interesting features. There will be installed a 400-kw. turbo-generator in the power house, to be driven by a 600-hp. Westinghouse-Parsons steam turbine. The dredge boats are being built by the Marion Steam Shovel Company, of Marion, Ohio. On these boats will be installed induction motors, aggregating a total of about 500 hp., and varying in size from 7½ to 100 hp. The fact that a plant of this nature is to be installed in such a distant country, far from the manufactory and possible repairs, shows the confidence engineers place in this type of unit. The power house will be located at Dawson City, and the dredges will operate on the Yukon River and its tributaries. Lines for transmitting power will be strung from the station to the boats, wherever they may be working.

Personal.

Dr. Rudolph Gahl, formerly of the Electric Storage Battery Company, of Philadelphia, has associated himself with the Denver Laboratories, assayers, analysts and electrochemists at Denver, Colo. The members of the firm are Messrs. R. Gahl, H. C. Parmelee and C. H. Bryan. Dr. Gahl, who is well known to the electrochemical fraternity through excellent and extended research work in storage battery engineering, will devote himself especially to consulting work in connection with storage battery and electrochemical engineering in general, since electrochemical methods are now attracting considerable attention in the metallurgical works of Colorado.

Mr. Andrew Carnegie has taken up literary work again, and is writing a life of James Watt, of whose work he has long made an extended and enthusiastic study. The book is nearly ready.

Dr. George W. Maynard, who recently returned from Arizona to New York, is now leaving again to examine mining properties in Durango, Mexico.

Mr. Charles W. Cross, formerly of the Roberts & Abbott Co., of Cleveland, Ohio, and later electrical engineer for the Eastern Ohio Traction Co., has entered the employ of the Crocker-Wheeler Co., of Ampere, N. J., and is attached to their Cleveland office.

Digest of U. S. Patents.

Compiled by Byrnes & Townsend, Patent Lawyers, National Union Building, Washington, D. C.

Molten Electrolytes, Miscellaneous.

(Concluded from page 88.)

689,286, December 17, 1901, Guillaume De Chalmot, Leaksville, N. C.

Natural phosphates are mixed with a small proportion of silica sand and fused in an electric arc furnace. A certain proportion of anhydrous phosphoric acid is volatilized and absorbed by moist lime or phosphate rock in trays. The phosphate in the furnace is permitted to overflow as fast as melted, in order to avoid too great reduction of its phosphoric acid content. The molten product drops onto water-cooled iron drums together with sand to prevent adhesion to the iron, thence into water, and is crushed and utilized as a fertilizer. The material in the trays, as well as the product from the furnace, contains a large proportion of available phosphoric acid. The operation is not dependent on any electrolytic effect, and alternating currents are preferred in order to avoid reduction of the phosphate.

Electric Smelting and Reduction Processes

319,795. June 9, 1885. Eugene H. Cowles and Alfred H. Cowles, Cleveland, Ohio.

Smelts ores, as of zinc, aluminium, silicon, magnesium and boron, by the heat generated by the passage of electricity through a body of granular material of high resistance, such as carbon or silicon. The ore, for example, aluminium hydroxide, alum, sodium chloride, calcium oxide or strontium sulphate, is usually mixed with the granular resistance material, which is contained in an air-tight chamber or non-oxidizing atmosphere to prevent oxidation. The carbon reduces the ore, but is otherwise unconsumed. Two forms of zinc furnaces are illustrated. The first is a horizontal tube of silica, embedded in powdered charcoal or mineral wool. The rear end of the tube is closed by a carbon-plate anode; the front end by a graphite-crucible cathode which opens outwardly, but is closed by a clay plug. One and one-half parts of zinc ore are mixed with one part of carbon, and the tube is filled nearly full of the mixture. When the current is passed, the zinc distills and passes through an opening in the bottom of the crucible-cathode, wherein it is condensed. The waste gases escape through a vent pipe. The second furnace is a horizontal chamber having a hearth which slopes downward from both ends to a tap-hole at the middle. The ends are closed by carbon-plate electrodes, one of which has an opening receiving the neck of a condenser. A series of hoppers open into the chamber at the top, enabling the resistance of the charge to be regulated by the amount of material fed in. Reduced non-volatile metals remain in the furnace, filling the interstices between the carbon. For the production of pure metals, pure carbon must be used.

324,658. August 18, 1885. E. H. and A. H. Cowles, Cleveland, Ohio.

Produces alloys, aluminium, silicon and boron bronzes, etc., by adding to the resistance charge described in the previous patent, an alloying metal or ore thereof. For the production of aluminium bronze, corundum, cryolite, or clay is pulverized and mixed with broken carbon and a pulverized copper ore, *e. g.*, the oxide. The mixture is placed in a horizontal column between carbon electrodes. Copper may be substituted for its ore, rods of copper being transversely inserted into the charge, or granulated copper being mixed with it. May also produce alloys, carbides, silicides, borides and saturate aluminium with copper and iron or silicon and manganese, by the use of a Siemens electric-arc furnace. For this purpose, the alloying metal or its ore are either mixed with the charge of the arc furnace or crucible, or an electrode of the alloying metal, *e. g.*, copper, tin, nickel or iron is used, the electrode being gradually melted and the alloying metal incorporated with the reduced metal.

324,659. August 18, 1885. E. H. Cowles, C. F. Mabery and Alfred H. Cowles, Cleveland, Ohio.

Produces pure aluminium and other metals and metalloids, such as silicon and boron, by reducing a mixture of the ore, granular carbon and an alloying metal, and then separating the alloying metal, leaving the residue aluminium in the form of an amorphous powder which is melted into an ingot. The smelting is performed by passing a current through the charge, serving as a resistance conductor, as in 319,795. The alloying metal, *e. g.*, tin, silver copper, manganese zinc or copper prevents absorption of carbon by the aluminium. The tin, silver or copper may be removed from the alloy by amalgamation with mercury, which does not dissolve the aluminium, the action being quickened by the use of sodium amalgam, acid, electricity or heat. Manganese, zinc or copper may be removed by lixiviation. The alloy may be broken into small pieces and leached with dilute nitric acid, and the solution may be quickened by other acids, heat or electricity.

335,058. January 26, 1886. A. H. Cowles, Cleveland, Ohio.

Maintains a uniform resistance in Cowles' incandescent furnaces by employing carbon-rod electrodes which are adjustable in openings at the ends of the furnace, and are gradually drawn apart as the specific resistance of the charge decreases during the run. An electrically-controlled mechanism may be used for adjusting the electrodes, that shown being a magnet in a shunt circuit, having a vibrating armature and clock work which operates a pinion engaging a rack attached to the electrode. The connectors consist of copper boxes filled with copper shot surrounding the electrodes. The furnace walls are of firebrick lined with pulverized charcoal. The charge normally consists of pulverized carbon and ore. In starting, the electrodes are placed near together, and are gradually drawn apart as the resistance falls. The charge is covered by granular charcoal, and the furnace is closed by a slab of fireclay with vent holes. The reduced metal is found in the interstices between the particles of carbon and on the charcoal floor of the furnace.

335,499. February 2, 1886. C. S. Bradley, Yonkers, and F. B. Crocker, New York.

Reduces sodium or potassium by heating a charge of the carbonate, charcoal and chalk, in a hollow cylinder of wrought iron, heated by passing a current through it. The cylinder is set in brickwork, at a slight inclination from the horizontal, spaces in the brickwork around the cylinder being filled with asbestos or mineral wool. The terminals are rings of copper surrounding the ends of the iron cylinder. The cylinder is closed at each end with covers having inspection holes closed by screw plugs. The door at the lower end has a pipe which delivers the sodium or potassium vapor to a condenser. Fresh material is charged into the cylinder at the upper end, and the residue is taken out from the lower end. For the reduction of zinc, a mixture of zinc oxide and carbon is heated in a resistance-cylinder of carbon, preferably the mixture of graphite and clay used for crucibles. Instead of a cylinder, the reduction chamber may be built up of plates of iron or graphite-clay bricks. For the production of aluminium chloride, a mixture of alumina and carbon made up into balls is charged into a vertical resistance-cylinder of graphite and clay. The cylinder is electrically heated, and chlorine is introduced at the bottom, the chloride distilling and being delivered through a pipe at the upper end. Instead of using a resistance-cylinder, a conductor may be placed in the bottom of the chamber only, or in the sides, or in the top, in which case the materials are heated by radiation from above. The resistance may be heated by a direct or alternating current.

403,752. May 21, 1889. John C. Hobbs, Lockport, New York.

Lines furnaces of the Cowles type with sawdust, surrounding the resistance charge. The heat from the charge carbonizes the wood. The resistance of the carbon lining is ordinarily increased, to prevent shunting of the current, by soaking it in lime water.

417,943. December 24, 1889. James B. Readman, Edinburgh. Scotland.

Reduces phosphorus by electrically heating the charge, acting as a resistance conductor. The charge is made by evaporating a solution of phosphoric acid or acid calcium phosphate, incorporating the residue with carbon and desiccating; or bone-ash, or alkaline, earthy, aluminious or metallic phosphates, or metallic phosphides may be mixed with carbon. Silica or silicious aluminates, basic or acid fluxes may be added. The charge is placed in a rectangular furnace closed at the top by a fire-clay slab. At one end of the chamber is an inclined passage for the introduction of materials, having a hopper with two valves. At the opposite side is an outlet for leading the phosphorus vapor to a condenser. Downwardly inclined carbon electrodes pass through the other two sides, and the current passing between them and through the charge heats it to incandescence. Each electrode consists of a set of carbon rods held in a cast-iron head which slides in a two-part casing.

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Boston Meeting American Electrochemical Society.

The programme published on another page of this issue for the annual meeting of the American Electrochemical Society, to be held during the last week of this month in Boston, shows that the local Committee, together with the Secretary of the Society have done energetic and excellent work toward making this meeting an especially memorable one. The list of papers, as far as it is published, is well balanced between scientific research and engineering application. The same fundamental tendency towards harmony between the art and the science is happily indicated by the cordial way in which both Harvard University and the Massachusetts Institute of Technology will extend alternately their hospitality to the Society. We are informed that a comparatively large number of the members of the society have already declared their intention to attend the meeting.

◆

Phases of Development in a New Process.

In the constant state of flux of metallurgy, numerous attempts to improve the present practice in details, or to develop some radically new process, are always being made. Growth is a necessary concomitant to life. A concern that does not grow is liable to decay. Consequently, an attitude of always trying to evolve something better from the old is a healthy one. Eternal experimenting is the price of progress. The course of development of a new process is susceptible to a searching out of the essential qualities for success.

Every successful process must be first of all scientifically possible. It is impossible to make water run up hill. It is likewise impossible to make a reaction proceed when the free energy of the counter reaction is against it. Next, the apparatus must be mechanically satisfactory. A furnace designed without allowing for the expansion of the firebricks, or a vat that is slowly corroded by the liquor, will cause the practical failure of the most beautiful set of reactions. Finally, and most important of all, the process should work on such raw materials and should produce such a product that it can be sold at a profit under market conditions. The ability to return profit to capital is the last test of any process. Very few of countless new ideas ever are commercially successful. The reason for this lies in the fact that all three factors must be present. Suppose all but one out of ten new proposals fail for scientific weakness. And suppose that all but one of the survivors fail for engineering weakness. Finally, suppose that all but one of a hundred fail for commercial reasons. On these assumptions, out of ten thousand new ideas only one has real merit. Many of these that possess real merit are never commercially successful because of poor handling, like "the seeds that fall by the wayside." The small number of [illegible] new projects is always characteristic of progress. As Michael Faraday says: "The world little knows how many of the theories of the scientific investigator have been crushed in silence by his own severe criticism; that in the most successful instances not a tenth of the hopes, wishes, the preliminary conclusions have been realized." Of the million of fish eggs of a shad, but few reach maturity.

Such being the case concerning a new process, a course of action in developing it should be followed that corresponds to these conditions, and in the order named. The first test should be to see if the process is scientifically all right. This can be done on a small scale, and the expense of such a test is slight. It should be next tested to see if the conditions found right in small experiments can be held on a larger scale. The final test is how efficiently the process works on a very large scale, and thus it can be estimated whether it will be a success. Such being the plan of campaign, it should logically and methodically be carried out. Every step should be the result of frank discussion by those interested. Now, the new process will be found to advance by the following stages, something according to the general scheme outlined above. First, the laboratory stage: Here it must be shown that it is scientifically possible to produce the results on a small scale. Second, the practical stage: This should be on a fairly large scale, say, 300 pounds of the material, and show that it is possible to hold the right conditions, found in the laboratory stage, for some time. Third, the engineering stage: This should operate on one or two tons of the material continuously for a long period. Many defects must be shown up and remedied. The apparatus must be proved durable and efficient, and the recovery must be high. The control over the purity of product must be shown. In brief, this third stage must give evidence that the passage to the fourth one is certain. This fourth stage is the commercial stage: Actual production of finished material for the market at a greater profit than under the old process. This is the goal toward which all the energies had been turned—commercial success.

From each stage it must be demonstrated that progress to next stage is as sure as things can be in this world of uncertainty. By such a careful, well-thought-out, conservative course, can the risk incident to starting a new enterprise be reduced to a minimum. In war, it is said that one ton of lead is necessary to kill each soldier. So many shots miss the mark. The same is true of industrial work. Well-planned schemes, carried out with reason and courage, will win in the end, barring chance and the happening of the unexpected. But in metallurgy, as in the whole world, nothing succeeds like success.

◆

Thermochemical Data for Metallurgists.

Numerous letters which have reached this journal, make it evident that the serial on metallurgical calculations from the pen of Prof. J. W. Richards, which was commenced in our last issue, is being received very cordially and even with enthusiasm by progressive practical metallurgists. The second installment, which is given in the present issue, is essentially a summary of those thermochemical data which should be of most importance for metallurgists. The thermochemical balance sheet of a metallurgical process should have the same importance for the engineer in charge as the financial balance sheet has for the business manager. It shows the efficiency of the whole process and its various parts; it makes it clear where it would be wise to try and make improvements, and where the highest possible efficiency has already been attained and further attempts of improvements must be failures. Prof. Richards' careful compilation and arrangement of those most important thermochemical data which have been determined, should, therefore, be much appreciated, since very few metallurgists have at their disposal the voluminous works of Berthelot and Thomsen. But, alas! how few are the data in some fields where the metallurgist needs them so badly—for instance, data on silicates and alloys. True, every scientific research, if made in the right spirit, will yield fruitful results. Nevertheless, it would seem that much time, money and effort is now spent on so many researches, the fruitfulness of which is not quite evident (as in organic chemistry), while no one knows the heat of formation of blast furnace slag, or the specific heat of iron ore, Fe_2O_3, to the temperature of the blast furnace, or the heat of formation of the most important alloys. Funds, granted by the Carnegie Institution for the determination of these badly needed data, would be well spent.

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Large Scale Production and American Metallurgical Practice.

If there is any difference at all between American and European metallurgy, such difference lies in the magnitude of the operations in America. In what is familiarly termed "the steel business," by those frank, energetic, breezy individuals "the steel men," we have seen for the past one hundred years a gradual increase in size of the furnace. The first small anthracite stacks of the Lehigh Valley and the wee little charcoal cupolas of the revolutionary time would not make each as much iron in a month as one of the monster 700-ton blast furnaces of the Carnegie Steel Co. or the Lackawanna Steel Co. do in a day. In lead smelting, there has been a constant increase to 200 tons capacity. Here there has been a halt, for lead smelting is a delicate operation, and too large units are unwieldy. But the Pueblo furnaces are large in comparison to the German furnace. Twenty-five years ago the usual capacity of the copper cupola was 50 tons a day, and the furnace was tapped intermittently. Mr. John L. Thompson, then superintendent of the Orford Copper Co., built the brick "raschette" furnaces of capacity of one hundred and fifty tons with the continuous siphon spout. This furnace was the subject of much discussion, and paved the way for the modern large water-jacketed blast furnace of 500 tons, such as those used at Ducktown and at Granby. To-day these are surpassed by the long furnaces of the Anaconda Copper Co. of 1200 tons capacity. In the reverberatory refining furnace in 1890 15 tons was a large capacity. The Baltimore Copper Co. soon increased this to 35 tons. Later, the Orford Copper Co. built its 50-ton furnaces. About four or five years ago the

Guggenheims built, at Perth Amboy, furnaces of 100-tons capacity, and later of even 150-tons capacity. These furnaces were built on a new design, and such a size as was only rendered possible by the use of the Walker casting machines.

This brief recital of individual achievements in certain lines is but typical of the general expansion of the metallurgical industry in the United States. The causes for this large development can be found in the peculiar conditions of American industry. In the first place there has always been an abundance of raw material. It is obvious that to handle large amounts of raw material, large units are necessary. Large reserves of raw material justify the erection of considerable machinery. In the next place unskilled labor is high, and it is inefficient. Skilled labor is still higher, but it is efficient. Consider the case of the large Anaconda copper furnace making 700 tons of slag a day. Careful work will keep this slag clean below 0.50 per cent Cu. Poor work can make a foul slag as high as 1.00 per cent Cu. This difference, 0.50 per cent Cu, or 10 pounds of copper to the ton worth, say, 10 cents a pound "in the matte," will amount to a dollar a ton or $700 total in a day. As such a furnace will employ eighteen to twenty men a day, at a total labor charge of not over $70, it is plain the price paid labor is low, provided it is efficient. Poor furnace work would be dear at any price, however low. Thus, the high price of American labor forces large installations, and these installations give the high-priced workman a chance to handle many tons of stuff efficiently, and so to work to advantage. The result of it all is that labor charge per ton is usually smaller in America than in Europe.

While labor conditions and cheap raw material have been the main causes, there are other contributing factors. One of these is that thermal efficiency is increased with increased size, just as is the case with electric furnaces. Thus, amount of fuel per ton of material treated lessens with increase of size. For instance, in the old copper-refining furnace the coal used was 600 pounds to the ton of copper. This was reduced successively to 400-300 down to 200 pounds in the 50-ton furnace, and, finally, in the case of the 150-ton furnace, to as low as 100-125 pounds. Although coal is usually cheap in this country, yet distance from the mine makes it often expensive and the cost, whatever it is, is cut down greatly. Again, one large furnace is less in capital cost than ten small furnaces each of one-tenth the capacity. Thus, in a large scale plant less money is tied up in furnaces, and more capital can be expended for auxiliary machinery. All these different causes have interacted and changed the complexion of the whole situation. They have also acted quickly, because the temperament of the American is conducive to change, especially if the change is one from smallness to bigness and cost of change will be soon paid for by cheaper work. The spirit of the American pioneers is seen to-day in American business enterprise. But we believe that in many branches of metallurgy the limit in size has been about reached. Increase will be made, but these will be in special lines. Further progress will be made in bringing all the plants of the country to more or less standard size. The real field of the future lies in the refinements of the apparatus, in the manipulation of the process, and in the development of by-products. In this Germany has been in the lead. But in Europe they are building large American blast furnaces. So America will develop the scientific side of metallurgy. Such a rivalry has more reason for the expenditure of productive capital than has the emulation of each other in building expensive destructive battleships.

◆

Electro-Cyanide Process.

A patent recently granted to Mr. E. L. Oliver and abstracted in the present issue, is interesting in various respects, although nothing is known concerning the operation of the process on a commercial scale. While in all commercially successful processes using electrolytic precipitation of gold from cyanide solutions, the solution of the gold is carried on in leaching tanks and the solution is afterwards withdrawn and the gold is precipitated from it in other tanks, the endeavor of numerous inventors during recent years has been to carry on both solution and precipitation simultaneously in the same tanks. The process of Mr. Oliver belongs to this class. While the advantages of such a process would be evident, yet, as a matter of fact, none of the processes of this kind has so far worked really well on a large scale, although several of them appeared quite promising on a laboratory scale. Dr. W. H. Walker, in discussing this subject two years ago in an American Electrochemical Society paper, pointed out what may have been some reasons of such failures. The fact that tellurides of gold are insoluble in potassium cyanide, and so far as electrolytic methods are concerned can be reduced only by direct cathodic reduction, explains some of the earlier failures, but would also seem to show the way towards improvements. If cathodic reduction of the ore particles is to be effected, it is necessary to bring every minute particle of ore in effective contact with the cathode.

While nothing is said of cathodic reduction in the patent under discussion, it would seem that Mr. Oliver has been working along these lines of thought. As his cell is constructed, the ore particles are carried along the amalgamated cathode plates, and will make effective contact with the same, and the inventor also claims to have escaped what would be a very great drawback to his scheme—namely, the scouring of the amalgamated plates by the ore particles passing along their surface. The method by which circulation is effected is also interesting, although not new; it depends on the fact that if in two communicating tubes containing the same liquid, a gas is supplied to the liquid in one tube, the latter will become lighter and will rise and may be caused to overflow into the other tube. This continuous circulation may be effected, and, by varying the gas supply, the circulation can be regulated. This same principle of agitation is successfully used in another electrochemical apparatus where thorough stirring is of decisive importnace; namely, in the hypochlorite cell of Haas and Oettel; in the latter case no gas or air is supplied from an outside source, but the hydrogen which is developed at the cathode is directly made use of.

American Electrochemical Society.

The [illegible] meeting of the American Electrochemical Society will be held in Boston on April 25 to 27. Professor W. H. Walker, of the Massachusetts Institute of Technology is chairman of the local committee. The hotel headquarters will be at the Hotel Lenox, Boyleston & Exeter Streets, Boston. The programme is as follows:

TUESDAY.

9.30 A. M. Session of the Society in Room 6 Lowell Building, Massachusetts Institute of Technology.

1 P. M.* Reception by President Pritchett at the Technology Club, followed by lunch tendered by the Institute of Technology.

2 P. M. Excursion to the General Electric Company's plant at Lynn.

8 P. M. Presidential Address at 6 Lowell Building, followed by a smoker at Hotel Lenox.

WEDNESDAY.

9.30 A. M. Session of the Society at Harvard University, Peirce Hall.

1 P. M. Reception by President Eliot at Harvard Union, followed by lunch tendered by Harvard University.

2 P. M. A visit to Harvard Buildings, or excursion to the New England Gas & Coke Co., at Everett.

7.30 P. M. Banquet for members and ladies at Hotel Lenox.

THURSDAY.

9.30 A. M. Session of the Society at 6 Lowell Building.

1 P. M. Lunch at the Technology Club.

2 P. M. Afternoon session of the Society at 6 Lowell Building, or inspection of Technology buildings.

The following programme has been arranged for the ladies:

TUESDAY.

1 P. M. Lunch at the Technology Club.

2 P. M. Excursion to Wellesley College.

8 P. M. Presidential address.

WEDNESDAY.

1 P. M. Lunch at Harvard Union.

2 P. M. Inspection of Harvard Museums, including glass flowers.

7.30 P. M. Banquet at Hotel Lenox.

THURSDAY.

1 P. M. Lunch at the Technology Club.

The list of papers, as far as announced, is as follows:

A. B. Albro, "The Microstructure of Silicon and Silicon Alloys."

W. D. Bancroft, "The Rotating Diaphragm."

Lucien Ira Blake, "Electrostatic Method for Separating and Concentrating Ores."

C. F. Burgess, "Notes on the Economic Temperature of Copper Refining Solutions."

H. R. Carveth, "Electrolytic Potential of Chromium."

Ernest Fahrig, "Treating Low Grade Ore and Tailings by Electrolysis."

W. S. Franklin, "Reversible and Irreversible Polarization."

H. M. Goodwin, "On Billitzer's Method of Determining Absolute Potentials."

C. Hambuechen, "An Optical Method of Observing Diffusion in an Electrolyte."

William S. Horry, "An Electric Switch."

G. A. Hulett, "A Low Voltage Standard Cell."

L. Kahlenberg, "Specific Inductive Capacities."

A. Lodyguine, "Some Experiments on the Reduction of Titaniferous Ores Into Steel and Titanate of Iron."

A. Lodyguine, "Experiments on the Reduction of Different Oxides of Lead by Electric Current."

A. Lodyguine, "Experiments on the Electrodeposition of Metals on Aluminum."

H. E. Patten, "On the Heat of Solution of Aluminium Bromide in Ethyl Bromide."

A. Reuterdahl, "The Interdependence of Atomic Weights and Electrochemical Equivalents."

J. W. Richards, "Conduction in Electrolytes."

A. J. Rossi, "On the Utilization of Blast Furnace Gases in Connection with Electric Smelting."

R. C. Snowdon, "Electrodeposition of Silver."

R. C. Snowdon, "Adherence of Nickel to Nickel."

C. P. Townsend, "A Diaphragm Cell for Electrolysis of Brine."

E. Weintraub, "Mercury Arc."

W. R. Whitney, "Electric Arcs."

W. R. Whitney, "Colloids."

Notes.

American Electrochemical Society.—At the March meeting of the board of directors of the American Electrochemical Society, the following gentlemen were elected members: T. Herbert Arnold, State College, Pa.; L. O. Kuhn, State College, Pa.; James C. Armor, Pittsburg, Pa.; David B. Carse, New York City; W. S. Franklin, Bethlehem, Pa.; Max LeBlanc, Karlsruhe, Germany; Bunjiro Masiyima, Kokura, Buzen, Japan. At the April meeting of the board of directors, the names of the following gentlemen will come up for election: Daniel McMaster, Rumford Falls, Me.; Arthur D. Little, Boston, Mass.; Marten Hedlund, Haustahammar, Sweden.

Lehigh University.—The Register of Lehigh University, South Bethlehem, Pa., just issued, copies of which may be had on application, shows the attendance of 630 students from twenty-four States and eight foreign countries, the largest in the history of the institution. Seven students are taking the course in electrometallurgy. There are 56 members in the teaching staff. Thirteen four-year courses of instruction are offered at the university: The classical course, the Latin-scientific course, the courses in civil, mechanical, marine, metallurgical, mining, electrical, and chemical engineering, analytical chemistry, geology, physics, and electrometallurgy. A list of graduates of the university, with their present occupations, 1399 in number during the thirty-nine years of its existence, indicates that this institution is exerting a marked influence on the industrial development of the United States and of foreign countries.

Rensselaer Polytechnic Institute.—This institute was subjected to some inconvenience during the past year on account of the burning of the Main building in the summer of 1904. All the laboratories are in complete working order, however, and have been since last October. By the end of this year a first-class modern building for recitation and drafting rooms will be ready. Mr. Andrew Carnegie has recently given $125,000 for this building. The trustees have just purchased a beautiful piece of property adjacent to that now owned by the institute, for which they have paid $125,000. The Carnegie Building will be placed upon this. Although the existing chemical laboratory is finely equipped, it is becoming too small for the large number of students. This number is the largest in the history of the school. For this reason Mr. J. J. Albright, of the class of '68, has given $50,000 towards the erection of a new chemical laboratory, and one to cost $100,000 or more will be built during the next year.

Franklin Institute.—On March 15 Mr. Byron E. Eldred lectured before the Franklin Institute on his process of regulating high temperatures. This process was described in our December issue, 1904 (Vol. II, page 495).

Misprint.—Line 14 of the second column of page 90 of our last issue should read, "low in silver and lead," instead of low in silicon and lead.

Electrometallurgy of Iron and Steel at the New York Elec-

trical Society.—At the meeting of the New York Electrical Society, held March 15, the general subject of discussion was electrometallurgy of iron and steel. President Sprague referred in his opening remarks to some experiments made by him ten years ago, with an induction furnace for steel-making in New York City; these tests did not lead to any commercial result at that time. Dr. E. F. Roeber then read a paper on the manufacture of ferro-alloys in the electric furnace, and spoke briefly on the electrolytic production of steel facings for cuts and electrotypes, and on electrolytic refining of iron. A large number of ferro-alloys and metals of considerable value were exhibited. Among them were some very large and beautiful specimens of metals and alloys, free from carbon, made by the aluminothermic method of the Goldschmidt Thermit Co.; especially pure chromium metal, molybdenum metal 98 to 99 per cent Mo, manganese metal 99 per cent Mn, ferro-titanium 20 to 25 per cent Ti, ferro-vanadium 25 per cent V. The Roessler & Hasslacher Co. exhibited specimens of ferro-chromium and ferro-tungsten. Mr. F. M. Becket, of the Niagara Research Laboratories, exhibited samples of the following alloys and metals, mostly for high-speed tool steels, but two of the same being used for other special purposes: Ferro-chromium 70.50 per cent Cr, 0.75 per cent C; 62.61 per cent Cr, 0.20 per cent C; chromium-iron-silicide 54.36 per cent Cr, 21.10 per cent Si; ferro-vanadium 38.00 per cent V; iron-vanadium-silicide 20 per cent V, 20 per cent Si; ferro-tungsten 70.80 per cent W; molybdenum metal 96.51 per cent Mo; 0.22 per cent C; 92.71 per cent Mo, 4.40 per cent C; titanium carbide; and copper nickel 51 per cent Cu, 49 per cent Ni. Dr. Paul L. T. Héroult then presented a very interesting review of his work in connection with making and refining steel, which has already been fully covered in these columns. (See also the abstract of a paper of Combes in the Synopsis of this issue.) The discussion which followed was participated in by Messrs. J. T. Morehead, E. Stütz, L. Ruhl, Leonard Waldo, C. A. Doremus and C. S. Hurd.

Dr. Doremus spoke especially on the reduction of pig iron from ores, and pointed out that it is simply a question of the relative cost of electric power and fuel. He showed, however, that there are other considerations to be taken into account, for instance, that it is not necessary to use a certain quality of coke in the electric furnace. Dr. Leonard Waldo gave some reminiscences of early electrometallurgical work done in this country, and spoke especially on Mr. Colby's and his own work with the induction furnace, which he said is quite alive not only in Sweden, but in this country. Dr. Héroult, replying to several questions, thought that the induction furnace has a certain disadvantage in the necessity of using very pure starting materials and in the difficulty of cleaning. It was found that while Kjellin gets very good steels, they are not always exactly the same grade. Mr. C. S. Hurd referred to the possibility of getting cheap electrical energy for electric steel furnaces by utilizing the waste gases of blast furnaces in gas engines, coupled to electric generators.

The Occurrence of Tantalum Minerals.

By David T. Day, Ph. D.

The article on the tantalum lamp in a recent number of this journal has called such considerable attention to the question of a proper supply of this substance that a general review of the occurrence of tantalum minerals in the United States may be valuable at this time.

The following table gives a list of the localities where tantalum minerals have been recorded in various scientific publications. Recourse has been made, particularly to Dana's Mineralogy, the American Journal of Science, and the Transactions of the American Institute of Mining Engineers. The table also shows the percentage of tantalum which can be expected where the minerals are carefully selected.

Quite a different question is the amount of these minerals which can be available in any given locality and the percentage of tantalum which can be expected when the minerals are mined on a large scale and without any very great possibility of carefully cleaning each specimen. Taken altogether, it is evident that the following localities are most promising: Llano County, Texas, Mitchell County and other nearby localities where mica is mined in North Carolina, and the classic locality in Massachusetts from which columbite was first obtained.

In addition to these localities where tantalum minerals have long been known to occur, another source of this element seems fully as promising as any of the others. For many years the Pennsylvania Salt Co. has been importing cryolite into this country from Greenland. This cryolite is never pure, but contains galena and other lead minerals, spathic iron ore, fergusonite, sulphurettes of iron and copper, pachnolite, thomsenolite, arksudite, geoarksudite and hagemanite, and possibly columbite, tantalite, yttrolite, tapiolite and samarskite.

It is very probable that the residues from working up cryolite are still in existence in large quantities at the works of the Pennsylvania Salt Co., which will be valuble for extracting the tantalum. This may very possibly constitute an important source of the metal, where the quantity at hand can be easily measured and the percentage of tantalum readily determined.

Even very low grade material already mined, as is this, should afford an attractive source of the desired element.

In regard to the well-known localities where minerals containing tantalum have been found, it should be noted that fergusonite has been found with quite a number of other rare minerals at a point 5 miles south from Bluffton, Llano County, Texas, and considerable mining has been done at this locality for thorium minerals, so that it will be easy to determine just how much tantalum minerals can be obtained. The deposit here has the form of a mound made up of huge blocks and masses of quartz and red feldspar, as described by Messrs. Hidden and Mackintosh in the American Journal of Science for 1899. This mound has been entered with trenches on all sides, and it is quite easy to trace the occurrence of these rare minerals by the green staining from uranium compounds.

While many localities have been noted for the occurrence of columbiun and tantalum minerals in North Carolina, the neighborhood of Burnsville in Yancey County, and in the adjoining Mitchell County, is the most promising locality for further search. All mica mines in North Carolina should be explored very carefully for tantalum minerals.

Dr. J. Lawrence Smith has also found tantalum near Rockford, Coosa County, Ala., containing 79.65 per cent. of tantalic acid.

Rockport, Mass., is the classic locality from which the original specimens of columbite came which led to the discovery of columbium. While mineral interest in this locality has practically ceased, it should now be prospected again.

The Amazon stone of El Paso County, Col., also contains columbite, and a careful search of that locality is advisable. No doubt the energy with which uranium and vanadium and other rare elements have been sought and found in Colorado within the last few years will be applied also to the search for tantalum minerals with a fair prospect of success.

In the meantime, however, the quickest results may well be sought in the Black Hills of South Dakota, where, during last year, material said to be fairly pure columbite, to the extent of about 2500 pounds, was mined and sold to mineral dealers. Tantalite and other tantalum minerals occur with the tin ores in the Black Hills. It was recorded in the thirteenth volume of the Transactions of the American Institute of Mining Engineers, that Prof. Charles A. Shaeffer, of Cornell University, in 1884, examined some specimens of supposed tin stone, in which he was surprised to find a very slight trace of the further investigation showed the mineral to be tantalite and a considerable amount of the material was found. This came from the classic Etta tin mine, near Rapid City, South Dakota, which has since yielded such large specimens of spodumene in commercial quantity. Professor Shaeffer was unable to find any trace of columbium or tungsten in this material, and drew the conclusion (doubtless justified) that the Dakota tantalite is much simpler in composition than the samples from North Carolina and elsewhere.

In commenting upon this paper by Professor Shaeffer, Professor Blake, who had already noted the black tantalite in the tin ore of South Dakota, calls particular attention to two localities where the mineral tantalite is found in the same dike or vein as cassiterite; first, in the Etta mine, and, second, in the Bob Ingersoll claim. In the Etta mine it should be noted that the tantalite is found in close association with feldspar in crystals from one to four or five inches broad, and from one-quarter of an inch to three inches thick. It is not mixed with the cassiterite and it can be easily taken out. It is easily distinguished from the tin stone by the black color and the dark streak which is very different from that of tin stone. Professor Blake states that it is rarely found in the greisen rock (coarse granite) with the tin ore, although a few crystals have been seen near the outer margin of one of the irregular masses of tin ore. Professor Blake found one very remarkable mass of tantalite which he estimated to weigh over 2,000 pounds, though this was the only body of the mineral seen in that part of the vein. The mineral was found also in a "greisen-like aggregation of mica," not tin-bearing. Professor Blake did not find it in the tin ore veins of the Hill City district.

Prof. Frank B. Carpenter, now of Denver, Colorado, also reports in the Transactions of the Institute of Mining Engineers for 1888, Volume 17, the occurrence of large masses of tantalite or columbite in the Bob Ingersoll claim. His examinations, however, of specific gravity tests, make him pronounce this material columbite instead of tantalite, but he notes that tantalum is always present, but not to such an extent as to be a predominating element.

An analysis of the material from the Etta mine gave 79 per cent of tantalic oxide to Professor Shaeffer, but Doctor Headden of the Dakota School of Mines found only 18 per cent of tantalic oxide but 54 per cent of columbic oxide.

From the descriptions given above, it would seem entirely probable that an adequate supply of tantalum can be easily found for the Siemens-Halske lamp. Further, it is possible that the examination of the heavy sands of the Pacific Coast, to be undertaken by the Division of Mineral Resources of the United States Geological Survey, may yield further supplies of such materials.

Mineral	*Sample from.*	*Percentage of Ta.*
Hatchettolite		
Tantalo-niobate of uranium, near pyrochlore.	Occurs in Mitchell Co., N. C	
Microlite		
Essentially a calcium pyrotantalate [illegible]	Virginia	68
	Greenland, Ferg.	6
Fergusonite	Ytterby, Sweden, wy.	27
Essentially a metaniobate (and tantalate) of yttrium with erbium, cerium, uranium, etc.	Ytterby, Sweden, brn.	8
	Ytterby, Sweden, brn.	10
	[illegible]	[illegible]
	Karafvet, Yttr. gry.	43
	Burke Co., N. C.	4
Columbite	Mineral Hill, Pa.	83/100
Niobate and tantalate of iron and manganese [illegible]	Standish, Me.	[illegible]
	Craveggia, Italy	13
	[illegible]	[illegible]
	Branchville, Conn.	19
	Branchville, Conn.	52
Columbite	Bodenmais, Bavaria	23
	Bodenmais, Bavaria	31
	Haddam, Conn.	29
	Amelia Co., Va.	53
	Northfield, Mass.	57
	Sanarka, Manganotantalite.	80
	Etta Mine, Black Hills, S. D.	18 to 53
	Elsewhere in Black Hills, S. D.	18 to 45
	Mitchell Co., N. C.	9
	Elk Creek, S. D.	34
	Nigger Hill Dis., S. D.	58
Tapiolite		
Tantalate and niobate of iron.	Occurs in Sukula in Finland	74
Yttrotantalate		
The so-called yellow yttrotantalate of Ytterby and Kararfvet belong to fergusonite.	Occurs at Ytterby in Sweden	46
Samarskite.		
Mineral related to samarskite has been found in granitic debris on Devil's Head, Douglas Co., Col.	North Carolina	18
	North Carolina	14
	Devil's Head, Colo.	27
	Devil's Head, Colo.	28
	Devil's Head, Colo.	19
Hielmite.		
Stanno-tantalate (and niobate) of yttrium, iron, manganese, calcium. Formula doubtful.	Occurs in Kararfvet	54
	In Sweden	72
Polymignite		
Niobate and titanate (zirconate) of the cerium metals, iron, calcium.	Occurs at Fredriksvarn and Svenor in Norway. Reported from Moravia and at Beverly, Mass.	1
Polycrase.		
Niobate and titanate of yttrium, erbium, cerium, uranium, like euxenite.	Occurs at Hittero in Norway, and at Slattakra in Sweden	4

CORRESPONDENCE.

Electric Induction Furnace.

To the Editor of Electrochemical and Metallurgical Industry:

Sir.—In view of the great interest which the splendid work of Mr. Kjellin, in Sweden, with the electric induction furnace for steel-making has aroused, I would like to call attention to three fundamental patents for the induction furnace which were granted to Mr. E. A. Colby (of the Baker Platinum Works, in Newark) in 1890. They are Nos. 428,378, 428,379 and 428,552, May 20, 1890

The first claim of No. 428,378 reads as follows: "The combination with a closed primary circuit and means for supplying electric currents thereto and controlling the same of a refractory mass adapted to contain metal to constitute a secondary circuit and adapted to place such metal in inductive relation to the primary circuit, whereby electric currents may be induced therein by the action of electric currents in the primary circuit."

Patent 428,552 refers especially to the melting, refining and casting of metals in such a manner they will be free from blisters, pin-holes and occluded gases. A. N. H.

New York City.

Experiments with an Electro-Thermic Muffle Furnace.

By F. A. J. FitzGerald.

When a boiler has been designed and built it is a comparatively easy matter to make quantitative tests of its performance. From the amount of coal used in heating the boiler we can calculate the total energy used, and by measuring the amount of water evaporated and the steam pressure, the energy obtained in the form of steam can be determined. In dealing with electric furnaces the problem is different, for, while it is a very easy task to measure the energy put into the furnace, it is by no means easy to estimate how much has been used in useful work, and how much has been lost. In commercial work furnaces may be usefully compared by determining the amounts of power used and the output of the product manufactured. In this way the relative efficiencies of the furnaces may be compared, and so far as purely commercial work is concerned, the amount of manufactured material produced per kilowatt-hour is the important point. In making experiments with the electric furnace it would be of great advantage to have some method of determining the efficiency, so that calculations might be made as to what could be done with larger furnaces.

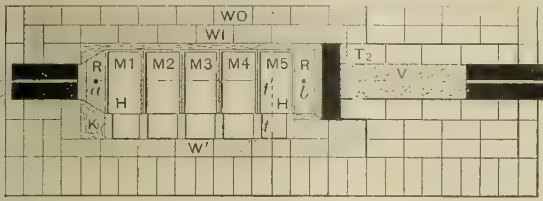

FIG. I.—SECTIONAL PLAN OF FURNACE.

In what follows, there is a description of a muffle furnace built by Mr. Bennie and the writer in order to determine the behavior of such a furnace, and what could be expected of it in actual work. In addition to this a detailed account of certain experiments made with the furnace will be given. It will be very evident that both the construction of the furnace and the method of carrying out the experiments are crude in many respects. Nevertheless, the experiments yielded a certain amount of information that may be used as a guide in the construction of more elaborate furnaces and the carrying out of more refined experiments.

The total power available for the experimental furnace was 20 kilowatts, and this could be used either at 110 volts with 180 amperes, or at 55 volts with 360 amperes. It should be mentioned here that the power factor of the circuit is unknown, so that it may introduce a certain error into all the experiments, but as the latter were not very refined in their details the errors are probably of no serious consequence.

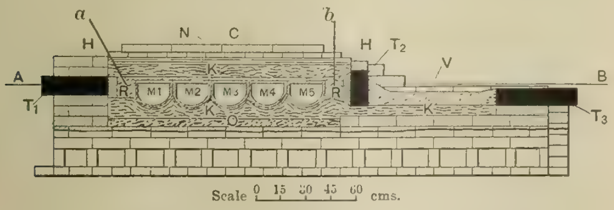

FIG 2.—PARTIAL SECTIONAL ELEVATION OF FURNACE.

In Fig. 1 is shown a sectional plan, and in Fig. 2 a partial sectional elevation of the furnace, the drawings being made to scale. The plan is a section through the line *AB* in Fig. 2, and the elevation a partial section through the line *PQ* in Fig. 1. The base of the furnace has a cellular structure and is about 12 cms. thick above the cells. The furnace contains the five muffles, *M*1, *M*2, etc., and is connected in series, with a regulating resistor. *T*1 and *T*3 are the main terminals of the apparatus, *T*2 the terminal common to the furnace resistor *R*, and the regulating resistor *V*. The connections between the furnace resistor *R* and the terminals *T*1 and *T*2 are made at *H* with graphite. The resistor has two branches, one passing over the top, the other at the bottom of the muffles. The bottom of the furnace is filled with magnesite *O* and the muffles with the resistor embedded in amorphous silicon carbide *K*. At the top of the furnace the silicon carbide is covered with four layers of stout asbestos board, and finally four large tiles *C* are laid on top, resting on the walls of the furnace; *a* and *b* are carbon rods to which voltmeter connections can be made. All the muffles except *M*5 were filled with sand. A porcelain tube passed into *M*5 at *tt'*, so that the inside could be observed during the working of the furnace. The regulating resistor consists simply of a trough filled with granular carbon *V*.

In the following table the dimensions of various parts of the furnace are given:

Furnace:		
Distance between terminals (*T*1 and *T*2)	145	cms.
Total width	91	"
Thickness of wall *WI* + *WO*	23	"
Thickness of *K* between muffles and *WI*	4	"
Distance from open end of muffles to *W'*	18	"
Thickness of *K* above muffles	7	"
Thickness of *K* below muffles at thinest place	5	"
Thickness of furnace bottom	7	"
Thickness of *O*	5	"
Muffles:		
Length	38	"
Width	23	"
Height	16	"
Resistor:		
Width	38	"
Length of upper branch	115	"
Length of lower branch	140	"
Heating surface above muffles	43.7	dcm.^{2}
Heating surface below muffles	53.2	"
Thickness of lower resistor	4	cms.
Thickness of upper resistor		varied
Regulating resistor:		
Length	88	cms.
Width	21	"
Thickness		varied

Before building the furnace described above a similar furnace was built with only three muffles, in order to form an estimate of the energy that would be required to obtain in the muffles a temperature of at least the melting point of steel. The preliminary experiments with this small furnace indicated that 100 watts per square decimeter would be ample. The total heating surface of the resistor amounts to 97 square decimeters; therefore, the furnace should require about 10 kilowatts.

From the preliminary experiments it was also determined that a total thicknss of 8 cms. for the resistor would give the proper resistance. Here, however, a difficulty arose, for, on account of the shape of the muffles the lower resistor had a greater length than the upper resistor, hence, in order that the quantity of energy developed in each should be the same, the thickness of the resistor would have to be inversely as their lengths. Thus, if the lower resistor is 4 cm. thick, the upper resistor should have a thickness of

$$\frac{115 \times 4}{140} = 3.2 \text{ cms.}$$

But another consideration arises at this point, for it was desired to heat the muffles as uniformly as possible, therefore, the number of watts per square decimeter should be the same, both above and below the muffles. It follows then that since the energy developed in the resistors will be inversely proportional to their resistances, the latter should be inversely proportional to the heating surfaces. If r_1 is the resistance of the upper, and r_2 the resistance of the lower resistor, then

$$\frac{r_1}{r_2} = \frac{53.2}{43.7} = 1.22$$

Therefore the proper thickness for the upper resistor would be

$$\frac{3.2}{1.22} = 2.6 \text{ cms.}$$

Still another complication exists, for hitherto we have tacitly assumed that the resistivities of both resistors would be the same. This, however, will not be the case, for the resistivity of the lower resistor will be less than that of the upper resistor, because of the greater pressure on the former by the muffles and their contents. This effect cannot be calculated, so it seemed best to leave the determination of the size for the upper resistor to experiment. Alterations in the upper resistor could easily be made by removing the top of the furnace.

In Fig. 3 there is shown a general view of the 3-muffle furnace. The three bricks projecting from the side wall of the furnace indicate the position of the muffles, as these could be removed so that the interior of the muffles might be inspected. The bottom of this furnace was only 7 cms. thick, and it was observed that after a run of a few hours the loss of heat through the bottom was very great, hence, a considerably greater thickness was used in building the next furnace. The first stage in the construction of the furnace now under consideration is shown in Fig. 4. The lower resistor is in place, and the five muffles, the openings into the latter being shown. The scoop is lying on the top of the muffles. In Fig. 5 is shown the same side of the furnace as in Fig. 4 after the building was finished. In Fig. 6 the other side of the furnace is shown with the rod *b* projecting and connected with the voltmeter. At the left side is shown the regulating resistor with bricks laid on the granular carbon to decrease the resistance.

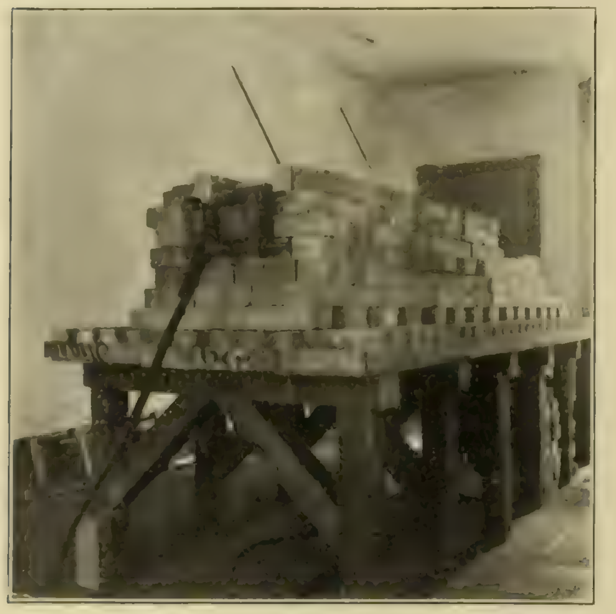

FIG. 3.—GENERAL VIEW OF FURNACE.

After the furnace was built some preliminary experiments were made to find out the best method of manipulating the regulating resistor *V*, so as to obtain the desired generation of energy in the furnace. In order to observe the temperature in *M*5, a thermometer reading to 300° C. was introduced by means of the porcelain tube at *t*. When the temperature in the muffle was too high for the thermometer a carbon rod with a little cup-shaped hollow bored out at one end was put into the porcelain tube and in the hollow a small piece of metal was placed, so that the time of its fusion could be observed. The preliminary experiments were made one afternoon and only lasted for about one hour. The temperature indicated by the thermometer in *M*5 at the end of the experiment was 113°. Next day an experiment was begun in the morning and lasted for nearly 9 hours. Voltmeter and ammeter readings were taken every 15 minutes or oftener, and from these the kilowatts used and the resistance of the resistor calculated. The results are shown in the curves of Fig. 7, where the full-line curve shows kilowatts and the broken-line curve resistance. The temperatures are given at various points on the kilowatt curve, the figures representing degrees centigrade and the word "lead" indicating that lead melted at this point.

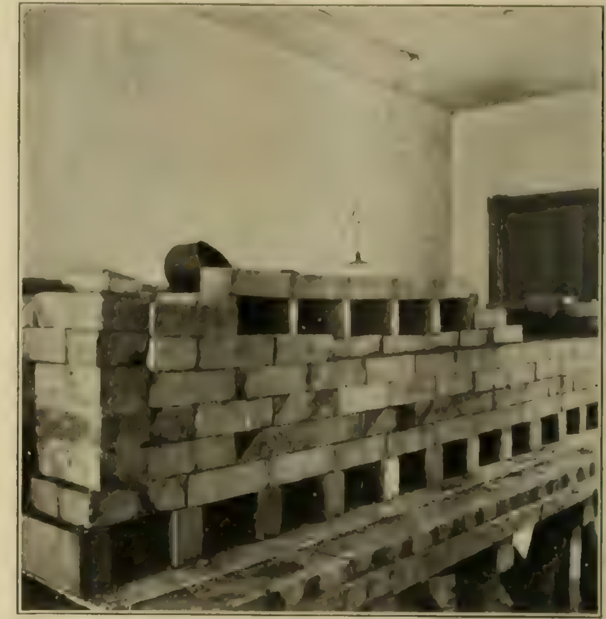

FIG. 4.—FIRST STAGE OF CONSTRUCTION.

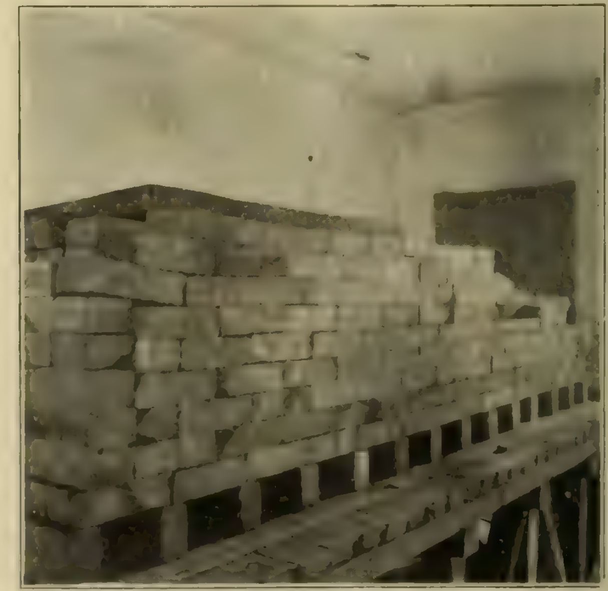

FIG. 5.—FINISHED STRUCTURE.

Considering first the run of the furnace up to 7.75 hours, it is found that the mean ordinate of the power curve is 3.84 kilowatts, which gives approximately 40 watts per square decimeter of heating surface. At 7.75 hours the inside of *M*5 just showed a dull red heat, but the temperature was not high enough to fuse aluminium, introduced by means of the carbon rod. No doubt, in time, aluminium would have fused and the temperature of the muffles and their contents would have reached at least a good red heat, but, except for special purposes, this gradual heating would not be desired. The experiment showed that the regulation of temperature was well under control.

At 7.75 hours the voltage at the main terminals was increased to 110 volts, and the resistance of *V* increased. The result of this is shown in the curves, and it is somewhat surprising, for the resistance of the furnace resistor is increased by 21 per cent at once, and for three minutes more continued to increase until it was 47 per cent greater than before. After that it decreased, and would undoubtedly have reached its original amount if the experiment had been continued. This curious phenomenon is explained by a characteristic of granular carbon resistors.

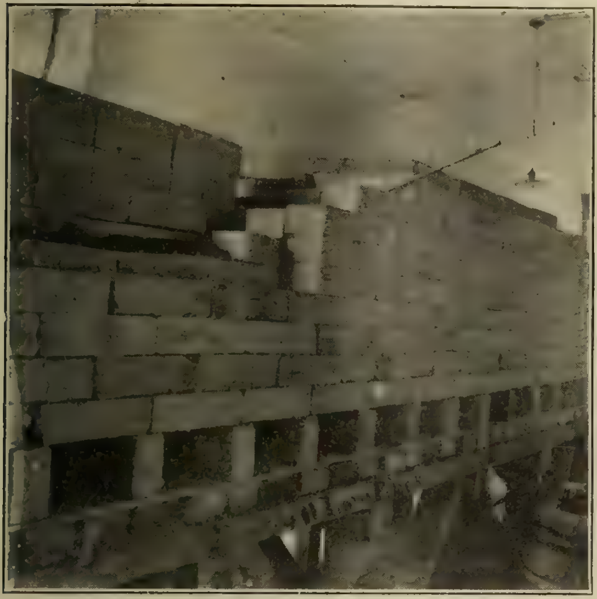

FIG. 6.—OTHER SIDE OF FURNACE.

If a current is passed through a loosely packed granular carbon resistor hundreds of minute arcs are usually formed at first between the carbon grains. After a time, as the resistor becomes hot, these disappear and the crackling noise accompanying the arcs ceases. Apparently, after heating, somewhat better contact is made between the grains, and there is no longer any arcing. Now suppose that there are 50 volts across the terminals of the resistor, and when the arcing has ceased the potential is suddenly increased to 100 volts, it will frequently happen that the crackling noise, due to the formation of the arcs, will begin again and after a time die down when better contact is made. Accompanying this phenomenon is an increase in the resistance of the resistor.

This is what happened in the experiment described above. At the beginning of the run there were 55 volts between terminals *T*1 and *T*3; then the resistance of *V* was gradually decreased by piling bricks on the granular carbon till 6.25 hours, when the e. m. f. between *a* and *b* was 42 volts. No further change was made till 7.75 hours, when, after increasing the resistance of *V*, the e. m. f. between *T*1 and *T*3 was increased to 110 volts. This increased the e. m. f. between *a* and *b* to 65 volts, causing the temporary increase in resistance observed.

In building this furnace the upper branch of the resistor was 4 cms. thick; that is to say, it was made of the same thickness as the lower resistor, in spite of its being shorter and having a smaller heating surface. The observations made in *M*5 showed that although the upper resistor was so thick the generation of heat in the lower resistor was more rapid, for when the muffle first showed a red heat this was noticed in the bottom. This showed that the effect of the pressure of the muffles on the lower branch of the resistor made the resistance lower than that of the upper branch.

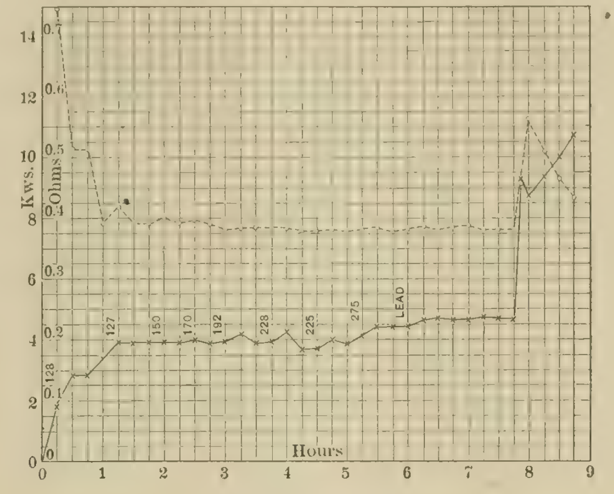

FIG. 7.—CURVES OF POWER AND RESISTANCE.

Although this uneven heating was observed, it was decided before making any change to try another experiment, using more energy. The experiment was made on the day following the previous run, and the furnace started with a temperature well over 100° in *M*5. The results obtained are shown by the curves in Fig. 8. When silver melted in *M*5 at 3.25 hours, the power was cut down with the object of finding out whether, having reached this temperature, the furnace could be held at that point with a much smaller consumption of energy. It was found, however, that the furnace cooled off rapidly, and although at the end of an hour *M*5 still showed a red heat, silver would no longer melt, and after increasing the power it was two hours before a melt was again obtained.

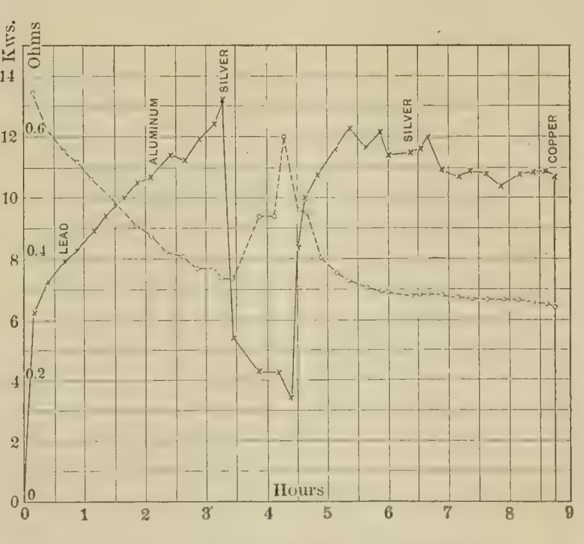

FIG. 8.—CURVES OF POWER AND RESISTANCE.

The resistance curve of this furnace is instructive. When the power was cut down there was at first no change, but presently the resistance began to increase rapidly, due to the cooling of the furnace. It will be observed that the resistance increases to close on the value that it had when the experiment was started, although there can be no doubt that the temperature was much higher than at first. This is due to the formation of carbides in the resistor, the carbides acting as fairly good conductors, while very hot. There was therefore a permanent increase in the resistance of the resistor. The effect was not produced after the first experiment, because the temperature of the resistor was never high enough to form carbides. This explanation of the change in resistance was confirmed later, when the upper branch of the resistor was exposed, for the temperature had been high enough to form a quantity of amorphous silicon carbide on the carbon grains. It is always important to keep in mind the probability of forming carbides when experimenting with resistance furnaces of this kind, for their formation frequently explains effects which at first appear to be somewhat puzzling.

The fact that carbides had formed in the upper branch of the resistor showed that its temperature was far above that observed in $M5$, for amorphous silicon carbide does not form till a temperature well above that of molten steel is reached. The temperature conditions in the other muffles were observed during the run by thrusting copper wires through the interstices of the wall into the sand in the muffles. These observations showed that the temperature in these muffles was higher than that of $M5$ for copper fused in them before fusion was obtained in $M5$. The spaces between the openings of these muffles and the wall W' being filled with sand there was less loss of heat than from $M5$, where the muffle and the space in front was empty.

Before making the next experiment the upper branch of the resistor was uncovered and some more carbon put on it, so as to make it thicker, thus reducing the resistance. The results obtained in the experiment are shown in Fig. 9.

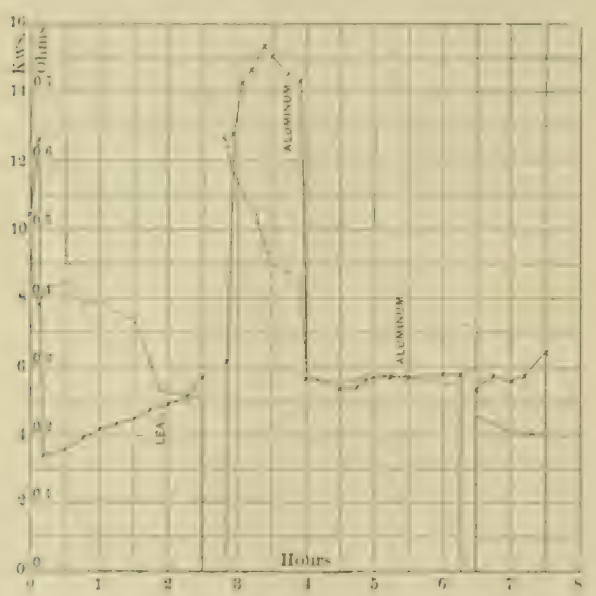

FIG. 9.—CURVES OF POWER AND RESISTANCE.

Although the upper branch of the resistor had been made thicker, the resistance was now much higher than before, because of the carbide formation. It had been hoped that with this increase in the thickness of the upper branch of the resistor, the resistance would be lowered sufficiently to make a run with an e. m. f. of 55 volts between $T1$ and $T3$, but evidently this was not possible, as enough energy could not be developed in the furnace using this low voltage. Accordingly, it was decided to try another experiment. The current was cut off, and after removing the tiles from the top of the furnace and taking off the sheets of asbestos, ten bricks were laid on the amorphous silicon carbide K. When the current was thrown on it was found that instead of a decrease in resistance there was an increase. Evidently in laying the bricks on K some of the amorphous silicon carbide had been forced down into the resistor, thus increasing its resistance. However, with 110 volts at the terminals, the furnace was rapidly heated up and the melting point of aluminium reached in $M5$.

It will be remembered in the experiment of Fig. 8 it was found that when the power was cut down the furnace cooled off rapidly, and it was some time before the temperature could again be raised to the point it was at when the cut was made. In the present experiment another attempt was made to cut down the power and try to hold the temperature constant at the melting point of aluminium. The e. m. f. between the terminals was therefore changed from 110 to 55 volts, and the watts in the furnace held at about 5.5 kw. As nearly as could be observed, this held the temperature, which would indicate that this rate of developing heat was about sufficient to supply the losses by radiation. This seemed to be a serious loss, and indicated that the heat insulation of the furnace was very defective.

The experiment also showed that the upper resistor was unsatisfactory, since its resistivity could be so easily varied by the granular amorphous silicon carbide which covered it. This difficulty might probably have been overcome by covering the resistor with some material in the form of tiles; but nothing of this kind being available at the time, it was decided to use a resistor composed of a mixture of coarse and fine granular carbon, so that it would form a mass, into which the amorphous silicon carbide would not sift. It has been shown elsewhere that as a general rule in making resistors, it is better to use a carefully graded carbon, as otherwise it is difficult to get a homogeneous structure. The present case, however, is a special one, for the resistor is in the form of a wide, thin plate, and it was therefore easy to obtain a homogeneous structure with a mixture of coarse grains and powder. A granular carbon resistor made in this way has much lower resistivity than a resistor formed of coarse grains alone, and experiment showed that the proper thickness was about 1.5 cms.

An examination of the granular carbon composing the upper resistor in the previous experiment showed that it contained considerable quantities of amorphous silicon carbide, which formed a coating over the surface of the carbon granules. The surface of the muffles also showed that the refractory material of which they were composed had been superficially fused, and it was undoubtedly from the silicon contained in them that the amorphous silicon carbide coating the grains of the resistor was derived. As Moissan has shown, silica has an appreciable vapor tension below its temperature of fusion, and when actually fused vaporization is comparatively rapid. This having taken place in the upper resistor it was plain that the same thing must have happened in the lower resistor and consequently it was not surprising to find an increase in its resistivity.

The results obtained from the final experiment are shown in Fig. 10. At the end of 4.75 hours copper was at once fused when introduced into muffle No. 5, and when pieces of iron wire were thrust into the sand contained in the other muffles they were instantly fused. About this time it was observed that the readings of the ammeter became very unsteady, and the curve shows that the resistance of the resistor was varying considerably. This made it evident that some serious disturbance was taking place in the furnace, the most probable explanation being fusion of the muffles. Meanwhile, the temperature in $M5$ had diminished, and just before cutting off the current at the end of the experiment copper would no longer melt. The fusing of the muffles would greatly increase the resistance at the point where the fusion took place, the generation of heat at that place would be increased, diminishing the generation of heat at mufflle No. 5, and consequently allow it to cool. When the furnace was taken down after cooling, it was found that two of the muffles had fused, as shown in an illustration to a previous article (Electrochemical Industry, Vol. II, No. 12, page 439).

An examination of the granular carbon of the resistor showed that at the muffles where fusion had taken place the temperature had just reached the point of the formation of crystalline silicon carbide.

From this experiment, as well as from those that went before, it was plain that the heat insulation of the furnace was very far from being sufficient. At the end of the run the bricks covering the cells at the bottom of the furnace were red-hot, as were the pieces of asbestos board on the top of the furnace. The side walls were also very hot, being well above 100° C. A further evidence of defective heat insulation was found in the muffles, for it was observed that while the sand in the center parts of the muffles had been heated well above the melting point of iron, the temperature of the sand at the ends of the crucibles had only reached the melting point of copper. In the experiment of Fig. 9 it was shown that about 5.5 kilowatts were used to supply the heat lost by radiation from the furnace, since with that rate of generation of energy the temperature remained stationary, and in the experiment now considered, probably a much larger amount of energy was dissipated in this way, since the temperature was higher.

It was the evidence of this loss of heat that led to the use of a cellular bottom to furnaces having the cells filled with some good heat insulator, such as infusorial earth. The design of such a furnace bottom is shown elsewhere (Electrochemical and Metallurgical Industry, Vol. III, No. 2, page 55), and has been found to effect a great saving of heat. With the same object, the side walls should, when possible, be double, with the space between the walls filled with a heat insulator. Finally, the top of the furnace should be insulated in a similar manner. A close attention to heat insulation is of less importance when dealing with much larger furnaces, since the ratio between the superficial area of the furnace and its volume is much less than in the small apparatus, but it is doubtful if sufficient attention is directed towards this subject.

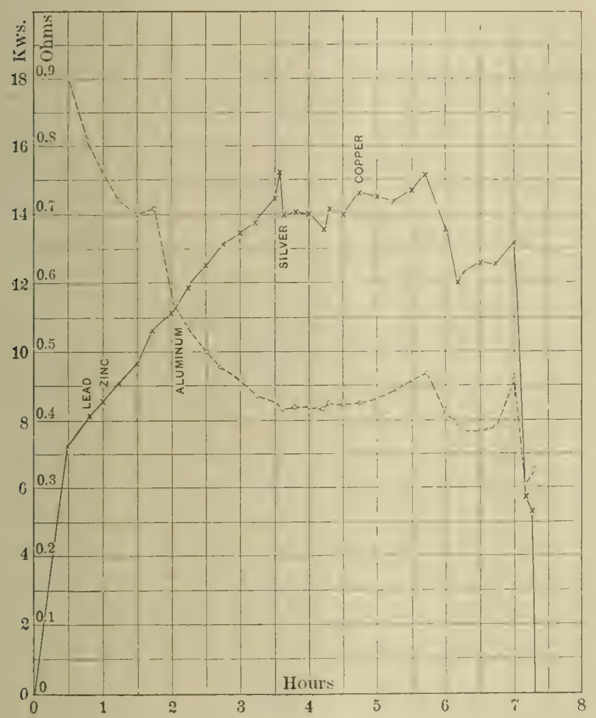

FIG. 10.—CURVES OF POWER AND RESISTANCE

The calculation of the efficiency of an electric furnace is always very difficult, as was pointed out before, because of the lack of trustworthy data on which to base the estimates, but an attempt will be made to obtain some notion of the efficiency of the furnace described above.

In this furnace the useful work will be considered as the heating of the muffles and their contents. The main constituent of these materials may be assumed to be silica in the form of quartz, and a conservative estimate will place their increase of temperature at 1100° C. in the last experiment at the end of six hours' run. We then have the following data:

Mean ordinate between 0 and 6 hours, 12.14 kw.
Kilowatt-hours, 72.84.
Silica raised 1100° in temperature, 250 pounds.
Mean specific heat of quartz, 0.3.

Total heat units given to silica, the heat unit being that required to raise 1 pound of water 1° C., was $1100 \times 250 \times 0.3 = 82{,}500$.

Total heat energy developed in the furnace was

$$1890 \times 72.8 = 138{,}000.$$

Hence, the efficiency of the furnace was approximately 60 per cent.

The figure used for the specific heat of quartz is based on Pionchon's determination for quartz between 400° and 1200°. Other observers working at lower temperatures give other figures varying from 0.2 to 0.3. If the figure taken is 0.25, instead of 0.3, the efficiency of the furnace would be 50 per cent. The important point to observe, however, is that the efficiency is decidedly low, and shows very clearly the faulty design of the furnace.

It is, of course, obvious that if the muffles were unloaded while the furnace was running, and then loaded again, the efficiency would be very much higher for the next heat, for it would not be necessary to expend a large amount of energy in raising the temperature of the furnace itself.

In order to heat a muffle furnace of this kind rapidly it is necessary to use a very high resistor temperature, and to do this some other material than that used in making ordinary muffles must be employed. For this purpose silico-carbide is indicated. One of the great advantages of electrical heating is lost if we cannot push the temperature of the resistor to a very high degree, thus saving time in heating the material. It is, of course, obvious that in building an electrical muffle furnace along the lines of that described, the muffles should have a square cross-section, as the building of a curving resistor is by no means easy.

In the last experiment, if we calculate the watts per square decimeter of heating surface, from the value of the mean ordinate, we find that they amount to 125. With a furnace of better design as regards heat insulation, it is evident that 100 watts per square decimeter would be ample where the object was to raise the temperature of the muffles and their contents to the melting point of steel.

The granular carbon resistor as used in the furnace showed some objectionable features. It was evidently very susceptible to mechanical influences which caused great changes in its resistance; but a still more serious objection was the increase in resistance produced by the formation of carbides. This trouble will always be experienced where there are silicious bodies at a high temperature in proximity to the resistor. Probably the difficulty could be overcome to a great extent by using a refractory substance such as magnesia in contact with the resistor, since magnesia will not form carbide by heating with carbon to a high temperature. Some experiments made following this idea seem to be promising, but have not yet advanced very far. Finally, the experiments with the furnace suggest that solid resistors would have certain marked advantages; but a discussion of these must be reserved for another article.

Hittorf's Migration of Ions.—In *Science*, of January 27, F. H. Getman describes a model for explaining in the lecture room Hittorf's conception of migrating ions. The cations and anions are represented by black and white balls, respectively, sliding on wooden rods. Above the rows of balls a scale is provided to measure the speeds.

Refractory Materials for Furnace Linings.

There is scarcely a detail in metallurgical furnace practice which requires greater attention than the provision of a refractory furnace lining. According to the temperature of the furnace, different furnace linings will, of course, be used. Materials specially useful for the temperatures in electric resistance furnaces were discussed very fully in an article of Mr. F. A. J. FitzGerald in our November issue, 1904, page 430.

The subject of refractory materials for furnace linings has recently been the subject of an able paper by Mr. E. Killburn Scott before the Faraday Society. In the following we give an abstract of this paper.

The author divides refractory materials into:

I. Carbon for the highest temperatures.

II. Silicon-carbons made in the electric furnace, such as carborundum, or crystallized silicon-carbide, siloxicon and amorphous silicon carbide.

III. Magnesia made in the electric furnace.

IV. Ordinary fire brick, magnesium brick, etc., for the lower ranges of electric furnace temperature.

The author deals with the second and third of the above groups. He emphasizes that it is not necessary that the refractory linings should be homogeneous throughout, but that such ordinary materials as firebrick, magnesia brick, etc., may be washed over a depth of half millimeter or so, with a highly refractory material with greatly improved results.

Carborundum, Amorphous Silicon-Carbide, and Siloxicon.

In discussing first the use of carborundum as refractory material, Mr. Scott says that the corborundum is ground up very fine and mixed in the proportion of three parts by weight of carborundum to one part by weight of silicate of soda (water-glass). After thoroughly brushing the newly set fire brick to get rid of dust etc. (the mixture will not stick to a surface which has been already fired), the carborundum is painted on to the depth of about half a millimeter. It is then left for twenty-four hours to dry, and afterwards the firing started up gradually.

The result is that the fine particles of carborundum become cemented over the whole surface of the fire brick lining, cracks and all, and if properly done, it adheres quite soundly.

Carborundum taken from the core of the electric furnace is infusible at 7000° F., and is unaffected by oxygen, ozone, or sulphur at 3000° F. It is also unaffected by chlorine, bromine, or boiling concentrated sulphuric acid, gaseous hydrochloric, nitric, or hydrofluoric acid.

To show how this coating will resist high temperatures, Mr. Scott mentions that during the Düsseldorf exhibition, a welding furnace with coke firing and blast used for the welding of tubes worked daily from 9 o'clock until 7 at night for nearly six months, and the carborundum lining was practically in the same condition at the end as at the beginning of the run.

Where basic slags or basic materials have to be taken into consideration, fire clay is employed as a binder, instead of the water-glass, the proportion being usually six parts by weight of carborundum, to one part by weight of fire clay.

Carborundum, mixed with fire clay in equal parts, has been used for repairing cracked gas retorts in Berlin with considerable success. This was done whilst they were still hot, and they ran for a considerable time afterwards, perfectly gas-tight.

For pointing the joints in brickwork of coke ovens, for recovery of by-products, a mixture of carborundum and fire clay has been found most useful. The mixture is tamped in to about half an inch deep, and it completely closes the oven chamber against the flues, so that there is no entering of gases through the heating walls.

Carborundum has also been used for lining iron and other metals. In this case, three parts by weight of carborundum are mixed by weight with two parts of silicate of soda. Where it is used for this purpose, however, the metal must remain constantly heated, as otherwise the coating will burst off when the metal cools down.

Carborundum coatings have proved very suitable for coating iron, etc., which is subjected to considerable heat; thus, in the case of some central iron pipes for a particular blast furnace, their life was increased from two and one-half to nine months.

The temperature limit to the use of carborundum is reached when it decomposes according to the equation:

$$SiC = Si + C$$

the silicon escaping as vapor, while the carbon remains as graphite. When the temperature is so high that this decomposition occurs, there appears to be no refractory material that can be used, except carbon, and the best form of carbon for the purpose is charcoal, since its electrical resistivity is so high. Possibly further study of electric furnace products may produce other highly refractory materials.

Mr. Scott exhibited various samples, among them fire-clay bricks with a carborundum surface, to be used at the highest possible temperature; also, a piece of Glenboig fire brick coated with carborundum and taken from a furnace heated with coal dust firing; the brick was exposed to a very high temperature, but the carborundum surface had protected it in an exceedingly good way.

Mr. Scott also showed an injector furnace, one-half of which was coated with carborundum. This furnace was heated with a blow-pipe, whereby the uncovered part commenced to flux, while the corborundum covered part remained perfectly intact. A fire clay crucible coated with carborundum, which had been exposed to oxy-hydrogen blow pipes, was exhibited to show how efficiently carborundum had acted as a protector. A carborundum crucible made from 90 per cent carborundum and 10 per cent fire clay, samples of carborundum powder for refractory purposes, and carborundum crystals as obtained from the electric furnace, were also exhibited. Mr. Scott then gave a synopsis of the properties and applications of amorphous silicon carbide and siloxicon. Concerning the same, the reader may be referred to the article of Mr. FitzGerald quoted above.

Magnesia.

In a communication published on page 454 of our November issue, 1904, Mr. Scott referred to some experiments made by him on electrically-shrunk magnesite and some remarks were made on the same subject on page 455 by Mr. F. A. J. FitzGerald. Mr. Scott now deals on this subject at greater length, and for this reason this part of his paper is given in full as follows:

"Most of the magnesite at present used comes from Styrie in Austria; the Island of Euboea, in Greece; from California, and Salem in Southern India. The ordinary calcined mineral, generally in the form of brick, is now recognized as the best material for lining basic open-hearth furnaces, cement kilns, etc. It may be employed to advantage wherever high temperatures and chemical reactions are usually detrimental to dolomite, chromite, and silica. America uses practically all calcined magnesite for steel furnaces, whereas in England dolomite is chiefly used, while magnesite is coming in, although it is double the price. The distinctive characteristics of a calcined magnesite lining are durability, freedom from moisture and silica and resistance to corrosion when exposed to the action of basic slags and metallic oxides. These qualities make the lining cheaper than most others in the long run.

"The usual method of dealing with the crude magnesium carbonate which is found in a more or less pure state is to drive off the carbonic acid gas as far as possible by calcining, in a kiln. The kilns of the Anglo-Greek Magnesite Co., on the Island of Euboea, were originally roughly built kilns fired with wood, but now modern shaft calciners are employed fired with lignite coal. The temperature, however, is not sufficient to thoroughly shrink the magnesium carbonate, and it is even doubtful whether rotary kilns would do it satisfactorily.

"About two years ago the writer carried out some original experiments in Meraker, Norway, with a view to thoroughly shrinking the magnesite in an electric furnace. The furnaces were of the Siemens & Halske type, built for the manufacture of carbide of calcium on the intermittent system, and they were supplied with current from 900-kw. three-phase generators, driven by water turbines. The material on which the writer worked was crude magnesite, brought from Salem, in Southern India,* which is found in a remarkably pure state.

"The writer found that it was possible to simply pass the magnesite down the chutes in the same way as the raw materials for making carbide of calcium. When once the arc was fairly started the raw magnesite could be fed in at a rapid rate and completely shrunk, that is to say, all the CO_2 driven off, without the assistance of any carbon or other material.

"After a two hours' run with 3500 amperes at 65 volts, a block about 15 inches cube was obtained which was practically pure crystallized magnesite. When broken into pieces with a sledge-hammer the material showed a fine crystalline structure with a beautiful iridescent color, and it had evidently been in a semi-plastic condition. Attempts were made to mould the half-shrunk material into bricks and then treat these in the electric furnace, but with poor success.

"On returning to England in November, 1902, the writer found that Mr. Deighton, of the Deighton Patent Flue & Tube Works, Hunslet, was trying carborundum as a wash in the furnaces, and also on the refractory blocks surrounding the water-gas flame when heating the steel tubes. I showed the electrically-shrunk magnesite (magnesia) and suggested that it would probably be as good and considerably cheaper then carborundum.

"This electrically-shrunk magnesite has been tested as a wash over the fire brick lining of a carbide of calcium furnace, and it was found that the bricks lasted for 200 hours without repair, whereas the unprotected bricks required repair after a five hours' heat. There is no doubt that the crystallized material, without further preparation than being crushed to suitable dimensions, will prove of special value as a refractory material in metallurgical practice, and an important point in connection with its use as linings for electric furnaces is that magnesia, unlike lime, does not form a carbide with carbon.

"In the large open-hearth steel furnaces at the Homestead Steel Works, Pittsburg, the writer found that large quantities of ground calcined or partly shrunk magnesite were used for the underbeds of the furnaces, and it is probable that electrically-shrunk material would be still more effective.

"In replying to a letter by the writer in Electrochemical Industry for November, 1904, Mr. FitzGerald states that he considers that the electrically-shrunk magnesite is not as refractory as silico-carbides, or carborundum, and he instances a case of a furnace in which calcined magnesite was used for part of the lining. The temperature at the inner surface of the magnesia lining was well below that of the formation of the carborundum, but above that of the formation of the silico-carbides. The heat lasted about eight hours, and the lining, which was about 4 cm. thick, was completely fused. Probably the reason for this was because Styrian magnesite, which is so largely exported to America, was used. This magnesite unfortunately contains a somewhat high percentage of iron, and this causes it to flux easily. Further, the magnesite was not crystallized, but only partly burnt.

"A pure magnesite, such as the Indian, when crystallized certainly does not run at the electrical furnace temperature. In the writer's experiments at Meraker it was never more than in a semi-plastic condition, even the temperature obtained in the thermit compounds had no effect on it."

Mr. Scott gives the following report of his test made on October 29, 1902, to produce electrically-shrunk magnesite:

"Twelve bags of calcined magnesite screened through a sieve with meshes about half an inch by a quarter, and the material then fed into the chutes leading down to the furnace. Started furnace with a shovelful of charcoal, and allowed some magnesite to run in. The material took longer to heat up than in making calcium carbide, but in about fifteen minutes was able to raise the top electrode, which showed that it was going to be possible to treat the material *without* coke. Later on, found it was much better to work without coke.

"In three-quarters of an hour the handle raising the electrode had been given 8½ turns, which meant that it had been lifted 5¼ inches, and in one hour it had been raised 7½ inches.

"The process was continued for two hours, by which time all the twelve bags had been used up and the furnace filled to within 6 inches of the top. The furnace was left to cool, and on the following day the hard core immediately between the electrodes was taken out and weighed about 3 cwts. and measured about 15 inches cube.

"The operation throughout was very quiet, this no doubt being partly owing to the absence of coke. Occasionally, when the material ran short and the edge of the electrode became uncovered some material would be thrown up with considerable violence.

"On breaking the block with a sledge-hammer the center and bottom was found to be a very dark gray, leading off to white on the outside of the block. There was also a considerable amount of yellowish white fused material and some reddish brown.

"Throughout the two hours the voltage was 65 volts, and the current kept as near 3500 amperes as possible.

"To make 3 cwts. of crystallized magnesite took about 200 horse-power for two hours."

Electrolytic Treatment of Electrolytic Slime.

By Anson G. Betts.

Electrolytic slime has not a very pretty name, nor is it a very pretty material to look at or smell of, but it is the final mixture of everything known to the metallurgical world, and it presents a very pretty problem to the chemist, who has it forced on him to treat it for next to nothing, separate the worst possible tangle of metallic elements, save everything contained in its most valuable and marketable form, not lose any of it, do it in a hurry, without any complication in plant. But it has been found possible to meet all these requirements by electrochemistry. Electrochemistry seems destined to supplant almost all inorganic chemical and metallurgical processes, but only by the most painstaking effort. We need chemicals to make chemicals and metals, and they can be handled in no way as cheaply as electrochemically. Even as cheap a chemical as sulphuric acid, at five-tenths of a cent per pound, which will cover the cost of production and a fair profit, can be moved from one combination to another, allowing two volts, for about one-half kilowatt-hour, worth at a water-power, say, two-tenths of a cent.

Fusion processes have been used for slime, but the separations cannot be as complete as with wet processes, where we can deal with such individual properties of the different elements as electromotive forces of solution and solubilities in different solvents, and even practical requirements leave us some choice in the use of these properties. Fusion also causes loss by volatilization and absorption.

It was my intention to give a table showing all the wet processes tried, to illustrate their possibilities and objections, but it would occupy a good deal of space and not be interesting to most readers. For several years past I have been working on this problem, with the assistance of Dr. Edward F. Kern, and I think I have practically exhausted the possibilities of electrochemistry in this small field, and have gained in the laboratory the experience that could only be gained in a works in a much longer time, for it has been easy to make changes and try processes in a few hours that would take as many days or weeks in a plant. No scientific butterflies have been chased, for the only questions are, first, whether the solution it is desired to use is cheap enough, safe enough and easily enough contained, and then to see if it will work. What is presented here is, therefore, mainly the result of the elimination process.

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* These magnesite deposits have been known to the natives for many generations, and they use it for plastering the inside walls of houses. Being absolutely fireproof and self binding, it is an excellent material for the purpose. It has been suggested to make partition walls for houses from it by smearing onto a textile fabric as a base, and letting it set in a corrugated shape. The samples the writer has seen appear to promise well.

Electrolytic slime from lead refining varies in composition, generally much less than would be expected, except in content of bismuth, and analyses are given in Table 1.

Electrolytic slime from copper refining is of somewhat different nature. Arsenic, antimony and bismuth stand slightly above copper in the electromotive force series, and below lead by a considerable margin of some 0.3 volt or more, so that in lead slime these metals are found as metals, and in copper slime are present for the most part in the oxidized condition. The sulphur in copper slime comes from using anodes of blister copper, which, according to common theory, contains dissolved sulphurous acid, but more probably cuprous oxide and cuprous sulphide, which have not reacted on each other. On dissolving as anode, the cuprous sulphide goes into the slime, accounting for the high copper content of some slimes. A few analyses are given in Table 1.

TABLE I.—SLIME ANALYSES.

No.	Anodes.	% Cu.	% Ag.	% Sb.	% As.	% Pb.	% Bi.	% S.	% Fe.	Oz. Au.	% Se	% Te	
1	Lead, Trail, B. C.....	8.83	28.15	27.10	12.42	17.05	nil	......	1.27	......	....	....	*Trans. Am. Inst. Min. Eng.*, 1904, p. 182.
2	" "	22.36	23.05	21.16	5 40	10.62	nil	......	1.12	......	......	....	
3	Lead, Monterey, Mex	1.90	32.11	29 51	9.14	9.05	trace	......	.49	29.1	......	....	*Trans. Am. Inst. Min. Eng.*, 1904, p. 183.
4	Lead, Mexican........	9.30	4.7	25.32	44.58	10.30	.52	......	......	......	......	....	
5	Lead, Mexican........	6.38	3.90	50.16	15.23	5.30	19.74	nil	nil	.....	......	..	Original.
6	Lead, Trail, B. C.....	1 40	31.62	35.71	4.91	9.57	......	......	.....	180.33	.	....	*Mines and Minerals*, Vol. 25 (1905), p. 288.
7	Lead, Trail, B. C.....	6.60	32.21	24.60	2.20	12.60	......	......	......	81.99	......	...	
8	Rich Lead, Parke's Process	12.56	78 45	4.12	.....	3.00	.88	......	......	.....	.. ..	......	Original.
9	Lead, Trail, B. C.....	7.10	29.20	30.50	6.10	10.20	......	......	......	.....	......	.. ..	Original.
10	Lead, Trail, B. C	7.70	31.90	37.60	2.80	12.60	......	......	.....	.....	......	......	Original.
11	Lead from El Doctor Mine, Mex..........	7.82	2.44	75.34	0 24	12.23	1.95	......	.....	......	......	......	Original.
12	Copper, Montana Converter Anodes......	41.	24.	......	......	......	.....	......	......	18.	....	...	*Trans. Am. Inst. Min. Eng*, 1904, p. 310.
13	Copper, Montana Reverberatory Anodes.	18.	51 4	......	......	......	....	......	......	38.	.... ..	....	
14	Copper, Boston & Montana............	57.	14.80	2.00	2.60	5.26	5.70	......	......	......	2.0	1.0	Original.
15	Copper, Boston & Montana............	53.29	12.90	3.30	1.15	trace	1.55	11.96	......	.....	......	......	Original.

The particular reactions of the finely divided metals as they exist in slime, with which we are here interested, are about as follows:

Lead, with sulphates and oxidizing agents is changed, of course, to insoluble lead sulphate. In the final fusion with soda, lead sulphate is easily fluxed.

Readily changed by HCl in a current of air, chlorine, ferric chloride, etc., to sparingly soluble lead chloride; not dissolved by sodium hypochlorite, sodium chloride solution, or soda carbonate in air current, and somewhat dissolved by $NaOH$ in air current. Sodium thiosulphate in air current, sodium tetrathionate and sodium thiosulphate, and sodium sulphide in air current, produce only lead sulphide. Ferric fluosilicate, lead peroxide and fluosilicic acid dissolve lead very easily as led fluosilicate.

Copper very readily dissolves as cupric sulphate by treatment with ferric sulphate. The copper sulphide present in most copper slime is rapidly decomposed, but for good results the temperature should not be high enough to cause the separated sulphur to cohere. Readily changed by HCl in a current of air, and by other chloridizing agents, to cuprous and cupric chlorides. Ferric fluoride gives readily soluble cupric fluoride. Cupric fluoride and copper do not appear to react to form cuprous fluoride, analogously to cupric chloride and copper. Sodum hypochlorite gives no appreciable action. Sodium thiosulphate in air current makes some copper sulphide.

Ferric fluosilicate, fluosilicic acid and lead peroxide, give readily soluble cupric fluosilicate.

Silver, with ferric sulphate solution, readily dissolves hot

TABLE II.—SOLUBILITY OF SILVER SULPHATE IN 100^{cc} SULPHURIC ACID.

Temperature.	Distilled Water	5% H_2SO_4 Sol.	10% H_2SO_4 Sol.	20% H_2SO_4 Sol.	30% H_2SO_4 Sol.
20° C	0.774 grams Ag_2SO_4 0 536 " Silver	1.010 grams Ag_2SO_4 0.699 " Silver	0.939 grams Ag_2SO_4 0.650 " Silver	0.658 grams Ag_2SO_4 0 455 " Silver	0.518 grams Ag_2SO_4 0.359 " Silver
40° C	0.979 grams Ag_2SO_4 0.678 " Silver	1.637 grams Ag_2SO_4 1.133 " Silver	1.611 grams Ag_2SO_4 1.115 " Silver	1.159 grams Ag_2SO_4 0.802 " Silver	0 931 grams Ag_2SO_4 0.644 " Silver
65° C	1.185 grams Ag_2SO_4 0.820 " Silver	2.884 grams Ag_2SO_4 2.000 " Silver	2.735 grams Ag_2SO_4 1.893 " Silver	2.318 grams Ag_2SO_4 1.605 " Silver	2.072 grams Ag_2SO_4 1.434 " Silver
90° C	1.361 grams Ag_2SO_4 0.942 " Silver	4.040 grams Ag_2SO_4 2.800 " Silver	3.863 grams Ag_2SO_4 2.674 " Silver	3.644 grams Ag_2SO_4 2.523 " Silver	3.705 grams Ag_2SO_4 2.565 " Silver

as silver sulphate. The reverse reaction appears to take place on cooling to just the extent that silver sulphate crystallizes out, so that silver sulphate *in crystals* is reducible by ferrous sulphate. When sulphur is present in finely divided form this reaction appears to take place,

$$Ag_2SO_4 + S + 2FeSO_4 = Ag_2S + Fe_2(SO_4)_3$$

For solubility of silver sulphate in sulphuric acid of various strengths see Table 2. Silver in slime is readily changed by chloridizing agents, even by sodium hypochlorite to silver chloride. Hot ferric fluoride gives soluble silver fluoride, as does chromic acid and hydrofluoric acid. Sodium thiosulphate and air produce silver sulphide to some extent. Alkalies and air current give no action, but alkaline sulphides would probably give silver sulphide. Ferric fluosilicate, and lead peroxide and fluosilicic acid, give soluble silver fluosilicate.

Gold in the form in which it occurs in electrolytic slime is only incompletely dissolved with free chlorine and chlorine-generating compounds.

Bismuth is peculiar in that it exists in solution in two different series of salts, which fact appears not to have been previously called attention to. Bismuth and a cold ferric sulphate solution containing free sulphuric acid react very rapidly with almost complete solution. If this solution is heated the bismuth sulphate formed is changed, and basic bismuth sulphate separates. On cooling, the reverse change does not take place. The precipitate is not soluble in sulphuric acid, but if it be washed with soda or ammonia it readily dissolves in cold sulphuric acid. The reaction between bismuth and ferric sulphate takes place slowly on heating, as the metal becomes covered with an insoluble crust of basic sulphate, which protects it. The precipitation of basic bismuth sulphate, on heating, affords a means of precipitating pure bismuth compounds. In the ordinary slime treatment bismuth goes into oxide or basic sulphate. By washing the residue with soda, and extracting with cold 20 per cent sulphuric acid, it can be extracted, but not completely. Chlorine, HCl in air current and other chloridizing agents, readily convert bismuth to chloride. Ferric fluoride gives insoluble bismuth basic fluoride. Ferric fluosilicate and lead peroxide and fluosilicic acid react analogously to ferric sulphate. The combination with bismuth salts of sodium thiosulphate has not been found stable enough to be made use of. Alkalies and sodium sulphide under oxidizing influences will not dissolve bismuth. Sodium hypochlorite has no action.

Antimony is readily converted by ferric sulphate to insoluble trioxide or basic sulphate. This can be readily extracted from the slime with hydrofluoric acid. Antimony trifluoride is entirely dissimilar to antimony trichloride in its reaction with water, not being decomposed by dilution of the solution. It is a very soluble salt; the specific gravity of the saturated solution is nearly 2.5. By electrolysis of antimony trifluoride with lead anodes and cathodes, antimony is deposited, and hydrofluoric acid set free. Chlorine, HCl in air current, ferric chloride, and other chloridizing agents readily form antimony trichloride. Ferric fluoride and antimony pentafluoride readily give antimony trifluoride. Ferric fluosilicate acts analogously to ferric sulphate, but the solution takes up as much as 0.4 per cent of antimony. Addition of tartaric acid to ferric sulphate, even in large amounts, causes the solution of only a small amount of the antimony oxidized. Caustic soda in air current oxidizes antimony somewhat, but dissolves very little, contrary to an incorrect statement by Whitehead.[1] Sodium hypochlorite does not act on antimony. Sodium sulphide in a current of air oxidizes and dissolves antimony rapidly. Sodium carbonate in a current of air causes some oxidation. Sodium thiosulphate in an air current, no action.

Arsenic is the metal most easily dissolved in slime, as it readily dissolves under oxidizing conditions in both acid and alkaline solutions. Sodium hypochlorite dissolves it rapdly. The difficulty with arsenic is not so much to dissolve it, but to dispose of it in solution. It cannot be well electrodeposited, and stands so near antimony, copper and bismuth in the electromotive force series, that it cannot be precipitated by them. Copper can, however, be precipitated in presence of arsenic by electrolysis.

Slime Treatment Processes in Present Use.

Electrolytic copper slime was originally treated, I understand, with nitric acid, which was no doubt good enough when silver in copper was not paid much for, and when it was worth $1.20 per ounce, but the process has been improved, until slime is now generally treated with sulphuric acid and sodium nitrate, which takes out arsenic and copper, and would take out some silver but for the fact that copper plates are hung in the solution to throw dissolved silver out. The current statements that copper slime is treated with sulphuric acid, and air blown through for oxidation are misleading, for the reaction is very slow. Passing sulphurous acid in with the air to oxidize sulphurous acid to sulphuric is worse yet, for the reaction will hardly take place at all.

Air and steam are admitted into the solution containing the slime suspended, for the double purpose of agitation and heating. The copper sulphate produced is usually crystallized, and some plants have apparatus for purifying the solution from arsenic and antimony before crystallizing. The slime is filtered, sometimes with a filter press, and dried in large iron pans placed in or near a flue, or on a large iron floor with a fireplace underneath. Still containing the original antimony, bismuth, lead and sulphur, the slime is melted with soda to slag the antimony, and nitre is always used to oxidize the impurities from the bullion. The doré bullion produced is usually parted with sulphuric acid, and sometimes electrolytically with a silver nitrate bath.

The main objections to this useful process are: Cost of the sulphuric acid and sodium nitrate, and loss of time in completing the reaction. Loss of antimony and bismuth, cost of soda used to slag off antimony, and of nitre for refining. volatilization of antimony, and long time the bullion is in the furnace, both of which cause loss of silver. These objections, while making a serious enough item in the cost of refining a ton of copper producing 15 pounds of slime, make a cost altogether too high per ton of lead, producing 60 to 100, or even 300 pounds of slime per ton.

Electrolytic Slime Treatment.

The improvements brought about by electrolytic treatment are regeneration of the sulphuric acid used as solvent, and of the oxidizing agent at the same time, at a much lower cost than for new chemicals, recovery of the copper as electrolytic copper, and incidentally converting outside copper oxide into electrolytic metal, and recovery of the arsenic as arsenious acid if desired, and practically complete recovery of antimony and bismuth as pure metal. Simple melting of a smaller quantity of material, with a smaller amount of soda and no oxidizing agent, in closed crucibles or retorts in much less time and without loss by volatilization, the volatile antimony having been removed. Electrolytic parting of the silver, gold, bismuth, with traces of copper, antimony arsenic, by an improved method at less cost, and with better results.

A scheme of the process is given as Fig. 1.

The ferric sulphate solution as it comes from the electrolytic tanks contains 4-5 per cent of iron as ferric sulphate, 0.5-1 per cent as ferrous sulphate, 1 per cent copper as cupric sulphate, and from 4-6 per cent of free sulphuric acid. The solution is agitated in a lead-lined tank, by means of a jet of air and steam in the usual manner, and copper oxide, copper scale or similar material added, which dissolves rapidly, any cuprous oxide being entirely brought into solution because of the presence of ferric sulphate. The slime to be treated is then added, when these reactions take place quickly:

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[1] Mines and Minerals, Vol. 25, 1905, Jan., p. 286.

$Cu + Fe_2(SO_4)_3 = CuSO_4 + 2FeSO_4$

$Pb + Fe_2(SO_4)_3 = PbSO_4 + 2FeSO_4$

$2As + 3H_2O + 3Fe_2(SO_4)_3 = As_2O_3 + 6FeSO_4 + 3H_2SO_4$

$2Sb + 3H_2O + 3Fe_2(SO_4)_3 = Sb_2O_3 + 6FeSO_4 + 3H_2SO_4$

$2Bi + 3H_2O + 3Fe_2(SO_4)_3 = Bi_2O_3 + 6FeSO_4 + 3H_2SO_4$

$2Ag + Fe_2(SO_4)_3 = Ag_2SO_4 + 2FeSO_4$

It will be noticed from the reactions that acid is set free in the oxidation of arsenic, antimony and bismuth. This acid will be neutralized in the same way with copper oxide just before the next treatment, in order that the process may be regenerative. Some silver is almost sure to dissolve, even if an insufficient quantity of ferric sulphate is used, and this is precipitated before filtration by hanging copper plates in the solution. Several hours are necessary to throw the silver out. Silver can be largely dissolved, before filtration, and later precipitated separately by copper, but the precipitate is very voluminous, and the silver contains antimony and arsenic oxides, so that it is found to be cheaper to leave the silver in the slime, although the opportunity is present to remove most of it.

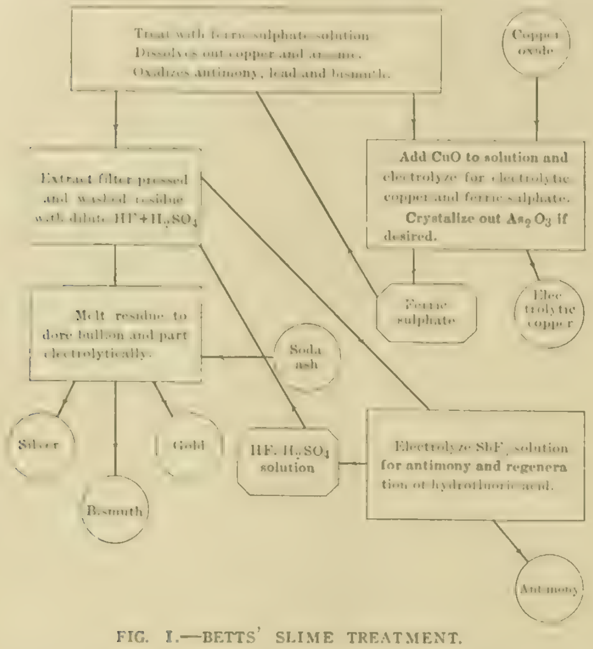
Treat with ferric sulphate solution
Dissolves out copper and arsenic.
Oxidizes antimony, lead and bismuth.
Copper oxide
Add CuO to solution and electrolyze for electrolytic copper and ferric sulphate.
Crystalize out As_2O_3 if desired.
Extract filter pressed and washed residue with dilute $HF + H_2SO_4$
Ferric sulphate
Electrolytic copper
Melt residue to doré bullion and part electrolytically.
Soda ash
Silver
Gold
Bismuth
$HF. H_2SO_4$ solution
Electrolyze SbF_3 solution for antimony and regeneration of hydrofluoric acid.
Antimony

FIG. I.—BETTS' SLIME TREATMENT.

For 100 pounds of electrolytic lead slime, containing perhaps 5 pounds of arsenic as a fair average, about 22 cubic feet of solution containing 4 per cent iron as ferric sulphate are used. The amount of arsenic after the first treatment rises to about .35 per cent.

The residue must be washed free from copper and iron solution, in order that the subsequent antimony extraction may give a pure deposit of antimony, which can be best done with a filter press. It does not pay to evaporate the wash waters, because ferrous sulphate is a very cheap salt, worth from 3 to 5 dollars per ton, and the wash water is accordingly run to waste over scrap iron to precipitate out copper. If 10 per cent of the solution is run to waste daily in this way, arsenic will eventually stand in the solution at from 1 to 3 per cent, which is not too high for the deposition of pure copper, under the conditions of temperature and current density used.

Arsenious acid is soluble in about ten parts of the hot solution and 100 parts of cold, so that we have here the means of saving the arsenic by crystallization from the solution, if it became necessary with special slimes, or if the arsenic should be worth saving.

The amount of bismuth and antimony dissolved is small, and they do not deposit with the copper. The reduced iron and copper sulphate solution, containing 3 to 4 per cent of copper, is then electrolyzed for copper and ferric sulphate, as described below.

The filter-press cakes of silver, lead sulphate, bismuth and antimony oxides and gold are then lixiviated in a lead-lined tank, similar to the one used for copper dissolving, with air agitation, in a solution of 2 to 4 per cent sulphuric acid, and 12 or 15 per cent hydrofluoric acid. The antimony dissolves in the course of 4 or 5 hours to the extent of some 90 per cent of the total. If the slime was previously well washed pure electrolytic antimony will be subsequently obtained.

The resulting insoluble portion, after filtration of the antimony on a cloth filter, can be dried and mixed with one-fourth to one-third of its weight of soda ash; that is, 7 to 10 pounds per ton of lead, and melted down to doré bullion, containing the bismuth of the slime. This is parted electrolytically, as described later in this article.

For copper slimes containing sulphur and little antimony it is preferable to extract the sulphur, antimony, selenium and tellurium by caustic soda solution, which reacts with the sulphur to form sodium sulphide, dissolving antimony as sulphantimonite.

Other methods of oxidation of the ferric sulphate were tried, as it was feared at first that the electrolysis of the copper iron sulphates for copper and ferric sulphate could not be practically carried out. For example, allowing the solution to trickle down cloths in the presence of air, crystallization of ferrous sulphate and calcination to basic ferric sulphate and resolution of the same were found to be far inferior to electrolysis.

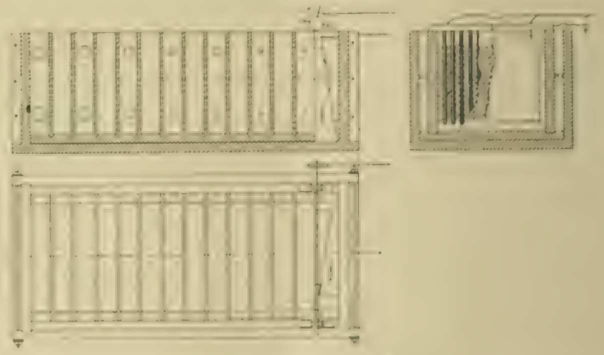

FIG. 2.—PLANT FOR DEPOSITING COPPER AND REGENERATING FERRIC SULPHATE.

Electrolytic Recovery of Copper and Ferric Sulphate.

After preliminary experiments a tank was constructed as shown in Fig. 2. The tank is intended to be lined with lead, but this was omitted in this tank, owing to the experimental nature of the work. The diaphragms are of cypress, five-eighth inch thick, perforated with five-eighth inch holes as closely as possible. The holes are closed with asbestos wads, put in wet, which do not drop out. The boards should be wetted before the asbestos wads are put in, otherwise the unequal expansion will split them. Arc-light carbons were used for anodes. One of the side channels in the tank contains the anolyte and the other the catholyte. Circulation is kept up by wooden paddle wheels, the solution flowing through the upper holes into the several compartments, and out again through the lower ones. The proper circulation is insured by boards placed longitudinally in the side channels, but not reaching quite to the ends beneath the paddle wheels. The sills and uprights of the distance frames are in separate pieces, and the whole is held together by wedges at one end. The spaces not closed by the wedges at one side are closed up with asbestos to prevent mixing of the anolyte and catholyte. The solution is fed to the cathode compartments, and siphoned off from the anode compartments, so that a slight difference in level exists between the anolyte and catholyte, which causes just sufficient flow through the asbestos.

The tank worked well for a few minutes, but soon polarization was set up, gassing occurred at the anodes, and they were soon covered with a slimy layer. Everything that was thought could help was tried, and finally I mounted the anodes on a frame resting on rollers, and gave them a reciprocating motion through the electrolyte. After the anodes were cleaned they remained clean, gassing stopped, the electromotive force dropped, and the efficiency went up to 100 per cent and remained there. The result was surprisingly good and even arc-light carbons will do for the work, but Acheson anodes are better. It is often stated that ordinary carbon anodes are good enough for electrolyses evolving chlorine at the anode, but that carbon will not stand in sulphate solutions. To this I do not agree, for the question is simply one of electromotive force, and not the particular nature of the anion liberated. The difference in voltage of a moving and stationary carbon electrode in ferrous sulphate is over one-half volt.

Deposition of Antimony.

Electrolysis of antimony trifluoride for antimony is carried out in a lead-lined tank, with sheet lead cathodes and lead rods as anodes, with a cathode current density of 10 to 20 amperes per square foot, and twice that at the anode, with an electromotive force of from 2.6 to 2.8 volts. There is a considerable tendency to form antimony pentafluoride at the anodes, which oxidized the cathode deposit. The current efficiency would be low were it not for a protecting layer of cotton cloth around each anode rod, by which means, and by the higher anode density, a current efficiency of 92 or 95 per cent is easily obtained. The antimony deposit is dense and solid, without projecting trees, and does not contain combined antimony salts, as some antimony deposits are said to do. Antimony contains about 1 per cent bismuth.

Parting.

Electrolytic parting is now carried on with a nitrate of silver bath, by either of two processes known as Moebius' and Balbach's. They are unquestionably good processes, but they have the limitation imposed by the lack of large quantity of free acid in the solution, or requiring more power than would otherwise be necessary, and are not suitable for handling anodes containing bismuth. It is also said that lead in the anodes goes into lead peroxide, which would not, however, appear probable from an electrochemical point of view. For parting, I use the silver salt of a non-oxidizing acid, with a large excess of the acid, the best results being obtained with methyl-sulphuric acid. The bismuth of the anodes easily goes into solution along the traces of antimony present, and the lead, copper and silver, and may accumulate to the extent of 4 or 5 per cent without precipitating basic salt. The silver deposit, particularly when 1 part of gum arabic or gelatine is present for every 12,000 to 15,000 parts of solution, is quite solid, and entirely free from projections. Electrolysis may be carried on for forty-eight hours easily without any danger of short-circuits from the growth of projections. It is found rather an advantage than otherwise that the silver deposit is not quite solid, for it can be scraped from the sheet-silver cathodes used, and they can be used over and over again, instead of melting them down with the deposit, as is done with copper and lead cathodes.

The copper and bismuth of the anodes accumulate in the solution at the expense of silver deposited on the cathodes. After silver is reduced to about 1.5 per cent, the solution should be replaced. For working up the solution and saving the bismuth, the remaining silver is first precipitated by metallic copper, and then copper and bismuth by metallic lead, and the lead methyl-sulphate decomposed by silver sulphate, giving lead sulphate and solution of silver methyl-sulphate to be used over again. The precipitate of copper and bismuth can be treated with ferric sulphate to remove the copper and the bismuth oxide can be melted to metal.

Cost of Treating Slime Electrolytically.

For a particular case, on the basis of treatment of 1 ton of lead bullion, containing antimony 1 per cent, silver 1 per cent, arsenic ¼ per cent, bismuth .2 per cent, copper ½ per cent, and 1 ounce of gold per ton, the cost is as follows:

Chemicals, as follows		$0.33
Ferrous sulphate, 5 per cent daily loss 24 pounds, at $5 per ton	6 cents	
Sulphuric acid for copper-iron solution, antimony solution, making silver sulphate, 9 pounds at 1 cent	9 "	
Soda ash, 10 pounds, at 1 cent	10 "	
Other chemicals, nitric acid for gold, losses in silver solution	5 "	
Hydrofluoric acid loss 1 pound, at 3 cents	3 "	
	33 "	
Power		
for depositing 20 pounds antimony	18 kw.-hrs.	
for depositing 52 pounds copper and making ferric sulphate	25 "	
Refining 20 pounds silver	1 kw.-hr.	
Total, at $30 per hp.-year of 365 days	44 kw.-hrs.	$0.20
2250 pounds steam for heating solution		0.05
20 pounds coal for melting antimony and silver, at $4		0.04
Labor, 10 men required for plant refining slime from 100-ton lead refinery		.25
Allow for supplies and repairs		.10
Total cost per ton of lead of analysis given		$0.97

The copper oxide used is not figured in as an element of cost, as copper in oxide is worth less than as electrolytic metal, so that a gain is made, rather than the cost increased.

This works out at a cost of about one and one-half cents per pound of slime, and when it is remembered that it costs copper refineries one cent per ounce, or more, to collect their silver—that is, with a slime carrying 50 per cent silver, seven cents per pound, or more, to treat it, besides throwing away the by-products and losing at least one per cent of silver, which loss may be apparently more than covered by the loss in commercial fire-assay—the superiority of the electrochemical method, working at less than one-fourth the cost, and saving everything of value, is apparent. The electrochemical process on a 100-pound lot of slime has given a return of silver equal to corrected fire-assay, or one and one-fourth per cent higher than commercial assay.

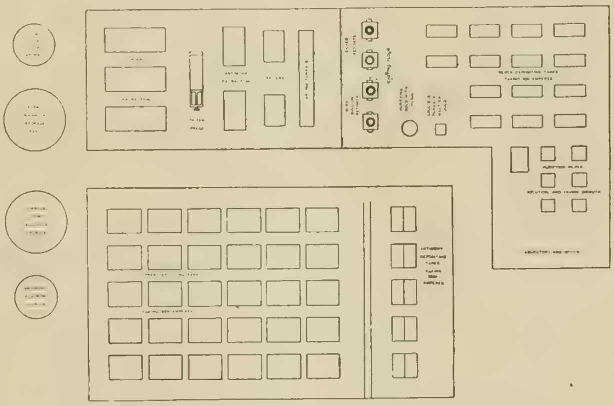

FIG. 3.—ELECTROLYTIC SLIME ARRANGEMENT OF PLANT

The general arrangement of the plant is shown in Fig. 3.

My thanks are due Dr. Edward F. Kern for much valuable assistance.

Metallurgical Calculations.—II.

By J. W. Richards, Ph. D.

Professor of Metallurgy in Lehigh University.

The Application of Thermochemical Principles.

The ordinary interpretation of the chemical equation by weight gives us the quantitative relations governing the reactions of substances upon each other, when the reaction proceeds to a finish. Unfortunately, much chemical instruction, as given in our elementary schools, and even in some of the higher ones, stops with the consideration of the weight relations and does not proceed to those equally important relations, the energetics of chemical reactions. In most of the reactions which occur in practical metallurgy, the quantity of the combustible used, or, more broadly, the amount of energy in the form of heat or electrical energy, necessary for producing the reactions desired, is the controlling factor regulating the practicability or impracticability, the commercial success or failure, of the process.

The relative values of fuels, the manufacture and utilization of gas, the principles of the regenerative furnace, the Bessemer process, electric reduction, and a host of metallurgical processes, depend essentially on the realization and utilization of chemical energy, and the only way to become conversant with the amounts of energy involved or evolved in these operations is to understand thoroughly the thermochemistry of the reactions concerned.

Thermochemical Nomenclature.

The heat evolved when compounds are formed from the elements (in a few cases heat is absorbed) is determined experimentally by the use of the calorimeter. This branch is sometimes called "chemical calorimetry;" it is practically a department of experimental physics. The data obtained give the heat evolved for the total change from the components at room temperature to the resulting products, at room temperature (or very near to it), and may be expressed per unit of weight of either component or of the substance formed. Thus, if carbon is burned in a calorimeter to carbonic acid gas, CO^2, the heat evolved may be reported or expressed as

8100 gram calories per gram of carbon burnt.

Or, 3037 gram calories per gram of oxygen used.

Or, 2209 gram calories per gram of CO^2 formed.

Of these three methods of expressing the results, the first is the more often used, especially by the physicist.

The chemist, however, finds it often more convenient and logical to express these heats of combination per formula weight of the substances combining and of product formed. *E. g.*: In the case of CO^2, which contains 12 parts of carbon and 32 of oxygen in 44 of the gas, the chemist would write,

$$(C, O^2) = 97.200,$$

meaning thereby that when 12 grams of carbon is burnt by 32 grams of oxygen, forming 44 grams of carbon dioxide, there is evolved 97,200 gram-calories. These are the laboratory units; for practical purposes, we call the weights kilograms and the heat units kilogram-calories (gram-calories are abbreviated to "cal."; kilogram-calories to "Cal."). Ostwald, in his thermochemical tables writes $(C, O^2) = 972$ K, where K represents a unit 100 times as large as a gram-calorie, if weights are taken as being grams. Berthelot writes $(C, O^2) = 97.2$, where the heat units are kilogram-calories, if the weights concerned are taken as grams. Both these methods of expression are liable to cause confusion; the writer prefers to follow the older thermochemists (Hess, Naumann) and to use the larger number, *e. g.*, 97,200, which then means gram-calories, if weights are taken in grams (laboratory units), and kilogram-calories, if weights are taken in kilograms (practical units).

If it is desired to work in "British Thermal Units" (1 B. T. U. is the heat needed to raise one pound of water one degree Fahrenheit), the weights represented by the formula may be called pounds, and then the expression is as follows:

$$(C, O^2) = [97,200 \times 9/5] = 174,960 \text{ B. T. U.}$$

The factor 9/5 is simply the relation of 1° C. to 1° F.; the reasoning is, of course, that if the combustion of 12 kilograms of carbon evolves 97,200 kilogram calories, or would heat 97,200 kilograms of water 1° C., that the combustion of 12 pounds of carbon would heat 97,200 pounds of water 1° C., or 174,960 pounds 1° F. From the equation as thus expressed and interpreted, the heat of combination per pound of carbon burnt, or of oxygen used, or of product formed, may be found in B. T. U. by dividing 174,960 by 12, 16 or 44, respectively.

Since it is very inconvenient, as well as unscientific, to have the two unrelated heat units, with their resulting double sets of experimental data, I strongly recommend the use of the metric data and metric-Centigrade heat unit. It is, however, sometimes convenient, when all the data of a problem are given in English weights, to use as the unit of heat the "pound — 1° C.," or the heat required to raise the temperature of one pound of water 1° C. This may be called the "pound cal.," as distinguished from the B. T. U. The advantage of using it is that all the experimental data of the metric system units are at once transferable to the English weights. *E. g.*, $(C, O^2) = 97,200$ pound cal., if the weights concerned in the formula (12, 32, 44) are called pounds.

The thermochemist gives all his experimental data in the form above explained, and we will now give all the important thermochemical data known which are useful in metallurgical calculations, *the data being for the reactions beginning and ending at* 15° C. (60° F.):

Thermochemical Data.

Heat of Formation of Oxides.

Formula.	*Molecular Weights.*	*Molecular Heat of Formation.*	*In dilute Solution.*
(Mg, O)	24 + 16 = 40	143,400	148,800
(Ba, O)	137 + 16 = 153	133,400	161,500
(Ca, O)	40 + 16 = 56	131,500	149,600
(Sr, O)	87 + 16 = 103	131,200	158,400
(Al^2, O^3)	54 + 48 = 102	392,600	......
(Na^2, O)	46 + 16 = 62	100,900	155,900
(K^2, O)	78 + 16 = 94	98,200	165,200
(Si, O^2)	28 + 32 = 60	180,000	......
(Mn, O)	55 + 16 = 71	90,900	......
(B^2, O^3)	22 + 48 = 70	272,600	279,900
(Zn, O)	65 + 16 = 81	84,800	......
(Mn^3, O^4)	165 + 64 = 229	328,000	......
(P^2, O^5)	62 + 80 = 142	365,300	......
(Sn, O)	118 + 16 = 134	70,700	......
(Sn, O^2)	118 + 32 = 150	141,300	......
(CO, O)	28 + 16 = 44	68,040	73,940
(H^2, O)	2 + 16 = 18	70,400 (solid) 69,000 (liquid) 58,060 (gas)	....
(Fe^3, O^4)	168 + 64 = 232	270,800	......
(Cd, O)	112 + 16 = 128	66,300	......
(Fe, O)	56 + 16 = 72	65,700	......
(Fe^2, O^3)	112 + 48 = 160	195,600	......
(Co, O)	59 + 16 = 75	64,100	......
(Mn, O^2)	55 + 32 = 87	125,300	......
(Ni, O)	58.5 + 16 = 74.5	61,500	......
(Sb^2, O^3)	240 + 48 = 288	166,900	......
(As^2, O^3)	150 + 48 = 198	156,400	148,900
(Pb, O)	207 + 16 = 223	50,800	......
(C, O^2)	12 + 32 = 44	97,200 (gas)	103,100
(Bi^2, O^3)	416 + 48 = 464	139,200	......
(Sb^2, O^5)	240 + 80 = 320	231,200	......
(As^2, O^5)	150 + 80 = 230	219,400	225,400
(Cu^2, O)	127.2 + 16 = 143.2	43,800	......

Formula.	*Molecular Weights.*	*Molecular Heat of Formation.*	*In dilute Solution.*
(Tl^2, O)	408 + 16 = 424	42,800	39,700
(Cu, O)	63.6 + 16 = 79.6	37,700	......
(Ba, O^2)	137 + 32 = 169	145,500	......
(S, O^2)	32 + 32 = 64	69,260 (gas)	77,600
(Pb, O^2)	207 + 32 = 239	63,400	......
$(S. O^3)$	32 + 48 = 80	91,900	141,000
(Tl^2, O^3)	408 + 48 = 456	87,600	......
(C, O)	12 + 16 = 28	29,160 (gas)	......
(Hg^2, O)	400 + 16 = 416	22,200	......
(Hg, O)	200 + 16 = 216	21,500	......
(Te, O^2)	125.5 + 32 = 157.5	......	78,300
(Pd, O)	106 + 16 = 122	21,000	......
(Pt, O)	195 + 16 = 211	17,000 (?)	......
(Ag^2, O)	216 + 16 = 232	7,000	......
(Au^2, O^3)	394 + 48 = 442	—11,500	......

HEAT OF FORMATION OF HYDRATES.

(Li, O, H)	7 + 16 + 1 = 24	112,300	118,100
(Mg, O^2, H^2)	24 + 32 + 2 = 58	217,800	......
(Sr, O^2, H^2)	87 + 32 + 2 = 121	217,300	227,400
(Ca, O^2, H^2)	40 + 32 + 2 = 74	215,600	219,500
(K, O, H)	39 + 16 + 1 = 56	104,600	117,100
(Na, O, H)	23 + 16 + 1 = 40	102,700	112,500
(N, H^5, O)	14 + 5 + 16 = 35	88,800	90,000
(Al^2, O^3, H^3)	27 + 48 + 3 = 78	301,300	......
(H, O, H)	1 + 16 + 1 = 18	70,400 (solid) 69,000 (liquid) 58,060 (gas)	....
(Tl, O, H)	204 + 16 + 1 = 221	57,400	54,300
(Bi, O^3, H^3)	208 + 48 + 3 = 259	171,700	......
(Zn, O^2, H^2)	65 + 32 + 2 = 99	83,500	......
(Te, O^2, H^2)	127 + 32 + 2 = 161	78,300	......
(Te, O^3, H^3)	127 + 48 + 3 = 178	......	99,500
(Se, O^2, H^2)	79 + 32 + 2 = 113	52,400	51,500
(Se, O^3, H^3)	79 + 48 + 3 = 130	......	79,300
(Tl, O^3, H^3)	204 + 48 + 3 = 255	43,800	......

HEAT OF FORMATION OF SULPHIDES.

(Li^2, S)	14 + 32 = 46	......	115,400
(K^2, S)	78 + 32 = 110	103,500	113,500
(Ba, S)	137 + 32 = 169	102,900	109,800
(Sr, S)	87 + 32 = 119	99,300	106,700
(Ca, S)	40 + 32 = 72	94,300	100,600
(Na^2, S)	46 + 32 = 78	89,300	104,300
(Mg, S)	24 + 32 = 56	79,400	......
(K, S^2)	39 + 64 = 103	59,300	59,700
(Na, S^2)	23 + 64 = 87	49,500	54,400
(Mn, S)	55 + 32 = 87	45,600	......
(Zn, S)	65 + 32 = 97	43,000	......
(Al^2, S^3)	54 + 96 = 150	126,400	......
(N, H^5, S)	14 + 5 + 32 = 51	40,000	36,700
(Cd, S)	112 + 32 = 144	34,400	......
(B^2, S^3)	22 + 96 = 118	75,800	......
(Fe, S)	56 + 32 = 88	24,000	......
(Co, S)	59 + 32 = 91	21,900	......
(Tl^2, S)	204 + 32 = 236	21,600	......
(Cu^2, S)	127.2 + 32 = 159.2	20,300	......
(Pb, S)	207 + 32 = 239	20,200	......
(Si, S^2)	28 + 64 = 92	40,000	......
(Ni, S)	58.5 + 32 = 90.5	19,500	......
(Sb^2, S^3)	240 + 96 = 336	34,400	......
(Hg, S)	200 + 32 = 232	10,600	......
(Cu, S)	63.6 + 32 = 95.6	10,100	......
(H^2, S)	2 + 32 = 34	4,800 (gas)	9,500
$(Ag^2.S)$	216 + 32 = 248	3,000	......
(C, S^2)	12 + 64 = 76	—25,400 (gas) —19,000 (liquid)	....
(I, S)	127 + 32 = 159	0,000	......

HEAT OF FORMATION OF SELENIDES.

Formula.	*Molecular Weights.*	*Molecular Heat of Formation.*	*In dilute Solution.*
(Li^2, Se)	14 + 79 = 93	83,000	93,700
(K^2, Se)	78 + 79 = 157	79,600	87,900
(Ba, Se)	137 + 79 = 216	69,900	......
(Sr, Se)	87 + 79 = 166	67,600	......
(Na^2, Se)	46 + 79 = 125	60,900	78,600
(Ca, Se)	40 + 79 = 119	58,000	......
(Zn, Se)	65 + 79 = 144	30,300	......
(Cd, Se)	112 + 79 = 191	23,700	......
(Mn, Se)	55 + 79 = 134	22,400	......
(N, H^5, Se)	14+5 + 79 = 98	17,800	12,800
(Cu, Se)	63.6 + 79 = 142.6	17,300	......
(Pb, Se)	207 + 79 = 286	17,000	......
(Fe, Se)	56 + 79 = 135	15,200	......
(Ni, Se)	58.5 + 79 = 137.5	14,700	......
(Co, Se)	59 + 79 = 138	13,900	......
(Tl^2, Se)	408 + 79 = 487	13,400	......
(Cu^2, Se)	127.2 + 79 = 206.2	8,000	......
(Hg, Se)	200 + 79 = 279	6,300	......
(Ag^2, Se)	216 + 79 = 295	2,000	......
(H^2, Se)	2 + 79 = 81	—25,100 (gas)	—15,800
(N, Se)	14 + 79 = 93	—42,300	......

HEAT OF FORMATION OF TELLURIDES.

(Zn, Te)	65 + 126 = 191	31,000	......
(Cd, Te)	112 + 126 = 138	16,600	......
(Co, Te)	59 + 126 = 185	13,000	......
(Fe, Te)	56 + 126 = 182	12,000	......
(Ni, Te)	58.5 + 126 = 184.5	11,600	......
(Tl^2, Te)	408 + 126 = 534	10,600	......
(Cu^2, Te)	127.2 + 126 = 253.2	8,200	......
(Pb, Te)	207 + 126 = 333	6,200	......
(H^2, Te)	2 + 126 = 128	—34,900 (gas)	......

HEAT OF FORMATION OF ARSENIDES.

(H^3, As)	3 + 75 = 78	—44,200 (gas)	......

HEAT OF FORMATION OF ANTIMONIDES.

(H^3, Sb)	3 + 120 = 123	—86,800 (gas)	......

HEAT OF FORMATION OF PHOSPHIDES.

(Mn^3, P^2)	165 + 62 = 227	70,900	......
(H^3, P)	3 + 31 = 34	4,900 (gas)	......
(Fe, P)	56 + 39 = 95	nearly zero.	......

HEAT OF FORMATION OF NITRIDES.

(H^3, N)	3 + 14 = 17	12,200 (gas) 16,600 (liquid)	21,000 ......
(Ba^3, N^2)	411 + 28 = 439	149,400	......
(Li^3, N)	21 + 14 = 35	49,500	......
(K, H^3, N)	39 + 3 + 14 = 56	30,700	......

HEAT OF FORMATION OF METALLIC HYDRIDES.

(Sr, H^2)	87 + 2 = 89	38,400	......
(Ba, H^2)	137 + 2 = 139	37,500	......
(Pt^{10}, H)	1,950 + 1 = 1,951	14,200	......
(Pd^{15}, H)	1,590 + 1 = 1,591	4,600	......
(Si, H^4)	28 + 4 = 32	—6,700 (gas)	......

HEAT OF FORMATION OF CARBIDES.

(Al^4, C^3)	108 + 36 = 144	232,000	......
(Mn, C^2)	55 + 24 = 79	114,400 (Ponthiere)	..
(Mn, C^3)	55 + 36 = 91	9,900 (Berthelot)	..
(Fe^3, C)	168 + 12 = 180	8,460	......
(Ca, C^2)	40 + 24 = 64	—6,250	......
(Na, C)	23 + 12 = 35	—4,400	......
(Li, C)	7 + 12 = 19	—5,750	......
(N^2, C^2)	28 + 24 = 52	—73,000 (gas)	—67,100
(Ag, C)	108 + 12 = 120	—43,575	......

HEAT OF FORMATION OF SILICIDES.

Formula.	*Molecular Weights.*	*Molecular Heat of Formation.*	*In dilute Solution.*
(Mn^7, Si^2)	385 + 56 = 441	47,400	. . . . .
(H^4, Si)	0 + 28 = 32	— 6,700 (gas)	. . . .

HEAT OF FORMATION OF FLUORIDES.

Formula.	*Molecular Weights.*	*Molecular Heat of Formation.*	*In dilute Solution.*
(Sr, F^2)	87 + 38 = 125	224,020	. . . . .
(Ba, F^2)	137 + 38 = 175	224,000	221,500
(Li, F)	7 + 19 = 26	. . . . .	116,880
(K, F)	39 + 19 = 58	110,000	113,600
(Ca, F^2)	40 + 38 = 78	216,450	. . . . . .
(Mg, F^2)	24 + 38 = 62	200,500	. . . . . .
(Na, F)	23 + 19 = 42	109,720	109,120
(N, H^4, F)	14 + 4 + 19 = 37	101,250	99,750
(Al, F^3)	27 + 57 = 84	. . . . . .	275,220
(B, F^3)	11 + 57 = 68	. . . . . .	219,345
(Mn, F^2)	55 + 38 = 93	. . . . . .	153,310
(Zn, F^2)	65 + 38 = 103	. . . . . .	138,220
(Si, F^4)	28 + 76 = 104	275,920 (gas)	. . . . . .
(Fe, F^2)	56 + 38 = 94	. . . . . .	125,220
(Cd, F^2)	112 + 38 = 150	. . . . .	121,720
(Co, F^2)	59 + 38 = 97	. . . . .	120,340
(Ni, F^2)	58.5 + 38 = 96.5	. . . . . .	118,980
(Fe, F^3)	56 + 57 = 113	. . . . . .	164,940
(Tl, F)	204 + 19 = 223	. . . . . .	54,405
(Pb, F^2)	207 + 38 = 245	101,600	. . . . . .
(H, F)	1 + 19 = 20	38,500 (gas)	50,300
(Sb, F^3)	120 + 57 = 177	. . . . . .	136,680
(Cu, F^2)	63.6 + 38 = 101.6	. . . . . .	88,160
(Ag, F)	108 + 19 = 127	22,070	25,470

HEAT OF FORMATION OF CHLORIDES.

Formula.	*Molecular Weights.*	*Molecular Heat of Formation.*	*In dilute Solution.*
(K, Cl)	39 + 35.5 = 74.5	105,700	101,200
(Ba, Cl^2)	137 + 70 = 207	197,100	198,300
(Na, Cl)	23 + 35.5 = 58.5	97,900	96,600
(Li, Cl)	7 + 35.5 = 42.5	93,900	102,300
(Sr, Cl^2)	87 + 71 = 158	184,700	195,850
(Ca, Cl^2)	40 + 71 = 111	169,900	187,400
(N, H^4, Cl)	14 + 4 + 35.5 = 53.5	76,800	72,800
(Mg, Cl^2)	24 + 71 = 95	151,200	187,100
(Al, Cl^3)	27 + 106.5 = 133.5	161,800	238,100
(Mn, Cl^2)	55 + 71 = 126	112,000	128,000
(Zn, Cl^2)	65 + 71 = 136	97,400	113,000
(Tl, Cl)	204 + 35.5 = 239.5	48,600	38,400
(Cd, Cl^2)	112 + 71 = 183	93,700	96,400
(Pb, Cl^2)	207 + 71 = 278	83,900	77,900
(Fe, Cl^2)	56 + 71 = 127	82,200	100,100
(Sn, Cl^2)	118 + 71 = 189	80,900	. . . . . .
(Co, Cl^2)	59 + 71 = 130	76,700	95,000
(Ni, Cl^2)	58.5 + 71 = 129.5	74,700	93,900
(Cu, Cl)	63.5 + 35.5 = 99	35,400	. . . . . .
(Sn, Cl^4)	118 + 142 = 260	129,800 (liquid)	. . . . . .
(Fe, Cl^3)	56 + 106.5 = 162.5	96,150	127,850
(Hg, Cl)	200 + 35.5 = 235.5	31,320	. . . . . .
(Sb, Cl^3)	120 + 106.5 = 226.5	91,400	. . . . . .
(Bi, Cl^3)	208 + 106.5 = 314.5	90,800	. . . . . .
(B, Cl^3)	11 + 106.5 = 117.5	89,100 (gas)	. . . . . .
(Ag, Cl)	108 + 35.5 = 143.5	29,000	. . . . . .
(Hg, Cl^2)	200 + 71 = 271	53,300	50,300
(Cu, Cl^2)	63.6 + 71 = 134.6	51,400	62,500
(As, Cl^3)	75 + 106.5 = 181.5	71,500	. . . . . .
(H, Cl)	1 + 35.5 = 36.5	22,000	39,400
(Sb, Cl^5)	120 + 177.5 = 297.5	104,500 (liquid)	. . . . . .
(Pd, Cl^2)	106 + 71 = 177	40,500	. . . . . .
(Pt, Cl^4)	195 + 142 = 337	60,200	79,800
(Au, Cl^3)	197 + 106.5 = 303.5	22,800	27,200
(Au, Cl)	197 + 35.5 = 232.5	5,800	. . . . . .

HEAT OF FORMATION OF HYDROCARBONS.

(All formed in state of gas, unless otherwise specified.)

Name.	*Formula.*	*Molecular Weights.*	*Molecular Heat of Formation.*
Methane (marsh gas)	(C, H^4)	12 + 4 = 16	22,250
Ethane (ethylene hydride)	(C^2, H^6)	24 + 6 = 30	26,650
Propane (propylene hydride)	(C^3, H^8)	36 + 8 = 44	33,850
Ethylene (olefiant gas)	(C^2, H^4)	24 + 4 = 28	— 11,250
Propylene	(C^3, H^6)	36 + 6 = 42	— 6,050
Toluene	(C^7, H^8)	84 + 8 = 92	5,650 (liquid)
Benzene	(C^6, H^6)	72 + 6 = 78	— 750 (liquid) — 7,950 (gas)
Turpentine	(C^{10}, H^{16})	120 + 16 = 136	7,550 (liquid) — 1,850 (gas)
Naphthaline	(C^{10}, H^8)	120 + 8 = 128	— 19,450 (solid) — 24,050 (liquid)
Anthracene	(C^{14}, H^{10})	168 + 10 = 178	— 39,050 (solid)
Acetylene	(C^2, H^2)	24 + 2 = 26	— 54,750
Methyl-alcohol (wood spirit)	(C, H^4, O)	12 + 4 + 16 = 32	65,050 (liquid) 56,050 (gas)
Ethyl-alcohol (alcohol)	(C^2, H^6, O)	24 + 6 + 16 = 46	73,250 (liquid) 63,150 (gas)
Acetone	(C^3, H^6, O)	36 + 6 + 16 = 58	69,650 (liquid) 62,150 (gas)

HEAT OF FORMATION OF CARBONATES.

Formula.	*Molecular Weights.*	*Molecular Heat of Formation.*	*In dilute Solution.*
(Ba, C, O^3)	137 + 12 + 48 = 197	286,300	. . . . . .
(K^2, C, O^3)	78 + 12 + 48 = 138	282,100	288,600
(Sr, C, O^3)	87 + 12 + 48 = 147	281,400	. . . . . .
(Ca, C, O^3)	40 + 12 + 48 = 100	273,850	. . . . . .
(Na^2, C, O^3)	46 + 12 + 48 = 106	273,700	279,300
(Mg, C, O^3)	24 + 12 + 48 = 88	269,900	. . . . . .
(Mn, C, O^3)	55 + 12 + 48 = 115	210,300	. . . . . .
(N, H^5, C, O^3)	14 + 5 + 12 + 48 = 79	208,600	202,300
(Zn, C, O^3)	65 + 12 + 48 = 125	197,500	. . . . . .
(Fe, C, O^3)	56 + 12 + 48 = 116	187,800	. . . . . .
(Cd, C, O^3)	112 + 12 + 48 = 172	183,200	. . . . . .
(Pb, C, O^3)	207 + 12 + 48 = 267	170,000	. . . . . .
(Cu, C, O^3)	63.6 + 12 + 48 = 123.6	146,100	. . . . . .
(Ag^2, C, O^3)	216 + 12 + 48 = 276	123,800	. . . . . .

Heat of Formation of Bicarbonates.

Formula.	Molecular Weights.	Molecular Heat of Formation.	In dilute Solution.
(K, H, C, O^3)	39 + 1 + 12 + 48 = 100	233,300	228,000
(Na, H, C, O^3)	23 + 1 + 12 + 48 = 84	227,000	222,700

Heat of Formation of Nitrates.

Formula.	Molecular Weights.	Molecular Heat of Formation.	In dilute Solution.
(K, N, O^3)	39 + 14 + 48 = 101	119,000	110,700
(Na, N, O^3)	23 + 14 + 48 = 85	110,700	106,000
(Zn, N^2, O^6)	65 + 28 + 96 = 187	......	131,700
(Pb, N^2, O^6)	207 + 28 + 96 = 331	105,400	98,200
(Cu, N^2, O^6)	63.5 + 28 + 96 = 187.5	......	81,300
(H, N, O^3)	1 + 14 + 48 = 63	34,400 gas	48,800
(Ag, N, O^3)	108 + 14 + 48 = 170	28,700	23,000

Heat of Formation of Phosphates.

Formula.	Molecular Weights.	Molecular Heat of Formation.	In dilute Solution.
(Ca^3, P^2, O^8)	120 + 62 + 128 = 310	919,200	......
(Mg^3, P^2, O^8)	72 + 62 + 128 = 262	910,600	......
(Na^3, P, O^4)	69 + 31 + 64 = 164	452,400	......

Heat of Formation of Silicates.

Formula.	Molecular Weights.	Molecular Heat of Formation.	In dilute Solution.
(Ca, Si, O^3)	40 + 28 + 48 = 116	344,400	......
(Mn, Si, O^3)	55 + 28 + 48 = 131	276,300	......
(Fe, Si, O^3)	56 + 28 + 48 = 130	254,600	......

Heat of Formation of Sulphates.

Formula.	Molecular Weights.	Molecular Heat of Formation.	In dilute Solution.
(K^2, S, O^4)	78 + 32 + 64 = 174	344,300	337,700
(Ba, S, O^4)	137 + 32 + 64 = 233	339,400	......
(Li^2, S, O^4)	14 + 32 + 64 = 110	333,500	339,600
(Sr, S, O^4)	87 + 32 + 64 = 183	330,200	......
(Na^2, S, O^4)	46 + 32 + 64 = 142	328,100	328,500
(Ca, S, O^4)	40 + 32 + 64 = 136	317,400	321,800
(Mg, S, O^4)	24 + 32 + 64 = 120	300,900	321,100
(Al^2, S^3, O^{12})	54 + 96 + 192 = 342	......	879,700
(N^2, H^8, S, O^4)	28 + 8 + 32 + 64 = 132	283,500	281,100
(Mn, S, O^4)	55 + 32 + 64 = 151	249,400	263,200
(Zn, S, O^4)	65 + 32 + 64 = 161	229,600	248,000
(Fe, S, O^4)	56 + 32 + 64 = 152	......	234,900
(Co, S, O^4)	59 + 32 + 64 = 155	......	228,900
(Ni, S, O^4)	58.5 + 32 + 64 = 154.5	......	228,700
(Fe^2, S^3, O^{12})	112 + 96 + 192 = 400	......	650,500
(Tl^2, S, O^4)	408 + 32 + 64 = 504	221,800	213,500
(Cd, S, O^4)	112 + 32 + 64 = 208	219,900	231,600
(Pb, S, O^4)	207 + 32 + 64 = 303	215,700	......
(H^2, S, O^4)	2 + 32 + 64 = 98	192,200	210,200
(Cu, S, O^4)	63.6 + 32 + 64 = 159.6	181,700	197,500
(Hg^2, S, O^4)	400 + 32 + 64 = 496	175,000	......
(Ag^2, S, O^4)	216 + 32 + 64 = 312	167,100	162,600
(Hg, S, O^4)	200 + 32 + 64 = 296	165,100	......

Heat of Formation of Bi-Sulphates.

Formula.	Molecular Weights.	Molecular Heat of Formation.	In dilute Solution.
(K, H, S, O^4)	39 + 1 + 32 + 64 = 136	276,100	272,900
(Na, H, S, O^4)	23 + 1 + 32 + 64 = 120	269,100	268,300
(N, H^5, S, O^4)	14 + 5 + 32 + 64 = 115	244,600	245,100
(H, H, S, O^4)	1 + 1 + 32 + 64 = 98	192,200	210,200

Heat of Formation of Borates.

Formula.	Molecular Weights.	Molecular Heat of Formation.	In dilute Solution.
(Na^2, B^4, O^7)	46 + 44 + 112 = 202	748,100	758,300

Heat of Formation of Cyanides.

Formula.	Molecular Weights.	Heat of Formation.	In dilute Solution.
(Ca, C^2, N^2)	40 + 24 + 28 = 92	......	41,650
(K, C, N)	39 + 12 + 14 = 65	33,450	30,250
(Na, C, N)	23 + 12 + 14 = 49	25,950	25,450
(K, Ag, C^2, N^2)	39 + 108 + 24 + 28 = 199	13,700	5,350
(Fe^7, C^{18}, N^{18})	392 + 216 + 252 = 860	—256,700	......
(Zn, C^2, N^2)	65 + 24 + 28 = 117	—24,550	......
(Cd, C^2, N^2)	112 + 24 + 28 = 164	—31,850	......
(Cu, C, N)	63.6 + 12 + 14 = 89.6	—20,375	......
(Pd, C^2, N^2)	106 + 24 + 28 = 158	—49,250	......
(H, C, N)	1 + 12 + 14 = 27	—27,150 gas	—21,050
(Hg, C^2, N^2)	200 + 24 + 28 = 252	—59,150	......

Heat of Formation of Cyanates.

Formula.	Molecular Weights.	Molecular Heat of Formation.	In dilute Solution.
(K, C, N, O)	39 + 12 + 14 + 16 = 81	105,850	100,650
(Na, C, N, O)	23 + 12 + 14 + 16 = 65	105,050	100,250
(Ag, C, N, O)	108 + 12 + 14 + 16 = 150	26,450	......

Heat of Formation of Ferrocyanides.

Formula.	Molecular Weights.	Molecular Heat of Formation.	In dilute Solution.
(K^4, Fe, C^6, N^6)	156 + 56 + 72 + 84 = 368	157,300	145,300
(H^4, Fe, C^6, N^6)	4 + 56 + 72 + 84 = 216	—102,000	—101,500
(K^3, Fe, C^6, N^6)	117 + 56 + 72 + 84 = 329	129,600	100,800
(H^3, Fe, C^6, N^6)	3 + 56 + 72 + 84 = 215	......	—127,400

Heat of Formation of Amalgams.

Formula.	Molecular Weights.	Molecular Heat of Formation.	In dilute Mercury Solution.
(Hg^{12}, K)	2,400 + 39 = 2,439	34,600	25,600
(Hg^4, K)	800 + 39 = 839	29,700	25,600
(Hg^6, Na)	1,200 + 23 = 1,223	21,900	19,000
(Hg^x, Au)	x + 197 = 197 + x	....	2,580
(Hg^x, Ag)	x + 108 = 108 + x	....	2,470

Heat of Formation of Alloys.

Formula.	Molecular Weights.	Molecular Heat of Formation.
(Cu, Zn^2)	63.6 + 130 = 193.6	10,143
(Cu, Zn)	63.6 + 65 = 128.6	5,783
(Cu^3, Al)	190.8 + 27 = 217.8	26,910
(Cu^2, Al)	127.2 + 27 = 154,2	21,278
(Cu^3, Al^2)	190.8 + 54 = 244.8	17,395
(Cu, Al)	63.6 + 27 = 90.6	1,887
(Cu^2, Al^3)	127.2 + 81 = 208.2	10,196
(Cu, Al^2)	63.6 + 54 = 117.6	—6,738

[On account of the amount of space already taken up, the discussion of the manner of using these data in metallurgical calculations must be deferred until next month.]

Errata.

In last number, page 104, insert atomic weight of Cadmium, Cd, 112.

On page 106, problem 1, an arithemetical mistake in requirement (2) gives moisture as 0.06, instead of 0.006, and SO^2 as 0.15, instead of 0.015. On page 107, problem 3, under oxygen needed; that for Fe should be 142.8 kilos.

The arithemtical corrections can be easily entered, if anyone desires to correct them. The principles of the solutions in both cases are correct.

Cast-Iron Car Wheels.—A very interesting and fully illustrated article on the manufacture of cast-iron car wheels at the new plant of the Pennsylvania Railroad at South Altoona, Pa., was given in *Iron Age* of January 5. This is the largest and most modern plant of its kind in existence, and is designed to have a capacity of 900 wheels per day, and is a splendid example of the extended use of electric power in modern iron and steel plants. There is an uninterrupted progress of the metal from the cupolas to the annealing pits, each movement being noticeably timed to a nicety, due to the perfect system of handling facilities. The overhead system of application of electric power comprises numerous electric traveling hoists and a special type of electric traveling crane. The tramway system is used chiefly to handle the ladles between the electric hoists and the cupolas, and again to take the hot wheels from the electric hoists to the annealing compartment. Thus, the railways carry the molten metal to the molding floors and carry the completed wheel away, while the electric hoists perform their function during the actual work of casting and molding. At this plant the ever increasing demand of the Pennsylvania Railroad Co. for wheels is to be met by the company itself.

Electric Smelting and Blast Furnace Gases.

By Auguste J. Rossi.

We have had occasion in these columns to treat briefly of the possibilities of electric smelting, as compared to the ordinary metallurgical methods, particularly in reference to the electrical manufacture of steel by what is called the "pig and ore," or the "mixed process:" The direct reduction of oxide of iron for pig iron and steel, electrically, is a subject which has given rise to the most extreme opinions, because we believe it has not always been considered in its proper light.

The economy of such a process does not, indeed, solely depend on the greater or less perfection of the apparatus, though obviously this is as factor of which the importance cannot be ignored, but also on other conditions, which make that, however well combined in all its details of construction the electric furnace may be; however perfect in securing the maximum possible yield from a given charge; however cheaply even the electrical energy may be secured, there exist other objections which limit the application of electricity for such purpose.

It has been said that, in order to obtain the reduction of oxide of iron, there is required a certain amount of energy or its equivalent in heat calories, and that, in the present state of our modern blast furnace development the fuel consumed per ton of iron represents in ordinary conditions of trade, an outlay much less than that at which the electric hp., which corresponds to this amount of heat, could be secured in the most favorable conditions of localities. One metric ton of iron per metric ton of coke is now a current average practice in blast furnaces, and even better results, such as 1850 pounds, and even less, have been obtained per gross ton of metal.

Calculations which can be readily made, and which have appeared at different times in these columns in one form or the other, show indeed that, for an iron ore containing some 57 per cent metallic iron, such ore as is daily smelted in the Pittsburg district, the diverse items which constitute the amount of heat necessary to obtain 1 ton (gross ton) of pig iron (such as reduction of the oxides, melting of slag and product, etc.), counting 1 hp. = 2545 thermal units, or 650 calories (figures adopted by some), there is required at least 140 to 145 hp. If we assume an efficiency of 80 per cent to 75 per cent, this represents some 200 hp per ton, with a margin of 10 per cent, more or less, according to conditions of ores and furnaces. This is a figure which we have stated as the result of several weeks' run in conditions not particularly favorable; the materials used, the metal obtained, the current supplied, being all *measured* and weighed; it is in fact a practical figure which includes all losses.

Ten dollars a year per electric hp. may be considered as an acceptable figure obtainable now; this represents per day an expense of 2¾ cents, and for 200 hp. $5.50, while the price of coke at furnace per ton of iron may be taken at $2.50 to $2.75 in ordinary conditions (it is $2.75 to $3.00 now). This is undeniable, but this assumes that the price of ores, fluxes, fuel, labor maintenance, general expenses, of all these elements of cost, in fact, are the same in both cases; that the electric furnace is to be established side by side with, or in proximity to, the blast furnace. But conditions can be conceived and realized in which, for instance, in certain localities, the ores and fluxes could be had at almost the cost of mining without charges for transportation, so to speak, when the electric horse-power itself could be secured or made available at its lowest figure, all elements which would go to offset the extra cost of the heat-supplying medium. We should mention, also, that the carbon required in an electric furnace in the shape of fuel, does represent but about one-third the quantity required in a blast furnace. In short, calculations made on a given ore, in given conditions of localities, could show in favorable circumstances a parity and even an economy in favor of the electric smelting.

But let this be so; let this be demonstrated, as it is possible to do it in individual cases, the amount of electrical energy required per ton of metal excludes all possibilities other than those of a local limited production.

An output of 500 tons daily, a figure exceeded even by some furnaces in the Pittsburg district, corresponds, at 200 hp. per gross ton day, to 100,000 hp., and even 75,000 hp., assuming the theoretical figure, and were it cheaper to establish such an electric plant than a blast furnace, the possibilities of creating it cannot be thought of in a district but exceptionally and certainly could not be duplicated but rarely, if at all, in the same locality. On this score the attempt to substitute on a large scale, indiscriminately and everywhere an electric smelting plant for our modern blast furnace, is seeking Utopia.

The modern blast furnace is an ideal metallurgical instrument in which the utilization of heat energy approaches perfection in this sense that the metallic product obtained represents, we may say, theoretically, all that a given amount of smelted materials can yield, and no electric furnace could do better on this score. The heat calories of the fuel are accounted for both in the pig iron obtained, slag melted and other contingencies of the process, and in the heat contained in the combustible gases which escape at the mouth. These gases, for many years, have been utilized to heat the blast in special stoves, and also, under boilers, to generate steam, to operate the machinery of the plant, blowing engine and auxiliaries, hoists, pumps, crushers, etc.

It is estimated that, in ordinary conditions, with a good ordinary ore, about one-half of the heat capacity of the fuel is found in these gases. True as this is, it should be remarked, however, that the manner in which these gases are, or have been utilized so far (with only a few exceptions, at least in this country), by far the largest amount of the accountable energy contained in them is now wasted. It is on the utilization of this waste that we desire, in this article, to call the attention, as it opens possibilities for electric smelting in economical conditions which cannot be overestimated, and the importance of this subject is such for the advance of electrometallurgy in general, that we will recall with some details the work done in this direction.

With this part of the gases which is supplied to the hot-blast ovens, we have, of course, nothing to do, and we have only to consider the balance, which is now burnt under boilers to generate steam. To give an idea of the wasted energy resulting from such practice it is enough to remark that a steam motor, working in the best condition, with superheated steam, rarely utilizes more than 5 per cent of the number of calories supplied to the boilers. As a consequence, it not unfrequently happens in some blast furnaces that coke or coal has to be burnt on grates under the boilers to supply the deficiency of the heat supplied by the gases in these conditions. A modern gas motor, on the contrary, as now built, gives readily an efficiency of 25 per cent. Hence, were these blast furnace gases utilized in motors of this kind, there would obviously be a large surplus of power available for other purposes than the requirements of the blast furnace plant. Not only is this efficiency of 25 per cent obtainable now, but the development of the gas engines has rendered possible the construction of machines of 2000-hp., 3000-hp., and even more, and their builders are willing to guarantee: one effective horse-power-hour for every 10,000 thermal units supplied to the motor, which is, indeed, 25 per cent of the theoretical possibilities: since 1 hp.-hour corresponds to about 2545 thermal unit hours, an efficiency of one-quarter, supposes 10,000 thermal units per horse-power hour, or thereabout.

This question of the utilization of the blast furnace gases in gas engines is not new; it has formed the subject of careful examination and actual tests by competent metallurgists in Europe—in England, Belgium and Germany principally—and already the number of engines thus operated with blast furnace gases in Europe represents a total of 200,000 hp. In this country, for reasons not very clearly explained, it only begins now to receive serious attention and practical application.

Mr. F. duP. Thomson, in a very able and extensive article[1], which has appeared in these columns, has already treated this subject with details both theoretically and practically, and we do not intend in the following to take exceptions to his conclusions, for, though very conservative as they are, they are sufficiently promising to still satisfy the most sceptical. But, owing to the importance of this subject, as bearing on electric smelting and as regards the amount of the available surplus of power, and other data bearing on the same questions, it may be interesting to give the results of experiments and practical tests made in Europe, as they will be found to agree very well with calculations and data, based on present regular blast furnace practice, or from personal observations, furnished by Mr. Edward A. Uehling[2], the well-known metallurgist, in a memoir bearing date, June, 1903. The general conclusion is the same as that to which Mr. Thomson has arrived, but it presents the question on a still more favorable aspect. For the sake of comparison, we will tabulate at the end of this article the figures given by Mr. F. duP. Thomson, Mr. Uehling, and by European metallurgists.

Whichever figure may be adopted, the most conservative, or a more favorable one, when one realizes what a stupendous amount of power, now wasted, could be created, even with the introduction of the gas engine in only a part of the blast furnace plants, one may well wonder why this application has been so far limited; as leaving aside, of course, the cost of first establishment, all the objections raised against the use of these gases have been successfully answered. The power thus created, transformed in electrical energy, would render possible, beside its utilization for lighting purposes, simultaneously with the manufacture of pig iron in the blast furnace, the establishment of a steel plant, open-hearth or other, for special steels, high-grade steels, etc., in conditions in which the electric smelting could assume a superiority as to cheapness of establishment and quality of product. We may add here, as another application, the manufacture of ferro metals, or metals for seasoning steel.

(To be concluded in our next issue.)

Notes on Electrochemistry and Electrometallurgy in Great Britain.

(From our Special Correspondent.)

Metallurgical Papers of the Month.

During February the Institution of Mechanical Engineers had no papers of metallurgical interest, so that those in London concerned with metallurgy were able to concentrate their attention upon two papers read before the Institution of Mining and Metallurgy at their meeting on the 16th. Of these, that by Mr. H. F. Marriott on *"Deep Borehole Surveying"* was of considerable technical interest in its succinct description of the electrical devices used for this work. The instruments employed fall into two categories: those which are continuously working, and those which are intermittently working. The general arrangement for either class of work is as follows:

An iron headgear stands over the mouth of the borehole to be surveyed, and the geared drum carrying the electric cable stands at a convenient distance of the rear of this. The drum can be actuated by either hand-power or mechanical methods. In this case a little 5-hp. steam engine is employed. In the headgear is a measuring wheel of extreme accuracy, a vital point where there are known obstructions at various points in the hole. The cable is ½ inch in diameter and contains two highly insulated conductors. It is designed to support its own weight and that of the instrument in a waterless hole of 5000 feet depth, if necessary, though it is unlikely that it would be required to act at this depth. One end of the cable is attached to the head above described, and the other is connected in the interior of the drum to two concentric contact rings, which are placed in communication by means of carbon brushes, with the galvanometer in the continuously recording instrument, or the source of electrical energy in the intermittently recording instrument. The galvanometer itself is of special design, and is calibrated to read, at a pressure of about 4 volts, the make-and-break records of the continuous clinometer, and it is also designed to record the rise in temperature indicated by an electrical thermometer. A small dynamo run by the steam plant is used to supply the necessary current for melting the paraffin wax in the compass and clinometer instrument. The compass and clinometer readings are made by the aid of a protractor of simple design, in which the instrument can be clamped in reverse positions, thus nullifying any errors due to lack of adjustment.

The other paper by Mr. A. Jarman and Mr. E. Le Gay Breretone was mainly intended for analytical chemists. It was entitled *"Laboratory Experiments on the Use of Ammonia and Its Compounds in Cyaniding Cupriferous Ores and Tailings."* Before giving the authors' general conclusions, mention may be made of a series of four tests showing the effect of the presence of ammonium salts in the solutions used for removing copper (Hirsching's process). The percentage of copper extracted varied from 89.8 to 92.2 per cent, an electrolytic assay giving 1.596 per cent Cu, as against an extraction 1.434 to 1.471 per cent Cu. Higher extractions are made when part of the ammonia is present as a salt than when all the ammonia is present as a hydrate. Reviewing Hirsching's method, the authors draw the following conclusions from their experiments:

1. That exceedingly large quantities of ammonia were needed to effect a sufficiently good extraction of copper to allow of subsequent cyanide treatment.
2. That it would be difficult to prevent loss of ammonia by leakage and volatilization.
3. That the recovery of copper and regeneration of ammonia from the solutions would also be costly.
4. That the consumption of cyanide would still be very high after this preliminary extraction of copper.

The authors then proceed to investigate Hunt's method, in which the ore or sands are treated direct by a solution of potassium cyanide to which ammonium hydrate has been added. The gold is thus extracted, together with some copper, and the metals are recovered by electrolytic precipitation, the gold, silver and copper falling to the bottom of the vats as sludge. After describing a variety of tests the authors express their conclusions as regards the Hunt method in the following terms:

1. That for quartzose ores containing cyanicides, particularly copper carbonate, ammonium cyanide is a more efficient solvent for gold than potassium cyanide, in spite of its more rapid deterioration.
2. That ammonium cyanide and potassium cyanide, when suitably protected by ammonia, are about equal in efficiency in the presence of copper carbonate, and are better than unprotected ammonium cyanide.
3. That the quantity of ammonia necessary to suitably protect cyanide solutions from dissolving large quantities of copper depends upon the percentage of potassium cyanide used, and increases with the potassium cyanide.
4. That whilst both ammonia and potassium cyanide are solvents for copper carbonate, yet a combination of these solutions can be obtained which dissolves less copper than does potassium cyanide alone, and it is very near to the point where ammonia exerts its greatest influence in this direction that the maximum gold extraction is obtained (*i. e.,* when using small percentages of ammonia).

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[1] Electrochemical and Metallurgical Industry, March, 1905.

[2] Edward A. Uehling, Memoir, Stevens Indicator, June, 1903.

5. That during treatment the copper in solution rapidly reaches a maximum, but afterwards diminished steadily.

The Faraday Society.

The annual meetings announced for February on page 66 of Electrochemical and Metallurgical Industry was unavoidably postponed until March 6. After the formal business, Mr. F. W. Harbord, of the Royal Indian Engineering College at Coopers Hill, will read a paper entitled: "Recent Developments in Electric Smelting in Connection with Iron and Steel." Such a paper must partake very largely of a resumé of the work done by, and report of, the recent Canadian Commission on "Thermoelectric Processes for Smelting Iron Ores and the Manufacture of Steel." The interest will center largely in the discussion with which I hope to deal in detail in my next letter. For Great Britain, with abundant coal, thermoelectric process are either of academic or financial interest—in their practical application they do not appeal to the British iron founder. In greater Britain—particularly in British India—the working of iron and steel by electrothermic processes is one of the problems of the next year or two. Probably Mr. Harbord will be catechized on this point, as his knowledge of Indian conditions is exceptional.

Market Quotations.

The prices ruling for chemicals do not show any great change. Bleaching powder (35 per cent) being quoted at £4.10.0 per ton in Manchester, caustic soda varying from £8.12.6 for 60 per cent to £10.10.0 for 77 per cent. Shellac continued to fall until the middle of the month, when the price fell to 140s. per cwt., from which price it has not further receded. Copper sulphate is now quoted at £22.15.0, a rise of 10s. during the month.

Para Rubber varies very greatly according to quality, nominally the price for fine is about 5s. 3d. Actually, however, some fairly large consignments from the Straits Settlements have been sold in London at 6s. 3d.

Platinum in small quantities has been sold lately at £4.3.6 per ounce. Cleveland pig iron has risen slightly to £2.8.11. Copper has risen slightly to £68.7.6. Tin has been irregular in price, closing at £131.5.0. Ingot and sheet lead have fallen a further 7/6 per ton to £12.10.0 and £13.15.0. Quicksilver has also fallen from £7.15.0 to £7.12.6 per 75-pound flask.

London, March 4.

Society of Chemical Industry.

One of the most interesting meetings held in this season by the New York Section of the Society of Chemical Industry was that of March 24, which was devoted to a discussion of gas as a source of power. The first part of the programme was devoted to the generation of gas, three papers dealing with this subject. Mr. W. F. Case discussed the composition and caloric value of various gases, dealing successively with producer gas (especially the Loomis-Pettibone gas producer, which he considered the only suitable one for bituminous coal), Mond gas and blast furnace gases. Mr. C. G. Atwater spoke on coke-oven gas, his description of various coke-oven gas works being illustrated by a long series of lantern slides. Dr. J. D. Pennock, of the Solvay Process Co., gave a most interesting account of the use of Mond gas in the soda ash works in Detroit, and of the Taylor producer in Syracuse, the results obtained in many years working being entirely satisfactory.

On account of the late hour, the discussion on the utilization of gas for power purposes—which was the second part of the programme—was somewhat brief and in no way as exhaustive as the discussion of the generation of power. Dr. Franz Meyer made a brief remark on the use of illuminating gas for power purposes. Dr. F. Schniewind spoke briefly on the best system for distribution of gas and pointed out that gas would be distributed not only for power purposes, but also for heating and illumination, and that the illuminating effect of the gas depends distinctly on its composition, there being no direct proportionality between the illuminating effect and the calorific effect. Bituminous coal should not be burned as done at present, but first the volatile constituents and by-products should be recovered. Mr. E. A. Uehling spoke on the blast furnace as the biggest power producer. He pointed out the rapid developments in Europe in recent years of utilizing the blast furnace gases for gas-engine driving. He thought the Lackawanna Steel Company, in making the first installation of this kind in this country, had commenced on too big a scale; since it was the pioneer plant it would perhaps have been better if they had commenced with a smaller plant and had used direct current, for the reason that paralleling of alternators driven by gas engines is a serious complication. However, the Lackawanna Steel Company has now overcome all difficulties, and the plant is now working in a perfectly satisfactory manner. The amount of power wasted in not utilizing the blast furnace gases in this country is somewhat like one million horse-power.

A Plain Talk on Trade Journals.

This was the title of a very suggestive lecture, held recently before the American Trade Press Association, by Mr. Arthur Warren, manager of publicity for the Allis-Chalmers Co. Mr. Warren has himself been a distinguished newspaper editor, correspondent and contributor, and has been the first to organize a department of publicity in the industrial world. His address was an imposing array of brilliant and incisive epigrams, somewhat of the Bernard Shaw or James Swinburne style, yet of a distinctly personal flavor. "Who reads the trade papers?" "Some years of experience have forced upon my attention a certain peculiarity on the part of the busy men who haven't the time to read the trade papers. It is this: they know, with an amazing quickness, whenever an article or a paragraph appears, actually or relatively, derogatory to their interests. Hence, it is clear that somebody reads the trade papers." * * * "Advertising space is not the sole instrument used by the Manager of Publicity in the industrial world. It is only one of his tools of trade. Industrial publicity is bounded by limitations which more popular forms of bold advertisement know nothing of. It is one thing to bespatter the universe with the praises of a soap, a breakfast food, a corset, or a pill; to make yourself Governor of a State by first familiarizing a nation with your manly countenance as the guarantee of the perfection of a boot; and it is another thing, a very different thing, to increase, by advertising, the sales of reciprocating engines, steam turbines, hydraulic turbines, gas engines, mining machinery, flour mill machinery, rock and ore-breakers, saw-mill machinery, hoisting and pumping machinery, electric generators, motors, transformers, controllers, and the like. The advertiser of the pill, the boot and the soap, sings his song to the world as it runs and travels and plays, and he sings it with a sledg-hammer accompaniment, a sort of anvil chorus. He invents a phrase and repeats it until you know it by heart, and speak it in your dreams; he entertains you with pictures, and amuses you with puzzles. Turn where you will you cannot escape him. He is omnipresent. If you believe the experts who expound their art so solemnly, he is omniscient. It doesn't matter what he says, it's all in the way he says it. The industrial advertiser has not this advantage. He speaks to a sophisticated audience, an audience trained by daily experience with the sharp cares of business, and founded, in many thousands of cases, upon a technical education along the very lines upon which he seeks to address it. The industrial advertiser speaks to a public which, to say the least, is quite as likely to know as much about the wares he advocates as he knows himself. And there is no one so cautious as your technical man when he touches publicity. I am not altogether sure that technical men wholly approve of publicity, although they may revel and wallow in the fruits of it. Submit to one of them the proof of an engine or electrical ad. His view-point is not yours. He will cover all your white space with black dissertations on shunt windings and efficiency curves, with talk of tests and measurements, weights, pressure, and electromotive force, while you are trying to tell in a few terse phrases what the machine is, what it is for, and what it will do." * * "The Publicity Man's business is publicity as applied to the business of selling machinery, not as applied to the art of constructing it. And the trade and technical press is a vehicle for industrial publicity. And because of that it flourishes" * * "Industrial publicity is a new thing. It is still young and learning to walk. I sometimes think that no one appreciates the possibilities of publicity so much as the Publicity Man himself. And its appreciation in the industrial field is a thing of yesterday morning. The day before, no one thought of it. But 'the old order changeth, yielding place to new.' For all that there is no industry, among all those represented on this occasion, that has all the publicity it needs; there is none that has begun to more than taste the fruits of advertising; there is no industry, there is no metal-working manufactory that spends money enough on publicity. Manufacturers are, no doubt, accustomed to say 'Publicity cost so and so.' They are not yet accustomed to think 'Publicity earns so much.' That is a part of the lesson they have yet to learn. And they will learn it, and appreciate it. Yet I once knew a manufacturer who said: 'We are getting too much publicity.' There are evidences that his opinion is changing." * * * "Every advertiser should buy and handle his own advertising space, design his advertisements, and, in short, carry his own Department of Publicity." A long part of Mr. Warren's address was devoted to a discussion of circulation. "In circulation we recognize two elements that make value, quality and quantity; and these two are the product of a third, character—the character of the publication."

ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS.

By George P. Scholl, Ph. D.

Electric Furnaces and Furnace Products.

Method of Agglomerating Magnetic Ore.—E. Gates, Chery Chase, Md. Patent 780,716, January 24, 1905. Application filed January 14, 1901; renewed November 5, 1902.

The invention relates to the treatment of magnetic iron ore sands, in order to bring about their agglomeration into larger lumps, and thus make them available for use in a blast furnace. The operation is performed by allowing the particles to be fused by an electric arc sprung between two inclined feed troughs, constructed of conducting material. The opposing ends of the inclined troughs are brought together when the process is started, and are then drawn apart, so as to establish the arc between them, and the magnetic sand is fed from hoppers upon the inclined troughs, down which it slowly descends. The falling streams of ore approach each other and the arc passes between them along the line of least resistance. Care is said to be taken that the sand is fed at a speed in excess of that necessary for the utilization of all the energy of the current. As a result, the sands are agglomerated to small lumps, varying from the size of a grain of wheat to that of a bean. They accumulate in a tray located beneath the arc, where they are partially cooled, but from which they overflow and drop into a hopper, while still hot. The fused particles are thus brought into contact with the partly cooled materials in the tray and form lumps of magnetite of increased size. The treated material then passes from the hopper into a revolving screen, where the lumps are screened out and from which the untreated sand is returned to the apparatus for retreatment.

Electric Furnace.—J. M. Morehead, Chicago, assignor to Union Carbide Co. Patent 782,917, February 21, 1905. Application filed October 19, 1903.

The furnace is specially intended for the production of calcium carbide, and is an improvement on the inventor's former furnace, which comprised a hood enclosing the electrodes, the charge being fed around the hood in order to exclude the air and to direct the waste gases into the hood. The inventor states that he has found that the charge may advantageously be fed directly into the hood, and that the waste gases will still collect in the hood, although obliged to pass through a considerable body of the charge before they reach the gas outlet. This upward movement of the gases is believed to be due to the fact that the gases rising from the zone of reaction at the lower end of the electrodes tend to work upwards through the charge along the electrodes, and that they will therefore pass along the electrodes through a path of greater length than through the charge itself. It is stated that this new construction enables the furnace to be operated with great regularity and without filling the surrounding atmosphere with dust from the finely divided charge. The furnace, shown in vertical section in Fig. 1, is of the Horry type, the working chamber consisting of a wheel 1 with removable cover plates 2. An iron hood, carrying electrodes 4 fixed in vertical adjustable holders 5, depends into the open end of the working chamber. The top of the hood is closed by a cast-iron plate, containing a chamber for water-cooling purposes, with water inlet and outlet 8 and 9. Parallel openings 10 are provided for receiving the electrode-holders, and a refractory luting 11 is filled into the space left between the walls of the openings and the holders, after the electrodes have been adjusted to their proper height. A pipe is provided for the purpose of introducing the charge, behind the electrodes, and another pipe serves for withdrawing the gases, and is located in front of the electrodes. During the operation the electrodes are adjusted to their proper height and the charge is fed in until it fills a considerable portion of the hood and thus seals its lower end.

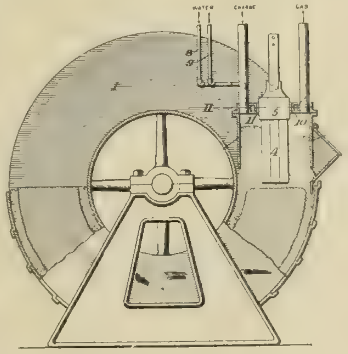


FIG. 1.—MOREHEAD'S CARBIDE FURNACE.

Process of Smelting Refractory Ores.—E. F. Price, Niagara Falls, assignor to Union Carbide Co. Patent 782,922, February 21, 1905. Application filed October 19, 1904.

The process relates particularly to the production of calcium carbide, and reference is made to patent 750,096, January 19, 1904, (described in Electrochemical Industry, Vol. II, page 109), obtained by Mr. Cowles. In the latter the current is passed through a charge in a furnace with downwardly converging side walls, in order to continuously increase the current density as the charge reaches the lower parts of the furnace. The electric current is passed by means of carbon electrodes, the upper electrode being constituted by a carbon ring in the upper part of the furnace, while the carbon hearth of the furnace forms the other electrode. It is claimed by the inventor of the present patent that carbon electrodes of the size and character required for this operation are expensive and perishable; that it is difficult to obtain large blocks of uniform composition and that the carbon is rapidly worn away by the attrition of the charge and its products, and in many cases reacts chemically with them. The inventor therefore proposes to interpose an electrically conductive charge of a calcium compound and a reducing agent, specifically a mixture of lime and carbon, between water-cooled metallic electrodes, and to smelt it by the heat generated by the resistance of the charge. The lower electrode is preferably constituted by a body of molten iron or other metal which will not combine with the carbide in the lower part of the furnace chamber. The molten iron in that case is stated to purify the carbide by withdrawing the silicon from it, which latter is one of the normal impurities in carbide, by which means the iron is converted into ferro-silicon. The furnace is shown in vertical cross-section in Fig. 2. It comprises a vertical stack with a body 1 of refractory material, such as magnesia or siloxicon fire brick, surrounded by a water-jacket and supported upon a cast-steel plate, with cooling chamber and electric terminal. The upper electrode is supported upon this body and insulated from it by a layer of non-conducting refractory material. It is constituted by an iron ring, surmounted by a water-jacket, the inner surface of the ring being bare, in order to insure contact with the charge. Bell and hopper and outlet flue are provided as usual, in an iron ring, which covers the top of the furnace. When the operation is started, the furnace is filled with a mixture of lime and carbon until its upper portion lies in contact with the electrode ring, initial current paths being provided if the charge is normally a poor conductor. The conductivity of the charge may be increased by using a mixture containing large pieces of coke, which lie in contact with each other at various points, and thus furnish direct paths for the current. A current of sufficient amperage is then passed, the temperature in the charge rising gradually downwardly on account of the decreasing cross-section of the furnace, down to a zone where the materials react to form carbide and the latter is brought into a molten condition. In normal operation the major portion of the body of iron will be maintained in a molten condition by the heat of the charge, the lower portion being pasty or solid, owing to the influence of the cooling water. The molten carbide accumulating upon the iron is intermittently or continuously tapped off through the tapping-hole, and fresh material is fed in as required. The waste gases preheat the charge and after passing out through a pipe, are used as fuel. The silicon withdrawn from the molten carbide by the iron converts the latter into ferro-silicon, which is removed from time to time by a tap-hole, more iron being introduced with the charge.

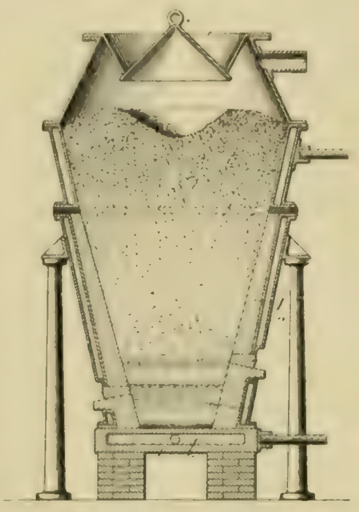


FIG. 2.—PRICE CARBIDE FURNACE.

Electric Furnace.—G. O. Seward, Holcomb's Rock, Va., assignor, to Willson Aluminum Co. Patent 783,736, February 28, 1905. Application filed May 3, 1904.

The furnace illustrated in this specification is of circular cross-section, with two electrodes of opposite polarity passing through the top cover. The improvement claimed in the invention consists in the construction of this top cover, in such a manner that electrodes of different sizes can be used with the same cover; that repairs are facilitated and the damages to the top due to the establishment of arcs from the electrodes are lessened. The cover used by the inventor is flat and made hollow for the purpose of circulating water for cooling. Apertures are provided in the cover for the electrodes, which apertures are somewhat larger than the electrode. Each electrode is fitted with a horizontal flange overlying the top of the cover and a packing ring of asbestos, asbestos cloth or other fireproof insulating materials is introduced between the lid and the cover. The electrodes are thereby insulated from each other, and the construction is stated to have the further advantage that in case of an accident, and the establishment of an arc from the electrode, it is only the special lid of that electrode which is damaged, and a new lid can be supplied very quickly without even removing the furnace cover.

Process of Utilizing Scale from Calcium Carbide Ingots.—T. R. Edmands and J. Scales, Sault Ste. Marie, assignors to Union Carbide Co. Patent 784,255, March 7, 1905. Application filed December 22, 1904.

The inventors propose to utilize the carbide scale by treating it with sufficient water in order to decompose the carbide it contains. The residue is then dried and smelted in the usual carbide furnace after sufficient carbon has preferably been added to make the total carbon contents of the mixture equivalent to its contents of lime. It is stated that in practice it is desirable to crush the scale into small pieces, and to effect the decomposition of the carbide contained in it in an acetylene generator, provided with agitators, so as to insure contact between the particles of the scale and the water. The gas is collected and utilized and the water is driven off from the residue by means of any suitable drier. The residue, when mixed with sufficient carbon, is then either smelted directly or added to the normal furnace charge of lime and carbon.

Electrolytic Production of Metals and Compounds.

Electrolytic Process of Reducing Metallic Sulphides.—C. E. Baker and A. W. Burwell, Cleveland. Patent 782,894, February 21, 1905. Application filed July 1, 1904.

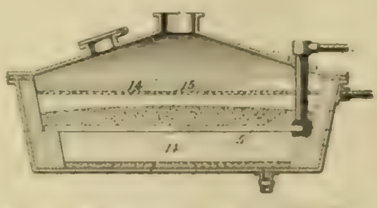


FIG. 3—APPARATUS FOR REDUCING SULPHIDES.

The process depends upon placing the metallic sulphide or sulphides in contact with or in proximity to the anode of an electrolytic cell in which a molten chloride is electrolyzed. The chlorine given off there replaces the sulphur in the sulphide, and the chloride thus formed melts and replenishes the electrolyte. The operation is carried out in a vessel shown in cross-section in Fig. 3. It consists of an iron vessel, the sides of which are provided with a refractory lining of magnesia fire brick or other non-conducting material. The anode is supported on shoulders, provided on opposite sides of the lining. The bottom of the vessel constitutes the cathode, and is connected to a source of current. The anode consists of a number of parallel bars 5, preferably of Acheson graphite, which are spaced apart and enter a horizontal bar conductor, from which rises a terminal post, which in turn is connected to the supply conductor. The bar and the post are preferably made of graphite. The vessel has a cover luted to it by means of asbestos cement. A charge opening and gas outlet are provided as usual, and a bushing insulates the post from the cover. In operation the cell is filled with an electrolyte 14 of molten chloride; for example, lead chloride. The metallic sulphide, for instance, lead sulphide, is preferably supported on the anode itself, although a separate shelf in proximity to the anode may be employed. The chloride is then electrolyzed, the lead being deposited as a molten layer on the bottom of the apparatus, whence it may be removed by means of a cock. The chlorine set free at the anode is stated to decompose the sulphide, freeing the sulphur and reproducing lead chloride. The sulphur may either be allowed to accumulate as a floating layer 15 on the surface of the electrolyte, and may then be skimmed off, or it may be volatilized and driven off through an opening. By regulating the current density so that an excess of chlorine is generated, sulphur chloride (S_2Cl_2) may also be formed. It is stated that a chloride of a metal other than that of the sulphide to be treated may be used as the electrolyte, and that, for example, molten zinc chloride has been employed for the treatment of lead, as well as zinc sulphide. Mixed or complex sulphides may also be treated.

Apparatus for Cyanide Treatment.—E. L. Oliver, Oakland, Cal. Patent 784,120, March 7, 1905. Application filed June 20, 1904.

The object of this invention has been to overcome the difficulty of scouring the amalgamated plates used for recovering the values from a cyanide solution, when slimes or sands are agitated with cyanide directly in the vessel in which electrolysis is being carried out. The apparatus is illustrated in Fig. 4, in vertical section. It consists of a tank *A*, with a conical bottom, provided with a discharge, which is controlled by a gate *b*. A casting *d* with openings *d'* is bolted to the bottom, and a pipe *D* fits into it. An air pipe leads from an air compressor into the interior of the pipe *D* and terminates a short distance above the bottom of the latter. Electrodes *E* and *F* (the latter not shown, but parallel to *E*) are located in the upper part of the tank, and a large number of them are provided, so as to provide a large surface. The electrodes *E* are the anodes, and are made of iron plates or other suitable material, while the electrodes *F* are the cathodes and made out of copper plates coated with mercury. The electrodes are suitably supported on block and strips *G* and *H*, and boards *T*. By means of a trough the copper plates are automatically provided with mercury, thus lengthening their period of use before they have to be taken out to remove the deposits. The operation of the apparatus is as follows: The material to be treated, and the cyanide or other solvent solutions are filled into the tank until they reach above the level of the pipe *D*. Air is then turned into the pipe *C*, which forces the solution out of the pipe *D* and draws it in through the apertures at the bottom, thus maintaining a continuous circulation past the electrodes. At the same time the electric current deposits the gold and silver in the solution on the amalgamated plates. It is stated that this agitation and circulation of the solution can be performed as gently as desired, and that though efficient contact is made with the amalgamated plates, there is no scouring action upon them.

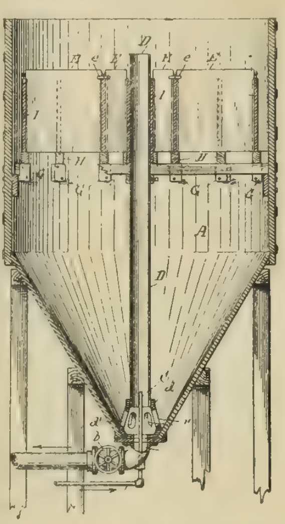


FIG. 4.—CYANIDE APPARATUS.

Storage Batteries.

Electrolytic Production of Superoxides in Alkaline Solutions.—H. Rodman, Philadelphia, assignor to Electric Storage Battery Co. Patent 782,989, February 21, 1905. Application filed May 17, 1902.

The object of the invention is to accomplish the production of metallic superoxides, such as that of nickel, directly from the metal, for the purpose of using them as active material in storage batteries. In the case of the production of nickel superoxide, the metal is constituted the anode, an iron cathode being used. As under ordinary conditions the nickel would be passive in an alkaline solution, a very dilute alkaline solution is used as an electrolyte, to which may be added an acid radical, either acid or a salt containing the desired radical being available for use. As example of such substances may be mentioned inorganic salts or acids, as sodium perchlorate, sodium nitrate, etc., also organic salts or acids, such as tartaric, acetic and oxalic acid, etc., and sodium salicylate, etc. An electrolyte suitable for the practice of the invention would be a one-half per cent solution of caustic potash, either by itself or containing one-tenth of one per cent of sodium perchlorate. A current of about half an ampere per square inch of anode surface, more or less, is appropriate. The temperature of the solution may be less than 100° F. Instead of using such a very diluted solution as mentioned above as an electrolyte, the bulk of the solution may be of ordinary strength, and only that part of it which is in immediate vicinity of the anode, may be made dilute. This may be accomplished by placing fibrous material, such as blotting paper, adjacent to the anode, which acts like a diaphragm, and permits the portion of the solution nearest to the anode to become dilute by the electrolytic action.

SYNOPSIS OF PERIODICAL LITERATURE.

A Summary of Articles Appearing in American and Foreign Periodicals.

Practical Applications.

Electrometallurgy of Iron and Steel.—Charles Combes, in a paper before the *Societe d'Encouragement pour l'Industrie Nationale*, discussed P. L. T. Héroult's work in the electrometallurgy of iron and steel. Since Dr. Héroult's work in this field has been dealt with repeatedly and at length in our columns (Vol. I, p. 63, 287, 449, 461, 467; Vol. II, p. 408, 481) we give only those parts of Combes' address which contain new information.

Ferro-Chrome.—For making ferro-chrome, Héroult uses a crucible lined with chrome ore, and the two electrodes are suspended into the slag above the molten bath (in the same way as in his steel furnace). By adjusting properly the amount of carbon for reduction, he makes ferro-chromes which contain 2 to 6 per cent. of carbon. It is possible to reduce the amount of carbon still further below 1 per cent by a refining reaction in the bath; for this purpose an excess of chromite is added to the slag, and the excess of carbon is removed by the reaction between the molten bath and the oxidizing slag. The "soft" ferro-chrome thus produced differs considerably from the "hard" ferro-chrome, containing 9 or 10 per cent. C. The soft alloy is white and compact, is not brittle and presents on fracture a crystalline structure which is the more developed the less the content of carbon. Sometimes parts of the alloy are obtained in form of magnificent crystals up to 1 cm. in size. They do not represent a strictly defined chemical compound. On the contrary, each crystal is made up of a series of concentric envelopes of different nature, one fitting always exactly on the next one. These alternating layers consist of two essentially different compositions; the one being soluble in dilute hydrochloric acid and containing no carbon in notable quantities; the other being insoluble in dilute hydrochloric acid and containing 5.4 per cent of carbon. The relative proportions of chromium and iron are a little different in the soluble and insoluble layers. In 100 parts of [illegible], the [illegible] which dissolve in acid have a composition of 46 per cent. Cr, 54 Fe, 0.03Si, 0.16Mn. The layers, insoluble in dilute acid, are soluble in hot concentrated hydrochloric acid, a residue of carbon being left. The composition is given as (Fe 1.5, Cr 1.5) C.

Reduction of Pig Iron from Ores.—Concerning Héroult's work in this field, Combes deals especially with the "economizer" and with another process for saving electrical energy which were already mentioned in our Vol. II., p. 408. The idea of the economizer is given as follows: "For the reduction of 1 kg. of pig iron, 2000 to 2500 calories are required for the reduction as well as for the fusion of the bath and slag. For the reduction, 330 grams of carbon are required, CO being taken as basis of calculation. The heat developed in the furnace during reduction is $24 \times 0.33 = 792$ calories. The difference amounting to 1200 to 1700 calories must, therefore, be supplied by electrical energy. But the burning of CO can furnish $0.33 \times 5600 = 1800$ calories. Let us now suppose that this CO is burned in a special apparatus in which the ore is fused and even overheated before it comes into the reduction furnace. In this way it is possible to recover the greatest part of the calories which are lost." The second method of saving electrical energy was briefly outlined on page 408 of our Vol. II., apparatus for this purpose being described on p. 467 of our Vol. I.

Steel Making.—As repeatedly described (f. i. Vol. II., p. 481) Héroult removes the sulphur and phosphorus from the molten bath by means of artificial slags, which are renewed as often as necessary (one method, for instance, being based on the action of CaC_2 added to the slag). The metal is afterwards recarburized by means of "carburite." Carburite is "a mixture of iron and carbon filings or shavings with carbon and the required quantity of binding material. This product is dense enough to pass through the layer of slag and come in contact with the metallic bath in which it dissolves rapidly on account of the high temperature of the arc. It is thus possible to realize the deoxidation and recarburation of the metal without addition of any foreign material such as silicon and manganese, and the required composition can be obtained very exactly by weighing in advance the quantity of carbon to be used." The author describes some mechanical details of construction of the furnace, all of which were already described in these columns.

Induction Furnace.—In the discussion of Combes' paper, Messrs. Lodin, Saladin and LeChatelier participated; the latter paying a special tribute to the services rendered by Héroult to the French industries. Saladin spoke on the induction furnace and remarked that not Kjellin, but Ziani de Ferranti (English patent 1885, Jan. 15) is the original inventor of this type of furnace. (See also the letter in the correspondence columns of this issue.) Schneider & Co. (Le Creusot) have devised a modification to produce circulation in the furnace. As described in our Vol. II, page 283, they construct the furnace in form of a tube of small cross section, which communicates at both ends with a chamber of large dimensions containing the greater part of the molten metal. The latest form of the furnace, as used by Schneider & Co., is shown in Fig. 1. It consists of a large reservoir communicating with two straight tubes, in connection with a three-core transformer. The complete paper of Combes, together with the discussion, is given in the January issue of *Revue de Metallurgie*.

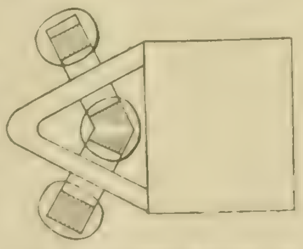


FIG. 1.—INDUCTION FURNACE.

Electric Furnace in the Iron and Steel Industries.—In *Cassier's Magazine* of March, H. Allen gives some calculations [illegible] to the thermochemistry of iron ore reduction and steel making in the electric furnace. The author concludes that "a study of the thermochemistry of the production of iron and steel by means of the electric current shows that when carried out on a system as nearly as possible in accordance with that required by theory, the capabilities of profitable applications only require scientific treatment to prevent an excessive charge for electric current from turning the scale between profit and loss, even in the field of the blast furnace."

Detinning of Tin Scrap.—In the *Elektrochemische Zeitschrift* of February, H. Mennicke begins to give a review of the industry detinning tin scrap by electrolysis since 1902. In the present instalment the author deals especially with the commercial side of the question. He points out that the financial success of a detinning plant depends no longer on the soundness of the process alone—detinning with an alkaline electrolyte is cheap and easy—but rather on the geographical situation of the works, on the supply of scrap, and especially on a good buyer of scrap. The pioneer German plants have formerly earned considerable dividends by this industry, but since that time the price of scrap has been greatly increased by competition, new works having been erected. The author does not think that the outlook is encouraging for the new works, since the whole supply of scrap is in the hands of the old firms.

Electrolytic Ampere-Hour Meter.—In the *Zeit. f. Elektrochemie*, March 10, H. Danneel discusses mercury voltameters. Mercury is specially suitable on account of its high equivalent weight, and for the reason that from mercurous nitrate solutions it is deposited monovalent, Faraday's law being fulfilled with great accuracy. The mercury which is deposited may be weighed, as in Danneel's arrangement in which the cathode is suspended on a balance, or its volume may be measured, as in the Wright meter. A modification by Lux of the latter is described; Fig. 2 shows the arrangement of the meter from the side and from the front. The anode *A* is on the top, and is always at the same distance from the platinum cathode *B*. The anodic mercury *A* rests on a narrow-mesh wire screen from which it does not drop off on account of capillarity. As soon as some mercury goes into solution from the anode *A* it is replaced by fresh mercury from the reservoir *D*, while some of the solution above *A* enters into *D*. The arrangement of the anode above the cathode provides automatic circulation of the electrolyte, the heavier anode liquid flowing downwards while the lighter cathode liquid rises. The first mercury which is deposited in the platinum cathode *B* adheres on it. When some mercury has accumulated on it, all mercury which is deposited afterwards drops into the funnel *J*.

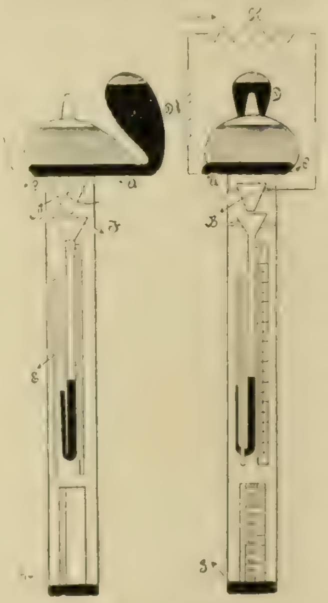


FIG. 2.—METER.

In the first moments the reading of the meter will, therefore, be too low. But after it has been in service for a short while, as much mercury drops into *J* as is deposited on *B*. The mercury accumulates gradually in the first and second vertical tubes until it rises to the top of the second tube. Then by a syphon action the whole mercury flows off through the third tube, the arrangement being quite clear from the illustration. The mercury then accumulates in the lower part of the apparatus. There are two scales, the lower scale acting like the hour hand of a watch, the upper scale like the minute hand. The meter is connected in parallel with an adjustable resistance *R*, which is inserted in the circuit carrying the current to be measured.

Kryptol Electric Furnaces.—In view of the increasing use of kryptol as resistance material in electric furnaces, we give the following references to publications on this subject, the principle having been repeatedly described in our columns. A. Voelker described various applications of kryptol in *Verhandl. d. Ver. z. Beförd. d. Gewerbfleisses,* 1904, Heft 5, a greater number of apparatus using kryptol being here illustrated. A shorter account which is also illustrated, was given by W. Bermbach in *Elek. Anzeiger*, 1904, Nos. 60, 61.

Theoretical and Experimental.

Bromine.—The potential of a halogen electrode depends on three variables, the concentration of the free halogen, the concntration of the halogen ions, and the temperature. The *Zeit. f. Elektrochemie,* January 27, contains a long paper by F. Boericke on experiments in which he studied the following problems: The solubility of bromine in potassium bromide and hydrobromic acid solutions; the potentials of bromine in potassium bromide and hydrobromic acid solutions; and the anode potentials during the electrolysis of neutral potassium bromide solutions. The main results may be summed up in the statements that the strong solubility of bromine in water, or in bromide solutions permits the reversible discharge of bromine ions at a platinized platinum anode at such anode potentials, which even with a higher current density are still below the bromine potential corresponding to saturation. On the other hand, at a polished platinum anode, on account of the changes of the electrode condition, very much higher voltages than correspond to the bromine potentials are required to pass a strong current through the anode.

Electric Combination of Hydrogen and Oxygen.—Rev. P. J. Kirkby has formerly described some experiments which exhibited the effect of passing electricity with a luminous discharge through hydrogen and oxygen, mixed in equivalent proportions and at low pressures. The experiments showed that a partial union of the gases into water vapor is the result, and that if the pressure and the potential difference of the plates during the discharge remain steady, the quantity of water vapor formed, is proportional to the quantity of electricity which has passed—in other words, to the product of the current and the time. The same investigator has now made further experiments to study; first, how changes in the distance between the parallel electrodes, between which the discharge takes place, would affect the results, and secondly, whether the same effects would be observed if non-oxidizable plates were used. His paper is published in the January issue of the *Philosophical Mag.,* and abstracted in the Lond. *Elec.,* January 27. He prepared the gases simultaneously by the electrolysis of pure barium hydrate contained in a small sealed glass vessel. By this method the gases were generated free from hydrocarbons, ozone and peroxide of hydrogen. The mixed gases were dried and admitted to a glass bell jar containing the parallel plates between which the current was to be passed. The parallel plates were connected to the poles of a battery of small lead cells, ranging up to 1080 volts, through a high resistance and a sensitive ammeter. In this way the quantity of electricity passed through was known, while the pressure before and after its passage were determined by McLeod gauges. The results showed that in spite of great variations in the pressure and in the distance between the electrods, the lowering of pressure in millimeters of mercury bore to the quantity of electricity in coulombs passed through the hydrogen and oxygen a nearly constant ratio, amounting to about 2, and ranging from 1.6 to 2.4. The distances experimented with, ranged from 0.25 cm. to 1.5 cm., and the pressures from 1.1 mm. to 9.3 mm. of mercury. Within these limits, at all events, from six to nine times as many molecules of water vapor are formed as pairs of ions. This result is independent of the capacity of the apparatus and of the nature of the electrodes.

Metallurgy.

Copper and Brass.

L. Guillet, in *Revue de Metallurgie* for February, publishes a long article of 24 quarto pages on the copper alloys and special brasses. The matter was the subject of a lecture given before the Société d' Encouragement pour l'Industrie Nationale on the 13th of January. A diagram is given showing the variation of tensile strength and extensibility for brasses up to 50 per cent zinc. The tensile strength of rolled brass, annealed, reaches its maximum of 57 kilos for square millimeter (81,000 pounds per square inch) at 30 per cent of zinc, the elongation its maximum of 49 per cent at 44 per cent zinc. Very beautiful photomicrographs are given of alloys with 8, 33, 41, 48, 57, 79, 89 and 93 per cent of zinc.

Under the heading of "special brasses," are considered those with lead, tin, aluminium and manganese. Using lead, up to 5 per cent, can be added without liquation, the micrographs showing that the lead enters mechanically between the crystals of brass, splitting large crystals into small ones, and so facilitating rolling and forging. Above 5 per cent the lead tends to liquate out in irregular masses, but the brass can be worked and rolled with up to 7 per cent. It is to be regretted that Mr. Guillet stopped with this amount, and did not investigate these very useful and important brasses with up to 25 or 30 per cent of lead, which make such superior bearing metal.

The study of the "Kalchoids," the tin-zinc-copper alloys did not reveal any new information. The naval brass of England and France contains 60 to 62 parts copper, 37 to 39 parts zinc and 1 to 1.5 parts tin.

In manganese brasses, it was found that up to 10 per cent of manganese it acts simply as so much zinc; an alloy with 45 zinc and 10 manganese, for instance, acting like a brass with 55 per cent zinc.

Aluminium brasses were particularly studied. Using brass with 40 per cent of zinc, the introduction of 0.5 to 1 per cent of aluminium gives a fine golden color; this color persists to about 5 per cent of aluminium, and then becomes rose; at 7 per cent a fine rose color is produced, and the alloy is dichroic; that is, it is gray, seen at a certain angle of incidence; at 10 per cent the alloy becomes as white as silver. Up to 4 per cent it can be worked hot; above that it is very difficult to work; above 6 per cent it can no longer be rolled; the 10 per cent alloy is so brittle that it cannot even be polished. As for the mechanical properties in the testing machine, the addition of aluminium copper-zinc alloy acts like a strong increase of zinc content, each per cent of aluminium acting somewhat like an increase of 3.5 per cent of zinc.

Concerning the quenching of copper-tin bronzes, the maxima of tensile strength and elongation were obtained by quenching from the temperatures, as given:

Copper.	*Tin.*	*Maximum Strength.*	*Maximum Elongation.*
95	5	450° C.	400° C.
91	9	700° C.	800° C.
87	13	725° C.	725° C.
84	16	600° C.	650° C.
79	21	600° C.	650° C.

In every case except the first (5 per cent tin), the strength and elongation decreased for quenching temperatures up to 400-450° C., and then rapidly increase to maxima at the temperatures given.

Gold.

Cyanide Process.—Cyanidation in the United States is discussed by G. H. Fulton in the *Eng'ing and Mining Jour.* of January 5. He first deals with electric precipitation. At the present time there are two such plants in the United States and one in Mexico. This recovery does not compete with zinc precipitation, as applied to ordinary gold-bearing cyanide:

it finds its application to foul solutions carrying much base metal, such as are obtained from weathered tailing, or from silver ore, where the bulk of the precipitate becomes very great. The Malm process is in use by the Gold Cotd. Mining Co., treating the old tailing of the Empire mine, near Maryville, Mont., as described by Mart. W. Anderson. The precipitation takes place in boxes (9 by 10 feet in cross-section, and 5 feet deep), divided longitudinally into three compartments. All electrodes are of No. 18 iron. The sub-plates are coated on the positive side with graphite. The current density is 0.25 amperes per square foot of cathode surface. The solution does not circulate through the boxes, but is permitted to stand in contact for 1.5 to 2 hours, when it is drawn off and replaced by fresh solution. The deposit forms in flakes on the negative side of the sub-plates, adhering to them loosely. It consists on the average of 51 per cent copper, 3.2 per cent gold and silver, some arsenic, antimony, lead and considerable lime. The author then describes E. M. Hamilton's electric precipitation in the Butters' plants at Minas Prietas, Mexico, and at Virginia City, Nev. This process was described at length in our Vol. II, p. 131.

The author then deals with the crushing of ore. He remarks that Chilean mills are finding increased application, both for large and small units, even replacing stamps. This is true, both for mills, which cyanide direct, and also for those which amalgamate previous to cyanidation. The roller mill presents the advantages of lower cost of installation, less power consumption, and large crushing capacity for small space. When they are used, rock breakers and coarse rolls prepare the ore for subsequent treatment. The author then discusses the treatment of slimes. While the ordinary decantation process or some modification of it is still standard in this country, yet its imperfection involving the loss of soluble gold and cyanide is becoming recognized and experimentation points to filter pressing. "Fine grinding, for high extraction and quick solution of of gold and silver, has been emphasized recently; instead of the prevention, the production of slime seems to be favored. In this connection, however, the question arises: What method of slime treatment is available? With most ores, the slime will have a higher value than either the sand or the original ore, and unless the filter press commends itself on the score of cost, excessive sliming presents a doubtful advantage, owing to the loss of dissolved gold inherent in the decantation process; in the Black Hills the loss in dissolved gold ranges from 5 to 9 per cent of the assay value. This fact has cooled any ardor to crush finer, and with ore that is too low grade to permit filter pressing, the production of an excessive amount of slime is to be avoided. It is probable, however, that the filter press will be modified in the near future so as to reduce the cost of operation and make it available for low-grade slime." The author then deals with the precipitation and treatment of precipitate, with the increasing use of sodium cyanide, and gives a review by States.

Electroplated Copper Plates in the Battery.—The *Journal* of the Chem., Metal. and Min'g Soc'y of So. Africa, of October, 1904, contains a paper by F. W. Cindel, advocating the introduction of silver-plated copper plates into metallurgical practice on the Rand. The author gives an outline of the practice of the Homestake Gold Mining Co., in South Dakota, where the ore under treatment is a hornblende, garnetiferous schist or slate infiltrated with free silica and pyrites; in a 200-stamp mill, there are 160 plates with an amalgamated surface of 8640 square feet. He makes a comparison with Rand practice where the area of plates for a 200-stamp mill is only 2400 square feet; he therefore recommends additional plates.

Miscellaneous.

Mineral and Metal Statistics.—The issue of Jan. 5 of the *Eng. and Mining Jour.* contained the carefully prepared and very elaborate annual statistics on mineral and metal production during the past year. The following figures give the United States production in 1904, the figures in parenthese giving the output in 1903 for the sake of comparison.

Non Metallic.				
Arsenic, white	198	(611)	short tons	$29,564
Barytes	22,592	(50,397)	short tons	74,736
Bauxite	48,966	(48,087)	long tons	189,325
Bromine	897,100	[illegible]	pounds	313,985
Carborundum	7,000,000	[illegible]	pounds	700,000
Cement, natural hydraulic	5,000,000	(7,030,271)	bbls. of 300 lbs.	2,500,000
Cement, Portland	22,000,000	(22,342,973)	bbls. of 380 lbs.	20,900,000
Coal, anthracite	69,749,962	(75,288,206)	short tons	149,990,870
Coal, bituminous (including cannel)	273,774,922	(277,076,986)	short tons	316,763,701
Coke	23,500,000	(25,262,360)	short tons	56,400,000
Cobalt, oxide	100,000	(120,000)	pounds	215,000
Copper sulphate (including a small amount made from metallic copper)	63,309,394	(43,124,454)	pounds	2,849,484
Copperas (including only that marketed as copperas)	12,404	(20,240)	short tons	868,315
Crushed steel	395	(378)	short tons	53,300
Fluorspar	26,180	(42,523)	short tons	153,081
Garnet	3,363	(4,413)	short tons	99,463
Graphite, artificial	3,195,251	(2,620,000)	pounds	221,686
Graphite, crystalline	4,594,282	(4,525,700)	pounds	173,120
Iron ore	29,366,654	(32,471,550)	long tons	36,708,318
Lead, white	118,079	(112,700)	short tons	11,807,900
Lead, red	13,800	(12,300)	short tons	1,759,500
Lead, orange mineral	800	(1,000)	short tons	124,000
Limestone and dolomite flux	8,788,000	(12,029,719)	long tons	3,515,200
Litharge	14,200	(12,400)	short tons	1,597,500
Phosphate rock	1,782,503	(1,570,228)	long tons	5,703,582
Pyrite	181,763	(199,387)	long tons	647,276
Sulphur	191,250	(39,310)	long tons	8,139,600
Zinc-lead	6,781	(4,500)	short tons	398,384
Zinc oxide	58,808	(54,034)	short tons	5,513,838
Zinc ore, exported	31,392	(39,418)	short tons	882,516
Total non-metallic				$629,295,184
Metallic.				
Aluminium	......	(7,500,000)	pounds	......
Antimony (existing only only as a constituent (20%) of hard lead)	4,633,036	(6,174,000)	pounds	$301,147
Bismuth (contents of ore mined, but not smelted in United States)	10,891	(no)	pounds	22,871
Copper	748,540,800	(689,045,796)	pounds	95,588,660
Gold	4,090,532	(3,560,000)	troy ounces	84,551,300
Iron, pig	16,563,9[illegible]	(18,009,252)	long tons	219,474,679
Lead	313,553	(276,694)	short tons	27,021,997
Nickel	100,000	(114,200)	pounds	47,000
Platinum	120	(110)	troy ounces	2,340
Quicksilver	43,499	(37,866)	flasks of 75 lbs.	1,783,459
Silver	53,603,000	(54,300,000)	troy ounces	30,672,173
Zinc	176,849	(158,502)	short tons	18,038,598
Total metallic				$477,504,224

The same issue contained a great many special statistical articles by various authors on the various minerals, metals and markets during 1904.

RECENT METALLURGICAL PATENTS.

Smelting Ores in Blast Furnaces.

J. E. Johnson, Jr. (788,044, February 21) purposes to feed separate bodies of ore and fuel into the blast furnace, to smelt the charge and increase the amount of heat available for smelting, and especially for those reactions which require a high temperature, by supplying a blast containing an excess of oxygen and free from moisture. He passes the gases escaping from the furnace through the incoming body of ore and maintains them out of contact with the incoming fuel, the waste gases thus being high in carbon dioxide and low in carbon monoxide and nitrogen.

Treatment of Matte.

Two patents of O. S. Garretson (782,123 and 782,124, February 7) relate to a metallurgical operation which involves the treatment of matte by the converting or bessemerizing process in a blast furnace underneath a column of ore and flux or silicious material. When rich mattes are thus produced, the slag is correspondingly rich and carries off values which should be recovered in order to render the process economical. The inventor proposes to combine the stack furnace for the converting or bessemerizing process with a reverberatory furnace or forehearth which receives the slag from the converting furnace. He supplies sulphur-bearing material to the slag in the forehearth, while subjecting the slag in a shallow layer to heat applied from above the level of the slag.

He thereby produces low-grade matte which combines with metal or matte contained in the slag, a flow of this low-grade matter being maintained back to the bessemerizing or converting furnace.

In his second patent a settling well is provided which receives the slag from the slag-spout of the converting furnace. The matte which settles there is drawn off from time to time, while the slag is treated in the forehearth as before.

Hardening Iron.

F. L. Ramon (784,124, March 7) patents a process of hardening iron, by alloying it with 12 to 15 per cent aluminium. The iron and aluminium are brought to white heat in a crucible; a small quantity of sulphur is then placed on the metals and the contents "are reduced to the molten state by the application of additional heat, so as to be ready for pouring for the formation of the finished product. The said product may be made in an electric furnace or coke furnace, if preferred. By the use of an electric furnace the process will require about one minute; but by using a coke furnace, the same will require from two to three hours." The hardness of the product is increased with the use of additional parts of aluminium, the maximum hardness being obtained by the use of 12 per cent. "The said product weighs about one-half the weight of an equal quantity of steel and is tougher than the same."

Copper.

G. H. Waterbury (783,600, February 28) extracts copper from ore by the following wet method. The pulverized ore is placed in a tank containing a solution of sulphurous acid, to which is added a compartively small quantity of sulphuric acid, and air and steam are forced upward through the ore from the bottom of the tank. When the leaching operation is complete, the solution is drawn off into a precipitating tank containing particles of metal, as aluminium or steel, through which the solution freely circulates. Air and steam are then introduced at the bottom of the precipitating tank and caused to pass up through the copper solution, whereby the copper is precipitated or caused to settle on the bottom of the tank.

Agitating Device.

W. B. Devereux (781,406, January 31) patents an agitating device in which liquids with solid particles are first agitated, then settled and finally decanted. The invention consists, primarily, in combining with a tank a propeller or a propeller pump or other mechanical equivalent of the ordinary marine propeller rotating within the tank at such a depth that the solid particles, after settling, will not interfere with the starting of the propeller, with a series of radial diaphragms placed vertically within the tank, so constructed that the liquids with solid particles contained therein will pass freely through, around, and under them. The purpose of the diaphragms is to prevent rotation of the material in the tank when being agitated without subdividing the tank into separate or independent compartments.

Hascal A. Hogel and Herbert A. Hogel (781,520 and 781,521, January 31) patent a wet process in which the pulverized ore is subjected to the action of dissolving solution and the solvent action is intensified by "atomizing" the mixed mass under the influence of heat. The main point is the atomizing action by means of an aspirator of a certain construction.

Recovery of Volatile Acids from Solutions.

E. R. Hewitt (783,783, February 28) patents an apparatus for recovering volatile acids from solutions, for instance, sulphurous acid from an acid solution containing soluble salts, the method is especially applicable in the leaching of bones for the separation of the bone salts. The apparatus consists essentially of a tank with an open-ended pipe extending into the tank near its bottom, for the admission of live steam, and a curved condensing-chamber surrounding the open end of the pipe, whereby condensation of the steam is induced and the liquid in the receptacle is given a whirling motion. As soon as the liquid becomes hot enough, so that all the steam is not condensed in the body of liquid, the sulphurous acid is carried mechanically by the steam up into a condenser, where the acid separates in the form of gas which is forced upward into a pipe.

Miscellaneous.

T. McDonald (781,615, January 31) patents details of construction of the deflectors employed in connection with blast-furnace-charging devices to distribute the charge. This deflector is made vertically movable, so as to be raised into inoperative position for a part of the charges, and lowered into operative position for other parts of the charges.

T. McDonald (781,293, January 31) patents a cinder-ladle consisting of an outer ladle and an inner thimble, so supported that its bottom is not in contact with the ladle; the space between the bottoms of ladle and thimble is packed with a refractory material, the removable bottom is secured to the thimble by means not in contact with the outer ladle.

BOOK REVIEWS.

Jahrbuch der Elektrochemie und angewandten Physikalischen Chemie.—Berichte über die Fortschritte des Yahres 1903. Edited by Dr. Heinrich Danneel. Halle: Wilhelm Knapp; 930 pages. Price, 26.00 marks.

Larger, more imposing, and later than ever before, we welcome this permanent record of the yearly advances of electrochemistry. It must be said that electrochemistry is moving too rapidly for a reproduction of the work of 1903, at this period, to command the interest which it would have done if published during 1904. It may be put down as a truism, proved by experience, that unless an annual review is published within the limits of the year succeeding the one which it chronicles, that it loses at least one-fourth of its value, and one-half of the interest which it would otherwise command.

We note also the extension of the title from the simple "Jahrbuch der Electrochemie," corresponding to the extension of the title of the "Zeitschrift für Electrochemie," and the change of title of the "Deutsche Elektrochemische Gesellschaft." Here we believe that the interests of concentration of purpose and timely appearance have suffered, that the book loses part of its individuality, becomes unwieldy and necessarily late in appearing. The title is also meaningless; for applied physical chemistry, if there is, properly speaking, such a thing, would refer equally to all the practical operations of analytical chemistry, technical testing, manufacturing chemistry, and much of practical metallurgy. The title is too diffuse, and an attempt to live up to it, in the compass of one volume, necessarily results in imperfect success and incompleteness, not to speak of the resulting delay in preparation and publication.

The part of the book which is not electrochemical contains some most interesting information about the advances in physical chemistry and some of its applications, but does not compare in completeness or in satisfactory treatment with the electrochemical portion. The compilation of electrochemical books published in 1903 would be more satisfactory if it really represented all that were published, instead of merely those which were received by the editor. This list ought to be as complete as possible, in order to correspond with the main electrochemical part of the book.

The best that the reviewer can say of this volume is that it contains a fine abstract of all the advances in electrochemistry in 1903, and in this respect is quite equal in value to its invaluable predecessors. This is not too high praise, and should be sufficient incentive to every electrochemist who reads German to purchase the book.

Pneumatic Pyrometer.

One new type of pyrometer which has found much favor in iron and steel plants, especially for indicating and regulating the temperature of hot blast for blast furnaces, and also in connection with annealing and tempering furnaces, is the Uehling pneumatic pyrometer with Steinbart automatic recorder, which deserves a more detailed description on account of the very ingenious yet simple principle on which its operation is based, and which differs radically from the methods utilized in other types of pyrometers.

The Uehling pneumatic pyrometer is built by the Uehling & Decker Co., of Passaic, N. J. The principle of the instrument may best be explained with reference to Fig. 1. If two apertures, A and B, respectively form the inlet and outlet openings of a chamber C, and a uniform suction is created in the chamber C' by the aspirator D, the action will be as follows: Air will be drawn through the aperture B into the chamber C', creating suction in chamber C, which in turn causes air from the atmosphere to flow in through the aperture A. The velocity with which the air enters through A depends on the suction in the chamber C, and the velocity at which it flows out through B depends upon the excess of suction in C' over that existing in the chamber C, that is, the effective suction in C'. The total suction remaining constant, the effective suction must decrease as the suction in C increases, hence the velocity at which air flows in through the aperture A increases, and the velocity at which air flows out through the aperture B decreases, until the same quantity of air enters at A as passes out at B. As soon as this occurs no further change of suction can take place in the chamber C. If the apertures A and B are of the same size, and the temperature of the air is the same while flowing through both, then equilibrium is established when the suction in C is one-half the suction in C'.

Air is very materially expended by heat. Therefore, the higher the temperature of the air the greater the volume, and the smaller will be the quantity of air drawn through a given aperture by the same suction. Now, if the air, as it passes through the aperture A is heated, but again cooled to a lower fixed temperature before it passes through the aperture B, less air will enter through the aperture A than is drawn out through the aperture B. Hence the suction in C must increase and the effective suction in C' must decrease, and in consequence the velocity of the air through A will increase and the velocity of the air through B will decrease, until the same quantity of air again flows through both apertures. Thus, every change of temperature in the air entering through the aperture A will cause a corresponding change of suction in the chamber C. If two manometer tubes p and q, Fig. 1, communicate respectively with the chambers C and C', the column in tube q will indicate the constant suction in C' and the column in tube p will indicate the variable suction in the chamber C, which suction is a true measure of the temperature of the air entering through the aperature A.

It will, of course, be understood that to embody the above described principle in a practical pyrometer, certain conditions must be strictly fulfilled. Fig. 2 shows a diagrammatic disposition of all the parts combined in the complete instrument. The interior of the pipe e, f, g, h, i from aperture to aperture, together with the branches q and s, constitute the chamber C of Fig. 1. Its inlet from the atmosphere is through the opening a at the bottom of filter I, and its connection with chamber C is through the pipe l.

The aspirator D exhausts into the chamber G, keeping it at a constant temperature 212°. The steam and condensed water, together with the air drawn through the aperture, escape through the pipe t at atmospheric pressure. Opening the valve 6, steam enters the aspirator D, and sucks the air through the tube m, out of the chamber C' and produces a suction, which is kept constant by the regulator H, as shown by the manometer p. By keeping the water in H at a constant height, a perfectly uniform suction is maintined. With a constant suction in C' and cocks 2 and 4 open, air will enter at a, pass through the filter I, where it is purified through the connection b into the "fire tube." It flows forward in the annular space between the two tubes c and f; as soon as it reaches the platinum tube d, which protrudes from the cooler, it becomes heated and enters through the aperture A into the chamber C, at the temperature surrounding the exposed end of the fire tube, which is the temperature to be measured. After passing A, the air flows through the pipe e, f, g, h, into the coil i, where it assumes the temperature of 212°, at which it passes through aperture B, thence by the connection l into the chamber C', from which it is drawn by the aspirator D through m, and discharged with exhaust steam and condensed water.

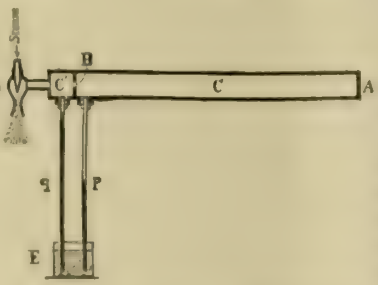


FIG. 1.—PRINCIPLE OF PNEUMATIC PYROMETER.

The branch pipes s and q' connect respectively with the recording gauge L, and the manometer q, which is placed in proximity to the temperature scale, as shown. This combination, therefore, fulfills all the conditions—viz., air is drawn through the instrument by a constant suction. It passes through aperture B at a constant temperature. Aperture A is so located that the air must enter at the temperature to be measured; hence, the indication of the manometer q will vary with the temperature at A, and the latter can be read off directly on the temperature scale placed beside the same.

The pyrometer is used in combination with the Steinbart recording gauge which has certain distinct advantages. There are no springs to get out of adjustment. The only point for friction is the contact of the pen against the paper. The temperature lines are printed on the paper just before passing under the pen, so that no error can occur by any movement of the paper up or down the feeding rollers. The gauge is driven

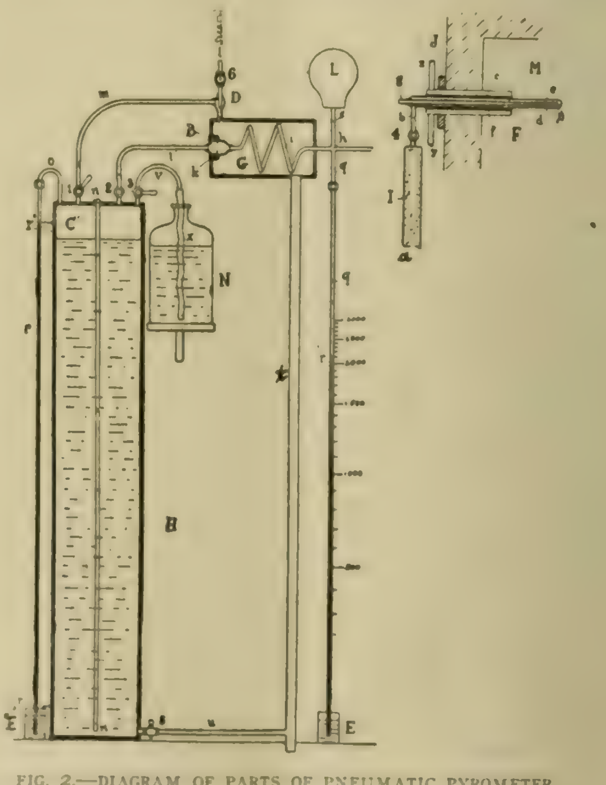


FIG. 2.—DIAGRAM OF PARTS OF PNEUMATIC PYROMETER.

by powerful clock-work actuated by two mainsprings, each capable of keeping the gauge going eight days. Seven hours of the record are always in view, and the whole can be seen by pulling it off the receiving roller. The working of the gauge is not sensitive to vibration; it will work quite well in the vicinity of moving machinery.

On account of these inherent advantages, the instrument has found much favor with practical furnace men, there being now more than 500 instruments in use in blast and annealing furnaces. Of these 30 or 40 are in England, 20 in Austria, 3 in Japan, 4 or 5 in France, and 8 or 10 in Germany, the balance being used in this country. The majority of the blast furnace managers prefer the double form of the instrument which indicates the temperatures of both the blast and gas.

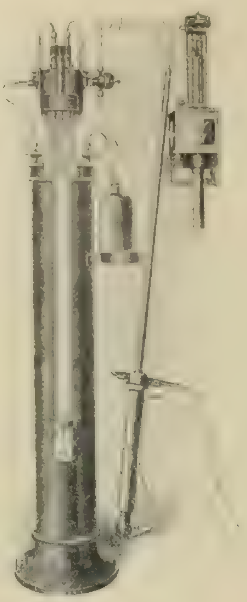
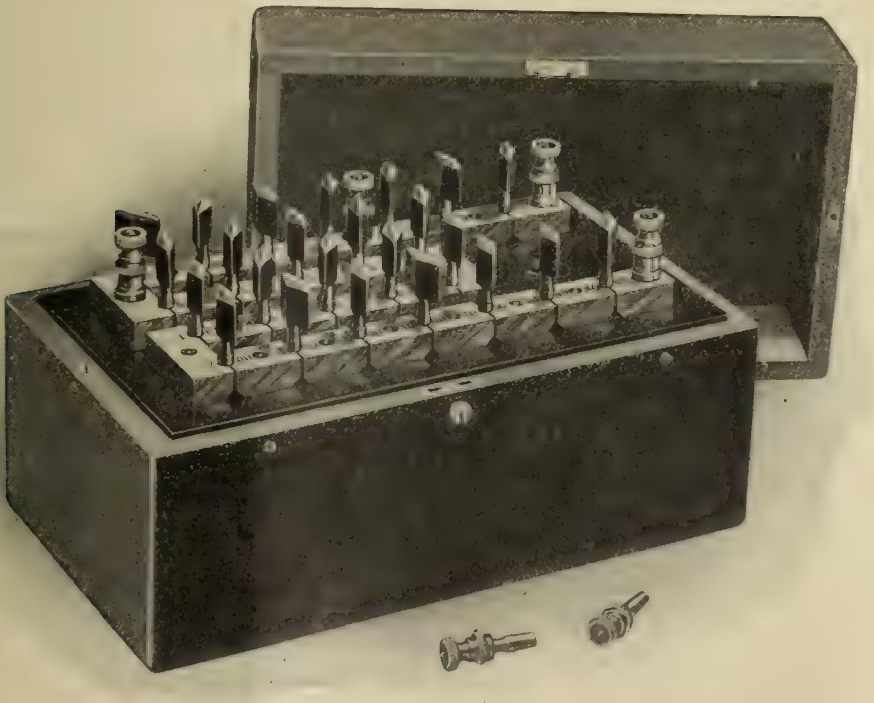


FIG. 3.—PNEUMATIC PYROMETER.

Fig. 3 is an illustration of the "simple pyrometer" with portable fire tube. The recording device is shown at the right side and is firmly connected with the main instrument on the left of the illustration. The portable fire tube is connected with the regulator by means of a rubber hose, so that the temperature may be measured at various points within a circle of about 30 meters.

Wheatstone Bridge Resistance Box.

The most compart and the most easily portable form of Wheatstone bridge is undoubtedly the plug contact form in which plug resistances are substituted for the stretched wire of the meter bridge. In the following we give a description of the "University of Texas Resistance Box," as built by the L. E. Knott Apparatus Company, Boston, Mass. It was designed with a view of obtaining the highest degree of precision of which the plug form of Wheatstone bridge is capable. The accuracy of a resistance box in practical use depends upon a number of conditions of which the mere accuracy of adjustment is but one. In the University of Texas Resistance Box, the object was to provide permanency of calibration in every detail.



UNIVERSITY OF TEXAS RESISTANCE BOX.

The coils of this box are:

Rheostat arm,				
1	10	100	1000	ohms
2	20	200	2000	"
3	30	300	3000	"
4	40	400	4000	"
Ratio arms,				
1	10	100	1000	"
	10	100	1000	"
Total			13,331	"

The one-ohm ratio coil can be thrown into either arm desired by means of one of the traveling plugs supplied with the box. An additional coil of 10,000 ohms is provided if desired. This coil can be thrown into either the rheostat arm or the nearer ratio arm, or can be cut out altogether by means of the ordinary bridge plugs; or by the traveling plugs it can be thrown into the other ratio arm. It is therefore available for a variety of purposes.

The theoretical range of the bridge would therefore be from

1-10,000th ohm to 111,100,000 ohms;

or without the additional 10,000 ohm coil,

1-1,000th ohm to 11,110,000 ohms.

The actual working range of any plug box is, of course, far inside even of the narrower of the above limits; being determined by plug contact resistances on the one hand, and insulation provisions on the other.

The coils of the rheostat arm are adjusted to 1-10th per cent, and those of the ratio arm to 1-20th per cent. The coils are made of manganin wire, which has a temperature coefficient practically zero over the limits of laboratory variations. Moreover, these coils are wound anti-inductively on highly insulated metal bobbins of special construction and open at both ends, which gives thorough ventilation from inside and outside of coils. In this way the heat which is developed in the coils is easily carried off.

The coils are mounted in a new manner which makes each independent of the others, so that in case of accident it can be removed, readjusted and replaced without disturbing the rest. In this method of mounting, each coil is elevated on short ebonite pillars, giving enormous insulation resistance at the interior of the box, which is not so easily accessible for cleaning.

The arrangement of the connection blocks was adopted after careful consideration of the dial patterns and other group arrangements as being simpler and capable of more perfect cleaning and insulation. It is the modern double-bent arrangement, which brings the X terminals at the right-hand end of the bridge, so that the box can be conveniently joined to another box, such as a sub-divided ohm, or to a slide wire. This arrangement, moreover, gives connections to the "unknown" which are of equal length and resistance. It also results in the most symmetrical and convenient grouping of the coil values.

The shanks of the bridge plugs are of special shape, which prevents loss of fit by wear of the plugs or holes—a small but very important detail. The shanks are of large diameter and, being most carefully fitted, give contacts of very low resistance. The taper of the plugs is carefully chosen, so as to prevent wedging the blocks apart and yet to allow the insertion of the plugs without undue forcing.

An important new feature of the box is the addition of a heavy metal girder frame securely screwed to the substantial ebonite top. This keeps the ebonite absolutely rigid and allows of perfect alignment of the brass blocks, and of fitting the plugs to their holes perfectly and permanently. Between the ebonite and the sides of the box is placed a felt gasket, making the box dust-tight and practically air-tight.

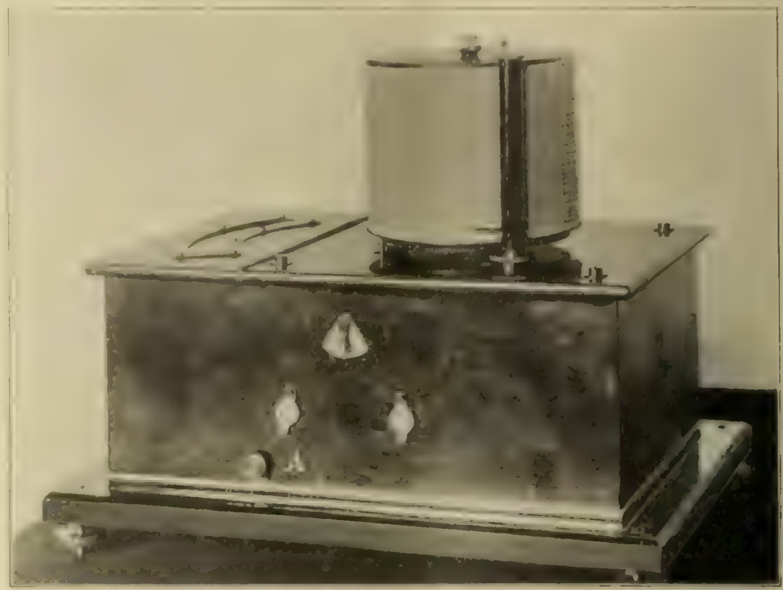
To keep the high initial insulation resistance permanent, reasonable care is, of course, necessary in any resistance box; to make this easy, special facilities for cleaning are provided, and to this end also acts a wooden box cover which is always placed on the box when it is not in service.

Recording Electric Pyrometer.

Many metallurgical and chemical processes, in order to be as efficient as possible, depend on exactly maintaining the required temperature. Maintenance or regulation of temperatures requires, of course, the possibility of measuring them exactly. As long as these temperatures are relatively low, the ordinary thermometer is sufficient. But the problem is of no less importance for the reactions at higher temperatures, and an enormous amount of ingenuity, hard work and good money has been spent in recent years for developing exact pyrometers.

The electric pyrometer using a thermo-couple has been known in principle for a very long while, since Pouillet made heat determinations on this principle as early as 1836. The electric pyrometer was later brought into prominence by Prof. Henry LeChatelier, and was introduced in a more practical form by Prof. Robert Austin, who provided it with a photographic recording adjustment. This latter point, however, has for a long while been a distinct disadvantage in the use of the electric pyrometer for many metallurgical purposes. When the record was taken by photography it had to be developed and fixed before it could be seen, so that the record was practically twenty-four hours behind the time when it was taken. Now, this is, of course, undesirable, since, after twenty-four hours have elapsed it is too late to make changes in the operation of the furnace which could have been easily made if the attendant had the record always in front of him, showing him the temperature at every instant.

In view of these considerations a new pyrometer made by Messrs. Queen & Co., of Philadelphia, is very interesting, since the former disadvantages of the electric type of pyrometer have been successfully overcome in this instrument. It makes a continuous record of temperatures up to 3000° F. It consists of a sensitive D'Arsonval galvanometer, the recording mechanism and the thermoelectric couple. The thermoelectric couple consists of the regular platinum and platinum rhodium wires, 3 feet long, although a couple of any length will be furnished if desired. The D'Arsonval galvanometer measures the electric current produced by the heating of the junction of the thermoelectric couple. It was always perfectly understood that the electric pyrometer is a most delicate and accurate instrument, but the objection was sometimes raised that this delicacy renders the instrument occasionally difficult to use, for instance, for the reason that it is sensitive to vibrations. The galvanometer used in the Queen instrument is specially designed for its particular work. It is extremely sensitive, absolutely dead beat and so constructed that vibrations will not effect it.



RECORDING ELECTRIC PYROMETER.

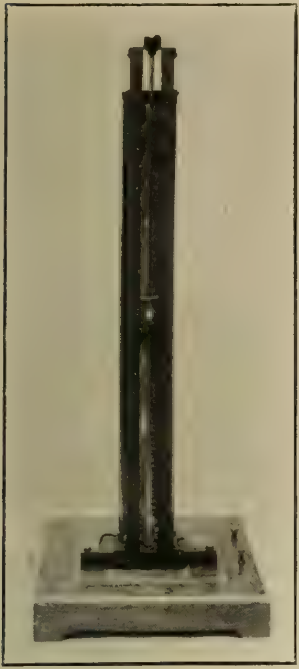
The recorder is so arranged that the pen traces on the chart the exact movements of the galvanometer pointer. The cylinder on which the chart is fastened revolves once in twenty-four hours. In the clock that drives the cylinder an electric contact is made every minute. This intermittent current, by passing through an electromagnet, clamps the pointer of the galvanometer, which in turn operates a relay and causes the pen to move either up or down, until it coincides with the reading of the galvanometer, when the pointer is released and swings entirely free until the next contact is made, thus the recording device in no way impairs the accuracy or sensibility of the pyrometer. It will thus be seen that in the Queen recording pyrometer all the well-known advantages inherent in the electric pyrometer have been reserved, while the former disadvantages of this type of instrument have been successfully overcome.

Standpipe Rheostats.

A series of rheostats has recently been designd by Messrs. Queen & Co., of Philadelphia, Pa., consisting of a continuous length of wire wound spirally on an iron standpipe, about 6 inches in diameter, of a length depending on the total resistance and the current carrying capacity required.

The wire is insulated from the iron pipe to withstand an alternating potential of 1000 volts. The pipe is closed at the bottom, and by means of a small pipe, on either side of the base, a stream of water can be introduced for cooling purposes. If the rheostats are not used at their maximum capacity, water is not necessary; when only used at their maximum capacity for a few minutes at a time, the pipe filled with water alone is sufficient; running water is recommended only when the rheostats will have hard continuous service at their maximum rated capacity. The standpipe is mounted on a hardwood base, provided with castors, so as to be moved about the laboratory easily. Mounted parallel to the upright pipe is an angle-iron guide, which carries a slider, to which are attached phosphor bronze springs, which, in turn, make contact against the wire on the pipe. All the parts are insulated except a rod in the center of the angle-iron guides, which is connected with the phosphor bronze springs by means of a split brass sleeve in the wooden handle. This gives a very smooth, even motion, and positive contact is made on every convolution of the wire, which allows adjustments of the current by almost infinitesimal steps.

These rheostats are especially suitable for laboratory work in electrolysis, where it is necessary to secure fine graduations of both current and potential. To secure a potential regulation the rheostat is supplied with two sliders, the terminals of the main conductor are connected with the main source of supply, and the wires to the apparatus are connected to the two sliders. By putting the sliders close together, the difference of potential is very small, whereas, by sliding them farther apart, it can be increased up to any desired point.



STANDPIPE RHEOSTAT.

Centripact Screen.

In our last issue we described in an article on the concentrating plant of the Green Mining & Milling Co., of Silverton, Colo., a novel and interesting screening device, the Centripact screen of the Traylor Engineering Co., of New York City. By a mistake of the printer, however, Fig. 5, on page 125, was reversed. We therefore give again the two illustrations of the screen. Fig. 1 shows the double-unit Centripact screen for dry work. As was explained in our last issue, the screen is approximately saucer-shaped, and is mounted on a vertical shaft which revolves, while simultaneously successive vertical impacts are given to the shaft and screen, so that the particles on the screen are subjected to two forces—the centrifugal force tending to drive the particles towards the periphery of the screen, and an upward impact tending to make them leap from the surface.

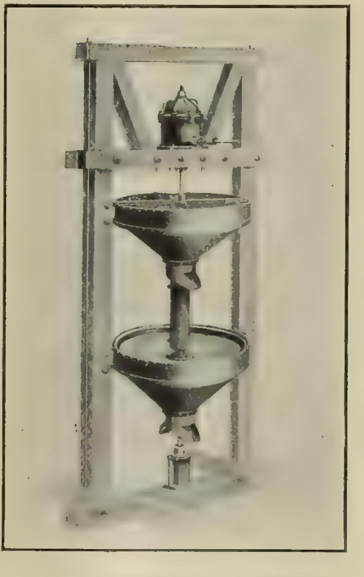


FIG. 1.—DOUBLE-UNIT CENTRIPACT SCREEN FOR DRY WORK.

Fig. 2 is a section of a centripact screen unit for wet work. The most essential feature of this type is the provision of a sheet-steel pan conforming in shape to the screen and supported beneath the screen cloth about ¾ of an inch from the same. The water passing through the screen is caught in the pan and flows outward over it. Owing to the action of centrifugal force, the surface of the water assumes a form corresponding to that of the screen and pan. There is thus a layer of water immediately under the screen, and with each impact, the water is projected upward against the screen, serving to clean the meshes and at the same time thoroughly irrigating the material to be screened.

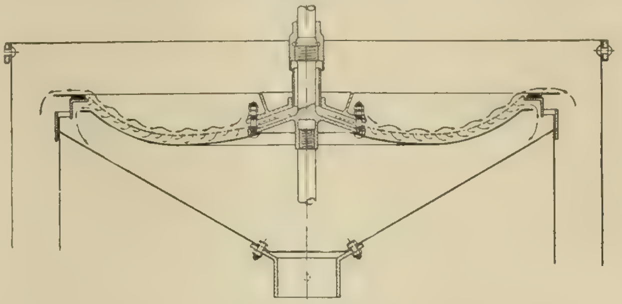


FIG. 2.—SECTION OF CENTRIPACT SCREEN UNIT FOR WET WORK.

Industrial Notes.

The J. T. Baker Chemical Co., in Easton, Pa., are producing a full line of high- grade chemicals for technical and analytical work. Mr. J. T. Baker, formerly of the Baker & Adamson Chemical Co., is president of the new concern. The plant is in Phillipsburg, N. Y., and comprises several new and commodious buildings. A fully equipped laboratory is maintained in a separate building in which the offices of the concern are located.

We have received from *The Fossil Flour Co.*, of 229 Pearl Street, New York, a folder on the use of fossil flour in rubber compounds, as a "rubber drug;" when added to any zinc or whiting compound, it is stated to increase the strength, increase the bulk and reduce the cost. Infusorial earth may also be used to good advantage as a heat insulating material in electric furnaces, as pointed out on page 55 of our February issue, and on page 139 of the present issue.

A clearly written and well illustrated pamphlet of the *Westinghouse Electric & Mfg. Co.* (Folder No. 4032) deals with wattmeters and how to read them. Circular 1099 deals with bipolar motors, type R, for direct-current circuits, circular 1097 with type K motors, direct current, series wound for crane, hoisting and similar service.

The *De La Vergne Machine Co.*, of New York City, has recently issued two nicely illustrated folders. One deals with economical gas and oil engines: "The gas plant of small capacity differs widely from the small steam plant, in that it enjoys an efficiency equal to the large gas plant, and not only saves the losses due to electrical transmission, but also attains an efficiency far superior to the steam-driven prime movers of the large central stations supplying the electricity. The development of small power high efficiency gas engines designed to utilize natural or producer gas is bound to react eventually, causing an increase in the number of small isolated plants at the expense of large installations." Several Koerting gas engines and Hornsby-Akroyd oil engines, built by the De La Vergne Machine Co., are illustrated. The other folder refers to refrigerating and ice-making machinery.

Electricity in Copper Mines.—The Mammoth Copper Mining Co., of Salt Lake City, has just placed an order with the Westinghouse Electric & Manufacturing Co. for a large addition to their electrical equipment. A total of nearly 800 horse-power in induction motors of different sizes, together with a motor generator set, transformers, switchboard and three mining locomotives, is included in the order. The motors, two of which have a rated capacity of 200 horse-power each, and five of 50 horse-power each, will operate on three-phase 2000-volt circuits. This order is merely indicative of the progress being made in the use of electric power in mining operations.

Flange Joint Gaskets.—The history of flange joint packing has been, of necessity, closely identified with and influenced by that of all steam service. As the temperatures and pressures have increased, the earlier forms of flange gaskets have become less and less efficient, necessitating continual improvements to meet the severity of modern requirements. The advantages of asbestos for this purpose are described in a pamphlet "Flange Joint Gaskets," issued by H. W. Johns-Manville Co., 100 William Street, New York.

The *Charles E. Sholes Co.* announces the establishment and equipment of industrial laboratories for chemical research and analysis at 164 Front Street, New York, with Dr. J. E. Teeple, formerly instructor at Cornell, as director.

Digest of U. S. Patents.

PRIOR TO JULY, 1902.

Compiled by Byrnes & Townsend, Patent Lawyers, National Union Building, Washington, D. C.

ELECTRIC SMELTING AND REDUCTION PROCESSES.

(Continued from page 128.)

430,457. June 17, 1890. Thomas L. Willson, Brooklyn. New York.

Protects carbon electrodes and vessels by the introduction of a reducing gas, such as hydrogen or illuminating gas. The furnace shown is a carbon crucible resting on a carbon plate having a terminal, surrounded by a wall and covered by a slab having a vent hole. A depending carbon pencil passes frictionally through an opening in the slab. The carbon pencil may be tubular, for the introduction of gas, or a separate pipe of refractory material may be employed. The tubular electrode may be built up of four plates arranged in a rectangle, luted at the joints with a paste of carbon dust and molasses and held together by metal clamps and carbon dowel-pins. The conductor consists of spaced copper wires leading to a bar terminal. Produces boron, silicon, calcium, chromium, titanium, iron, steel, etc. For the production of aluminium bronze, two parts of broken copper or copper wire are placed in the bottom of the crucible and covered by one part of corundum. The material is smelted by an arc, from four to five pounds of material requiring 20 amperes at 50 volts. The direct current may be passed in either direction, preferably from the crucible cathode to the depending electrode, to prevent rapid consumption of the latter. The copper is melted and vaporized and the copper vapors circulate within the crucibles, condensing against the relatively cool sides and streaming down through the corundum. The operation requires from 15 minutes to 2 hours, the deoxidizing gas being supplied throughout. By partially melting down the fireclay cover, an aluminium silicon bronze is produced. By smelting iron oxide containing corundum, aluminium steel is produced. Aluminium bronze may be made from a charge of alumina and copper sulphide, *e. g.*, redruthite, the eliminated oxygen and sulphur being combined with a hydrocarbon gas.

486,575. November 22, 1892, Thomas L. Willson, Leaksville, N. C.

Production of aluminium bronze by electrical reduction. Alumina, in a state of fine sub-division and preferably in a porous condition, is thoroughly impregnated with a hydrocarbon liquid such as coal-tar, heavy hydrocarbon oil or paraffin. The hydrocarbon is heated to its boiling point and the alumina is added little by little with constant stirring, until the hydrocarbon is completely absorbed. When precipitated alumina is used the proportions are about 22 pounds of the alumina to each gallon of tar. Reduction is effected either by heating the product as above obtained in an incandescent electric furnace, or in an arc furnace of the Siemens type, the mixed alumina and tar being mingled with broken carbon in the former case. In either case granulated copper is supplied in sufficient proportion to absorb the reduced aluminium with the production of aluminium bronze. The statement is made that sodium chloride may replace the hydrocarbon as a reducing agent, chlorine escaping and the reduction of the alumina being effected by the sodium.

491,394, February 7, 1893, T. L. Willson, Brooklyn, N. Y.

A furnace for the reduction of refractory oxides comprises a carbon crucible resting upon a carbon plate and connected as the anode of an electric circuit, the cathode being a carbon pencil projecting into the crucible through a luted cover. For the production of aluminium bronze the charge consists of finely divided and preferably precipitated alumina, and finely powdered carbon, the oxide and the reducing agent being either mixed or fed alternately into the furnace. A suitable quantity of copper is added for the purpose of taking up the molten aluminium as it is liberated. The temperature of the furnace being above the vaporizing point of copper, this metal is constantly distilled and condenses in the upper portion of the furnace, the liquid copper flowing back through the charge and extracting from it the particles of aluminium. Aluminium steel is said to be produced by the simultaneous reduction of iron ore and corundum or by reducing an aluminium compound in the presence of fragments of iron. The process is stated to be applicable to the reduction and alloying of other highly refractory elements, such as boron, silicon, calcium, chromium, titanium, etc.

492,377, February 21, 1893, T. L. Willson, Leaksville, N. C.

The furnace comprises a carbon or graphite crucible, into which depends a carbon pencil, the crucible and pencil constituting opposite terminals of the electric circuit. Carbon plates resting upon the masonry walls enclosing the crucible serve to close the furnace. It is stated that when aluminium bronze is prepared by the reduction of alumina in presence of copper, and in all other reactions in which a molten mass is present in the furnace, there are violent fluctuations in the resistance of the furnace, which occur by reason of the ebullition of the fused bath. The present invention consists in suppressing this ebullition by the use of carbon in sufficient proportions to prevent the formation of a fused bath; under these conditions the operation proceeds quietly and with relatively high economy. The carbon may be added in the form of a hydrocarbon, the mixture being subsequently coked, or in the form of finely divided coke or other material. For the reduction of alumina it is found that carbon equal to 15 per cent by weight of the mixture is sufficient to suppress the fused bath and accomplish the above results. The aluminium may be recovered by introducing an alloying base metal, as copper, into the furnace, or it may be recovered as aluminium carbide in presence of the excess of carbon. The furnace may be modified as desired, for instance, by striking the arc between two carbon pencils in close proximity to the charge, but incandescent furnaces are excluded. It is stated that the process is applicable to the reduction of many metals other than aluminium, and the following statement constitutes the first patent reference to the electric furnace production of calcium carbide:

"For example, I propose to apply it (the process) for the treatment of refractory compounds or ores of metals, not necessarily for the production of the metals themselves, but for the production of other compounds thereof. For example, I have already employed it for reducing calcium oxide and producing calcium carbide."

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Boston Meeting of the American Electrochemical Society.

That the Boston meeting of the American Electrochemical Society—a full report of which is given in this issue—would be a most enjoyable and profitable affair, long to be remembered by all attending it, was a priori certain to everybody who knows the spirit of the city of the Massachusetts Institute and Harvard University, and who knows the pluck and energy of the local Boston members, with Dr. Walker and Dr. Talbot as chairman and secretary. The meeting could not turn out anything but a success. The only question was how many out-of-town members would attend. Their number was about forty, and may seem small in comparison with the total membership, but we have attended general meetings of far older and larger national engineering societies which were afterwards voted a great success, and yet had a smaller attendance than the Boston meeting. Nevertheless, the question is decidedly open for discussion, what can be done to attract a larger proportion of the general membership to future meetings.

It is an old saying that two general meetings a year are one too many, and this argument seems to be substantiated by the fact that most national engineering societies hold only one general meeting a year. However, in the case of the American Electrochemical Society this argument is refuted by the quantity and quality of papers which its members have been able to present twice a year at each meeting since the formation of the Society. On the other hand, the fact remains that the American Electrochemical Society has enlisted in its membership a great many members of the American Institute of Electrical Engineers, the American Chemical Society and the American Institute of Mining Engineers, and these three Societies are to hold general meetings at different places either in May or June, thus closely following the Boston meeting. The present strength of electrochemistry rests on the fact that it deals with a borderland of sciences and arts, full of virgin soil, and as President Eliot, of Harvard University, remarked in his delightful little speech at the reception in Harvard Union, it is on such borderland that most important developments generally take place. It is very similar with the development of nations.

The reason for this is evident; the separation of sciences and arts is not natural, but inherent in the historical development of mankind; each science and each art represents the results obtained by pushing a single point of view as far as it will go. Thus any science and any art, when getting fully developed, tends to become one-sided. The only possible remedy is then evident. Electrochemists, while apparently specializing, are really uniting chemical, metallurgical and electrical sciences and engineering. It would thus appear to be decidedly a step forwards in the work of the American Electrochemical Society if it would try to coöperate more closely with its national sister societies. We have no idea of a "merger" or of giving up in the least the identity of the Electrochemical Society. But an attempt to hold, at least, once a year a joint meeting with one of the sister societies, should bring good results; one year it might be with the chemists, another year with the mining engineers, and another year with the electrical engineers. Such joint meetings would strongly tend to strengthen the cause in which the Society is enlisted. Moreover, a larger proportion of the membership would naturally be attracted to such meetings.

The programme of papers at the Boston meeting was excellent, although the distribution of the papers over the three sessions was somewhat unbalanced, the last session being too heavily loaded with splendid material. In one point the programme had a novel feature, which cannot be too strongly recommended. This is the presentation of various lectures illustrated by experiments and demonstrations, and covering a somewhat large subject. An instance is Dr. Whitney's most interesting lecture on colloids. While perhaps only a small part of this lecture contained hitherto unpublished information, yet it is certain that to most of those present the whole subject was a novel one, and that to all who listened to this lecture the lucid exposition of the subject offered plentiful food for thought. To read the printed paper is no substitute for listening to such a lecture. Thus the argument which one often hears that it is easier and more convenient to read the papers either in abstract after the close of the meeting in the engineering journals, or in full after publication in the Society's Transactions, breaks completely down with respect to such lectures with experimental demonstrations. By seeing for himself how a process takes place, one gets a far clearer and more intimate acquaintance with its nature than by simply reading a description of the process. Lectures of this kind have been in the past a regular feature of the annual conventions of the German Bunsen Society, and have contributed much towards their success. If the members of the American Electrochemical Society learn that they will have an opportunity to listen to several lectures of this kind at every general meeting in future, this will act more than anything else to attract a large attendance. For a joint meeting with another society it would not be difficult to select subjects which should be of interest to everybody.

◆

By-Products.

We have made repeated references in these columns to the changes in American metallurgy in the past century. It has always been our endeavor to view these changes in their broadest aspects, and to point out how the interaction of industrial forces results in equilibrium from change. Fundamentally the doctrine of evolution defines the course of these movements. Growth from the simple to the complex is the normal growth for any organism. It is thus but natural to find in all industries a tendency to increase the number of products, and especially by-products made from what would otherwise be waste material in the main process. Such a change is the [illegible] from simplicity to complexity of products. The creature that can use several things for food has a better chance for life than the creature dependent on only one kind of food. Examples of this phase of industrial evolution are seen everywhere in America. The boast of the great Western beef-packing establishments that they sell everything but the intangible squeal, is a trite commonplace of the daily press. Their profits are in their by-products. The Standard Oil Company transports and refines the crude oil cheaply, and sells besides its illuminating oil a variety of products ranging from naphtha to petroleum coke. The millions it disburses in dividends and investments from income are sustained by its large output and varied sources of profit.

About the prettiest example of by-product working can be seen at the works of the Pennsylvania Salt Company at Natrona. Here Rio Tinto pyrites is roasted to produce a sulfurous gas used for the manufacture of sulfuric acid. The cinder from this operation is mixed with common salt, and given a chloridizing roast to render the copper soluble. This copper is then leached out, and the leached "blue-billy" (nearly a pure oxide of iron) is sold to the neighboring iron or steel plants and used as an iron ore. The whole scheme is worthy of emulation at other places. A good deal of good iron in the form of iron silicate is going over the dumps at Rossland, Sudbury and Ducktown. Under favorable conditions this iron might have been brought to the center of consumption and used as an iron, copper and sulphur ore. The General Chemical Company, at Pulaski, Va., is developing this same treatment for copper-bearing pyrrhotite ores of Virginia. The reduction of bituminous coal to coke with the production of gas, tar and sulfate of ammonia is increasing in this country, although it never has had the vogue here it has in Germany. Despite its greater capital cost the by-product coke plant will displace in time the old bee-hive oven, as capital becomes more plentiful. The up-to-date brass mill can only meet competition by having special machinery to work up into specialties, like eyelets, etc., waste sheets which otherwise would have only a scrap value. In Germany one of the great causes for the popularity of the basic Thomas converter is the fact that there is a ready market as a fertilizer for its phosphorus-carrying slag. The electrolytic refining of copper has saved enormous sums to the world and reduced the cost of making copper two or four cents a pound. Many other examples could be cited, but these suffice to show the importance of the subject. The ramification of industry and the utilization of waste are logical events in our industrial developments.

Of course, the by-product business can easily be overdone and carried to extremes. If a large structure fails for improper design its failure causes more destruction than a smaller structure would. So an immense industrial organization planned to create by a *fiat* many co-ordinated factories, springing full-grown in the wilderness, fails from unwieldiness, and great is the fall thereof. A well-known ill-starred Western metallurgical and power company was all right in theory, but it never made much money as first organized. Sir Joshua Reynolds, the great English painter of the eighteenth century, when asked about the secret of his mixing paints replied, "With brains, sir!" By-products should also be mixed "with brains, sir." Some prosaic people think that investments should pay interest in coin of the realm and not in "hot air." Perhaps the greatest application of the by-product proposition can be made by the trusts. Within a great corporation proper co-operation allows of many "intercompany" economies. Stability of the trust is such that improvements and developments which could not be justified in a concern that "could not see itself ahead in ore contracts over two years," can be legitimately undertaken. In this case, paradoxical as it may seem, stability is the cause of progress. The advantages of diversified production are obvious. There is an increase in receipts without a corresponding increase in administration expenses. This alone makes for increased net profits. The concern is rendered independent of the market for any one commodity. It obtains its raw material for the by-product free on the spot, and (as in the case with the European zinc smelters) is often relieved of the expense that would be otherwise incurred in disposing of a troublesome waste material. The outside profit is always welcome, and doubly so when it helps tide over hard times and meet competition. "He who maketh two blades of grass grow where grew one before deserveth well of mankind." By-products do exactly this, and honor is due him who intelligently devises means to develop new ones. He is not alone helping himself, but, by the creation of wealth, he is helping the world, and, according to Mr. Andrew Carnegie, this is what makes the true aristocrat.

The Definition of Electrode.

The term electrode is due to Faraday. It was first used in article 662 of the seventh series of his "Experimental Researches." Faraday's words are as follows: "The poles, as they are usually called, are only the doors or ways by which the electric current passes into and out of the decomposing body; and they, of course, when in contact with that body, are the limits of its extent in the direction of the current. The term has been generally applied to the metal surfaces in contact with the decomposing substances; but whether philosophers generally would apply it to the surface of air and water, against which I have effected electrochemical decomposition, is subject to doubt. In place of the term pole I purpose using that of electrode (electron and hodos, a way), and I mean thereby that substance, or rather surface, whether of air, water, metal, or any other body, which bounds the extent of the decomposing matter in the direction of the electric current." The electrode is, therefore, defined as the boundary surface between a conductor of the first class (metallic conductor) and a conductor of the second class (electrolytic conductor). The characteristic property of an electrode is that when a current passes, a chemical change takes place at the electrode, this change being either reduction or oxidation. The latter qualification is necessary, since a chemical change also takes place at the boundary surface of two electrolytes, but in this case it is neither reduction nor oxidation. The fact that somewhere in an electric circuit reduction or oxidation occurs is, however, not sufficient evidence that there is an electrode at that place. In an electric furnace for making ferro-alloys the charge generally consists of oxide of iron, oxide of the metal to be alloyed with iron, and carbon. The chemical action which takes place throughout the furnace is reduction of the metallic oxides and oxidation of the carbon. But in this case the oxidation and reduction take place simultaneously at every place. On the other hand, in a circuit containing an electrolyte with two electrodes, the oxidation takes place at the anode and only there, the reduction at the cathode and only there, there being neither oxidation nor reduction at any other place within the electrolyte. Electrolytic action is thus always characterized by the fact that there is at the one end of the electrolyte reduction, at the other end oxidation.

Faraday's coining of the name electrode from electron and hodos, the way, becomes especially interesting if we consider, in view of the modern electronic theory, what happens at an electrode. Let us consider, for instance, the discharge of chlorine gas at the anode from a chloride solution. The change is from a chlorine ion—*i. e.*, a chlorine atom combined with a negative electron—into a chlorine atom. What happens within the electric circuit is, therefore, that a negative electron gets free from the chlorine ion and passes as a free electron into the metallic conductor. In metallic conductors the electrons are free, in electrolytic conductors they are combined with atoms or radicles in the form of ions. An electrode is, therefore, the door through which the electrons pass out of the combined state into the free state, or the reverse. We may also state the same idea by means of a familiar analogy, originally due, we believe, to Mr. William Stanley. In metals the electricity is transported along a row of men with shovels, each man shoveling the electrons to his next neighbor; in electrolytes the electrons are placed in constant quantities into wheelbarrows, and are carried in the same from one end to the other. The electrode is then the place where the electrons are loaded into the wheelbarrows, or where they are taken out of them.

Whenever there is electrolytic action, the term electrode is properly used for the terminals through which the current is introduced into the electrolyte, whether the latter is a solution or a fused bath. In recent years, however, with the growth of the electric furnace industries it has become customary to speak of electrodes where there is no electrolytic action; thus the carbon or graphite terminals in calcium carbide furnaces, in the Héroult steel furnace, in ferro-alloy furnaces, are often called electrodes. This is an improper use of the term—just as improper as if in the case of an ordinary incandescent lamp the leading-in wires of platinum with which the carbon filament is connected would be called electrodes. (On the other hand, with the Nernst lamp they are electrodes, since the filament is a solid electrolyte.) In some cases, however, we may have electrolysis in electric furnace work, where it is not wanted. An example is the observation of Mr. W. S. Horry, of the Union Carbide Company, that a higher frequency of the alternating current has a distinct beneficial effect on the production of calcium carbide, the explanation being that the furnace charge undergoes electrolytic decomposition if the frequency is too low, which means a diminished efficiency. In this case the graphite rods are simply terminals for the introduction of the current—as far as the desired effect, the production of calcium carbide, is concerned—but they are electrodes with respect to the undesirable decomposition.

Notes.

American Electrochemical Society.—At the meeting of the Board of Directors, held on April 1 in Philadelphia, the following gentlemen were elected members: Daniel McMaster, Rumford Falls, Me.; Arthur D. Little, Boston, Mass.; Marten Hedlund, Hallstahammar, Sweden. The names of the following gentlemen will come up for election at the May meeting of the Board of Directors: W. H. Kimball, Portland, Me.; A. S. Ramage, Detroit, Mich.; J. M. Woodward, Washington, D. C.; C. A. Hansen, Madison, Wis.; John Woods Beckman, Tarentum, Pa.

Lehigh University.—A committee has been formed to raise a fund of $80,000, for the purpose of erecting and equipping a Drown Memorial Hall on the campus of the university, to the memory of the late president, Thomas Messinger Drown, who, during the last years of his life, had wished to add to the equipment of the university a building that should be devoted to the social life of the students.—The senior students of metallurgical engineering are assisting Prof. J. W. Richards in the metallurgical laboratory in a series of important determinations concerning the specific heats, up to high temperatures, of the important metallurgical materials and important ores, such as fire-clay, silica, limestone, alumina, iron ores, zinc oxide, etc. A collateral investigation is being made of the latent heats of fusion and specific heats in the molten condition of some important metals which have not before been studied.—The course in electrometallurgy has now laboratory facilities, by the equipment of an electrical laboratory at a cost of about $1,000, rendered possible by the generosity of Mrs. Eckley B. Coxe, together with the liberality of the Board of Trustees. The chemical and metallurgical students are now given laboratory practice, illustrating electrometallurgical principles. To the Museum of the Metallurgical Department have recently been added electric furnace ferro-alloys, made by the Heroult process, and electric furnace steels, made in the Kjellin induction furnace.

University of Wisconsin.—We have received Bulletin No. 104 of the University of Wisconsin, College of Letters and Science. The Summer Session and the Summer School for Artisans begin Monday, June 26, and close Friday, August 4. Among the summer courses there will be courses in general chemistry, qualitative analysis, quantitative analysis, chemical preparations and research work in organic chemistry (Prof. Lenher), and courses in electrochemistry, advanced electrochemistry, physical chemistry, advanced physical chemistry, research work in physical chemistry and electrochemistry, and a course in the preparation of typical organic compounds (Prof. Kahlenberg). Letters are to be addressed to the Registrar, W. D. Hiestand, University of Wisconsin, Madison, Wis.

A Course for Chemical Engineers at Columbia University.—In the current issue of the *School of Mines Quarterly*, Dr. Edmund H. Miller gives an account of a four-year course which is to be started at Columbia University for chemical engineers. The problem of the proper training of chemical engineers—in contradistinction to chemists—has repeatedly been discussed in our columns; we may refer the reader to Mr. J. B. Herreshoff's American Chemical Society paper and Mr. W. M. Sanders' communication, printed on pages 171 and 292 of our volume II, respectively. Columbia University is to be highly congratulated on the plan of starting a special course for the training of "a kind of chemist who is not being produced; a man who combines, with a solid knowledge of chemistry, considerable familiarity with both mechanical and electrical engineering; a man who is able to look at any proposition or process from the standpoint of the engineer as well as from that of the chemist." We will give a longer account of Dr. Miller's paper in our next issue. The course will go into effect in September, 1905; it is based on the same entrance requirements as those in the School of Mines and School of Engineering, and leads to the degree of chemical engineer.

Vanadium and Titanium.

By Hans Goldschmidt, Ph. D.

"Many are called, but few are chosen," may well be said of the metals which have been tried for alloys, particularly alloys for special steels. The steels must first be subjected to the most varied tests and refinements before a definite result can be spoken of, and before the properties of a new material are sufficiently understood with regard to its relations with iron. Not only the substance to be added is of importance, but also the proportion in which it is to be added, the other constituents present in the alloy, the impurities, and the treatment of the alloyed steel as finally produced.

It is easy to realize how difficult it is to obtain clear results, when the fact is borne in mind that small variations in the composition of the alloy—which can be detected only with difficulty, if at all—may produce great differences in the properties of the final product. All the elements used in steel alloys, such as carbon, manganese, chromium, and nickel, as well as those which are present more or less accidentally and should be regarded as impurities, like sulphur, phosphorus, and silicon, have a good or bad influence upon the finished product according to the relative proportions in which they are present. Now, even with the few elements just mentioned, the mutual influences of the various amounts added are almost endless, and their investigation, even for these well-known substances, has taken place only within comparatively narrow limits; by the introduction of a new element the difficulty of the investigation is greatly increased, and to make the right choice is an "opus infiniti laboris."

When manufacturers of special steels have once arrived at satisfactory results, they must exercise the utmost care, not only to follow exactly the recipe in the manufacture, but to use always exactly the same starting materials. Manufacturers of special steels have often found that although they always followed exactly all points of the recipe, the finished product had not always the same properties. This occurs when a new raw material is used which had not been used before. Although it may fulfill all requirements of chemical anlysis, yet in practice the customary results are not obtained, for the reason that the uniformity of the new raw material is not sufficiently guaranteed. In this way the amount of scrap in special steel manufacture often becomes large beyond all proportions.

In this special point the manufacture of tool steels resembles the manufacture of a certain kind of other compounds which have nothing to do with the steel industry, but which can be made successfully only by following with absolute accuracy all details of a given recipe and by using always the same starting materials which have been found satisfactory. A very slight departure from the customary practice leads nearly always in this case to a failure. This case is the manufacture of artificial synthetic perfumes.

Moreover, it is not at all simple to follow a recipe for making special steels in spite of all imaginable precautions. This fact explains why only a few steel works have so far been able to furnish good and, above all, uniform special steels to their customers.

I do not need to say that I do not intend to speak in favor of following blindly a given recipe; it is clear that the recipe must be worked out on the basis of scientific principles, and of logically arranged experimental investigations. To give some suggestions for the difficult work in this field, is the purpose of this paper which the editor of this journal asked me to prepare. I hope to give some new information, supplementing what has already been published on vanadium and titanium—two elements which have been considered for some time as specially valuable for steel.

VANADIUM.

During the past few years vandium has been frequently tried in steel; indeed, it would not be any exaggeration to say that there is hardly a steel works of any importance in existence which has not experimented with this element, although its high price certainly cannot encourage the undertaking of experiments which are expensive enough in themselves. The high price of this substance, however, is only of relative importance, for the application of this element depends upon the results and on the required quantity of the element.

In both of these directions important progress appears to have been made recently, since notable improvements in steel have been successfully made with very small quantities of vanadium, so that the result attained seems to be by no means disproportionate to the cost.

In most of the earlier experiments ½ to 1 per cent of vanadium was added to the steel, but recently the quantity has been considerably reduced, and very important results have been obtained with 0.25 per cent of vanadium, and with even less. Prof. Arnold, of Sheffield (England), has carried out a series of scientific experiments on vanadium steels, and has had them published (see the note by A. F. Wiener, *Engineer*, July 1, 1904). These experiments may be reviewed here in the first place, so that we may draw further conclusions from them.

The most important observation that Prof. Arnold makes is a metallographic one: under the microscope the vanadium steel looks quite different from ordinary steel without vanadium. In ordinary steel (after the customary etching), two *distinctly* differentiated bodies appear under the microscope separated from one another, the one bright and the other dark, imbedded in the former and forming the true steel. The bright component is ferrite, the dark one pearlite. When vanadium is present, this distinct separation is considerably effaced and the appearance becomes more softened; that is to say, the pearlite, owing to the addition of vanadium, has partially dissolved in the principal constituent, the ferrite, and a more uniform structure has resulted. This observation holds when the amount of pearlite is small; that is to say, according to the German way of using the terms, it does not hold for steel* proper, but for a soft iron.

One microscopic test comes from a soft steel containing 0.22 per cent carbon; the limit of elasticity of this steel was 25.72 tons (40.51 kg. per square millimeter), its ultimate tensile strength was 30.64 tons (48.26 kilograms per square millimeter), and its elongation at rupture amounted to 33.5 per cent. The vanadium steel investigated micrographically contained 0.2 per cent of carbon and 0.27 per cent of vanadium. The limit of elasticity of the alloy was 37.4 tons (58.91 kilograms per square millimeter), its ultimate strength 47.08 tons (74.15 kilograms per square millimeter), and its elongation at rupture 22 per cent.

The following tables give some authentic results which have been obtained by Prof. Arnold by the addition of vanadium alloys to ordinary steel, and in combination with the other alloys which it has been customary to use heretofore. The rods which were investigated were circular in cross-section, three-quarters of an inch in diameter, and were forged from ingots of crucible steel and rolled.

Some months later a series of comparison experiments were carried out by Prof. Arnold, with the results given in Table 2.

TABLE 1.

	Elastic limit.		Tensile strength.		Elongation at rupture.	Reduction of area.
	tons per in.^{2}	kgs. per mm.^{2}	tons per in.^{2}	kgs. per mm.^{2}		
No. 682. Plain steel, about 0.25 C., 0.4 Mn......	25.72	40.50	30.64	48.26	33.5	60.1
No. 684. Above steel plus 0.25 vanadium......	37.40	58.90	47.08	74.15	22.0	51.4
No. 684. Plain steel plus 3.34 nickel............	32.60	51.35	42.20	66.47	26.5	52.8
No. 685. Above nickel steel plus 0.25 vanadium..	50.28	79.19	68.16	107.35	17.0	36.3

TABLE 2.

	Elastic limit.		Tensile strength.		Elongation at rupture.	Reduction of area.
	tons per in.^{2}	kgs. per mm.^{2}	tons per in.^{2}	kgs. per mm.^{2}		
Plain steel, 0.25 carbon	18.0	28.35	27.0	42.53	38.0	65.0
Steel containing:						
0.25 C, 1 per cent W, 0.4 Mn..................	23.89	37.63	40.83	64.31	33.5	45.7
The same plus 0.25 Va........................	35.61	56.04	56.27	88.63	15.5	20.9
0.25 C, 1 per cent Ni, 0.4 Mn..................	24.94	39.28	37.58	59.19	39.5	46.1
The same plus 0.25 Va (No. 741)..............	37.62	59.25	49.26	77.58	28.0	32.0
0.25 C, 1 per cent Cr, 0.4 Mn	26.48	41.71	41.15	64.81	35.5	55.2
The same plus 0.25 Va (No. 742)	39.76	62.62	55.08	86.75	26.0	53.0

The tables 3 and 4 show what a remarkable effect is obtained by introducing a very small quantity of vanadium into a high percentage nickel steel. The best results are those for a material containing 0.15 to 0.20 per cent carbon and 11.39 per cent nickel, as follows: Tensile strength, 88.8 tons (139.86 kgs. per square millimeter); elastic limit, 49.7 tons (76.70 kgs. per square millimeter); elongation, 12 per cent; contraction, 26 per cent. An addition of about 0.3 per cent of vanadium raised the elastic limit by 10 per cent, and the tensile strength by 12.5 tons (19.69 kgs. per square millimeter), or about 14 per cent above the highest figures which are obtained with nickel alone.

The introduction of ferro-vanadium into the steel does not give any trouble wherever the operations were conducted with care and intelligence. There is no loss of the valuable alloy, if

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* By the term steel, we (in Germany) designate usually a rather high carbon content; with us the expression "weicher Stahl" (soft steel) corresponds to the French "acier dour," and the English soft steel is little used.

TABLE 3.

Rolled bars of nickel vanadium steel received from A. F. Wiener, made by W. Makin & Son, Sheffield.

ANALYSIS.

Test piece:—	1 %	2 %	3 %
Carbon	9.141	0.135	0.121
Silicon	0.056	0.093	0.084
Sulphur	0.048	0.044	0.044
Phosphorus	0.044	0.038	0.034
Manganese.	0.512	0.599	0.470
Nickel	0.630	9.290	8.660
Vanadium	0.290	0.162	0.120

it is added at the proper time, and care is taken that it is absorbed by the steel and not by the slag or scum. A perfectly homogeneous steel has been obtained in every case.

Thanks to the excellent new plant at University College, Sheffield, Professor Arnold has been able to make in the Siemens-Martin furnace a very remarkable casting, which contained 0.3 per cent carbon and 0.15 vanadium. After the 10-inch ingots had been hammered down to 3-inch billets and rolled to 1-inch bars, 42 tons (66.15 kg. per square millimeter) tensile strength, 22 tons (34.65 kgs. per square millimeter) elastic limit, 28 to 34 per cent elongation in 2 inches, and 55 per cent redurtion of area were recorded by the test. Moreover, this steel stood four complete twists when a straight 1-inch square bar tooled down to ⅞ inch round was submitted to a torsion test. The value of such a material is evident.

So much for the interesting experiments of Prof. Arnold. It is evident from the above experiments that especially the so-called soft steels are preferably alloyed with vanadium, and that further results may be expected in this direction. It is well known how important it is to get as high a durability as possible to meet the demands of modern industries; extra cost does not count, as long as it is not disproportionate to the result attained. We may instance the requirements of our automobile industry, which is demanding materials of higher and higher resistance, so as not to increase the weight any more; and the same is the case in ship-building.

Very remarkable experiments have been made recently in the [illegible]n of vanadium steel wire, containing about 0.25 per [illegible]on, and the same percentage of vanadium. This is also a case of a soft steel.

But vanadium has also been used for especially hard steel, so-called tool steel. That vanadium is especially suitable for this purpose, is shown by the very nature of the element. Prof. Muthmann, of Munich, has recently succeeded in obtaining pure vanadium by a method of his own. He found that this pure vanadium is, perhaps, the hardest of all known metals, since it scratches even corundum. If, then, this hard vanadium could be alloyed and imbedded in the iron—the body of the alloy, the so-called ferrite—in such a manner that it is not more or less dissolved (as in the case of soft steel mentioned at the beginning of this article, according to microscopic tests), then it should be possible to attain a steel of extraordinary hardness. In this direction progress can be made only by means of experiments, which should depend, not only on chemical analyses, but also, above all things, upon the microscope.

Just at the present time it seems especially important to find a new alloy for hard tool steel, since that which is most used—tungsten—is difficult to obtain and subject to considerable variations of price. It is self-evident that a complete replacement of so important an element as tungsten for alloying for many purposes, cannot be spoken of; all that can be done, is to find some other material for the preparation of hard steel, which, in the most favorable case, may form a partial substitute. Experiments have also been made regarding the simultaneous application of vanadium and tungsten, but hitherto the results have been indefinite, particularly when rather large quantities of tungsten were present in the alloy. But, of course, these experiments are not yet concluded.

On the other hand, the simultaneous use of chromium and vanadium—even up to several per cent of chromium—appears to be advantageous. It is well known that chromium and tungsten are now generally applied together. Vanadium is not added to steel as such, but always as ferro-vanadium, generally with 25 per cent of vanadium.

The second part of this article, dealing with titanium, will be published in our next issue.

TABLE 4.

MECHANICAL TESTS.

A = original condition; B = annealed blood-red (610-648° C.); C = annealed cherry-red (740-760° C.); all cooled out in warm ashes.

Test piece.	1			2			3		
	A	B	C	A	B	C	A	B	C
Tensile strength—									
Tons per in.²	101.20	97.84	78.20	98.60	96.42	98.50	67.36	76.40	63.72
Kgs. per mm.²	159.39	154.10	123.17	155.30	151.86	155.14	106.09	120.33	100.36
Elastic limit—									
Tons per in.²	53.92	23.50	35.71	53.66	35.57	45.18	51.14	39.44	59.53
Kgs. per mm.²	84.92	37.15	56.24	84.51	56.02	71.16	80.55	62.12	62.26
Ratio, elastic limit to tensile strength	0.53	0.24	0.45	0.54	0.36	0.45	0.76	0.51	0.62
Elongation in % on 2 in..	11.0	12.0	20.0	10.0	7.0*	14.0	16.5	18.5	18.0
Contraction of area, %...	35.5	15.4	36.0	25.4	7.1*	42.2	42.0	36.0	45.7

* Broke short. No. 2, as received, was probably chilled.

CORRESPONDENCE.

A Society Badge.

To the Editor of Electrochemical and Metallurgical Industry:

SIR:—The advisability of selecting a suitable pin or badge for the members of the American Electrochemical Society has already been discussed in your columns. I venture to call the attention of members to the badge distributed at the Boston meeting. It is attractive, and, with its electric furnace design, it is characteristic. It would be well for the Board of Directors to adopt it officially for general use.

A. N. H.

New York City.

Boston Meeting of the American Electrochemical Society.

The seventh meeting of the American Electrochemical Society, held in Boston from April 25 to 27, was a most enjoyable social affair, and the local committee, with Dr. W. H. Walker as chairman and Dr. H. P. Talbot as secretary, as well as the authorities of Harvard University and of the Massachusetts Institute of Technology and other hosts of the Society deserve the highest praise for the true Bostonian spirit in which the Society was entertained. Considered purely from a professional point of view, the meeting turned out to be one of steadily increasing interest and of growing success, a remarkable climax coming on the last day; it was to be regretted that the time was then somewhat limited. Thus, the Convention was in every respect a very decided success.

At the business meeting, held on Wednesday morning, the reports of the secretary and treasurer were read and accepted; they show that the Society is in a very good financial condition. The proposed amendment to the constitution, by which the time of election of officers is slightly changed, was accepted. The result of the annual election of officers was announced in the Thursday meeting, and is as follows: President, Dr. Wilder D. Bancroft, of Cornell University; vice-presidents, Messrs. Carl Hering, Louis Kahlenberg, E. F. Roeber; managers, Messrs. C. A. Doremus, C. P. Townsend, W. R. Whitney; secretary, Mr. S. S. Sadtler; treasurer, Mr. P. G. Salom.

The three forenoons of April 25, 26, 27, were devoted to the reading and discussion of papers; a full report will be found below.

After the morning session of April 25, a reception was held by President Pritchett, of the Massachusetts Institute, at the Technology Club, followed by a lunch, tendered by the Institute. The afternoon was devoted to an excursion to the General Electric Company's plant at Lynn. Professor Elihu Thomson and Mr. Richard Fleming showed the visitors, with great courtesy, through the works, and the occasion was highly enjoyed. The visitors had an opportunity of seeing large and small machinery in the process of manufacture. We may only mention steam turbines and electrical measuring instruments, while a highly appreciated feature of the visit was a practical demonstration of the great variety of the spectra of various types of electric light (incandescent lamp, Nernst lamp, arc lamp, mercury vapor lamp, etc.) by showing the remarkably different color effects which these different sources of light produce in the same colored fabrics.

On the evening of the same day, after Dr. H. S. Carhart's presidential address, a smoker was held in the Palm Garden of the Hotel Lenox; it was conducted by Dr. W. H. Walker, somewhat in form of a German beer "commers," and was very "gemütlich," the tunes of the Prince of Pilsen alternating impartially with those of the Commersbuch.

After the close of the Wednesday morning session a reception was held at Harvard Union by President Eliot, of Harvard University, who made a felicitous little speech, in which he emphasized the importance of electrochemistry; it is in the borderland of sciences that the most radical achievements are often obtained. He sketched the ideals and tendencies determining the development of higher scientific and technical education in the United States. A lunch was then taken which was tendered by Harvard University.

For the afternoon two parties were made up. One visited the buildings of Harvard University, under the leadership of the professors of the various institutions, while the other party made an excursion to the works of the New England Gas & Coke Co., at Everett (Otto-Hoffmann system), where the visitors were shown with great courtesy through the different departments of the works. The plant uses normally per day 1,700 tons of coal, which are brought by ship directly to the works from Nova Scotia. A storage of coal, sufficient for three weeks' work, is possible.

On the evening of the same day a delightful banquet was held at the Hotel Lenox; Dr. A. E. Kennelly distinguished himself as toastmaster and introduced the speakers with humorous and pointed remarks. Dr. H. S. Carhart, as the retiring president, reversed an old joke of Dr. Wiley, who, at a former banquet, had tried to connect the initials of the Society —A. E. S.—through the Latin word *aes* with certain tendencies. Dr. Carhart reversed the order of the initials, and compared the Society and electrochemistry in general with a *sea* of unlimited possibilities. Dr. J. W. Richards took up this theme; after having referred to the formation of a Society in Philadelphia, he sketched the splendid future which lies before electrochemical and metallurgical industries. Dr. H. W. Wiley made one of his characteristic and charming speeches, full of humor and wit—at the expense of others as well as himself. He had in the beginning been opposed to the formation of the Society, but when it was founded he came in quickly, and was not sorry for it. Chemistry is a science of such fundamental and vast importance that it needs various societies to cover the ground thoroughly. Professor W. S. Franklin spoke of the necessity of utilizing forms of energy now wasted, and told a story of a man looking at Niagara Falls and saying: "What a waste!" But he was not "one of those electrochemists," he was a milkman.

After the Thursday session a lunch was tendered again at the Technology Club by the Massachusetts Institute. The afternoon was devoted to visits to the various buildings and laboratories of the Massachusetts Institute.

TUESDAY SESSION.

The first session was held at Lowell Hall of the Massachusetts Institute of Technology. President H. S. Carhart called the meeting to order.

REVERSIBLE AND IRREVERSIBLE ELECTROLYTIC POLARIZATION.

A paper on this subject, by Professor W. S. **FRANKLIN** and Mr. L. A. **FREUDENBERGER**, was presented by Professor Franklin.

The authors begin with some general remarks on the essential suppositions made in applying the principles of thermodynamics to actual phenomena, and then pass over to the subject of electrolysis. In a study of the phenomena of electrolysis, a thing of first importance is to determine, within the limits of experimental precision, whether the process approaches complete reversibility as it is made to take place more and more slowly. The great difficulty encountered here, aside from the difficulty of approaching a limit by experimentation, is that subsidiary actions take place during electrolysis (diffusion, local action, etc.). The ordinary point of view is to consider that those subsidiary actions may be legitimately treated as actions which are independent of the main phenomenon of the deposition and discharge of the ions on the electrodes by the current.

The authors refer to Caspari's investigation of the cathode polarization in the electrolysis of sulphuric acid when the current is barely large enough to produce visible generation of hydrogen. Caspari found that cathodes of the different metals show characteristic excess-voltages (Überspannungen), *i. e.*, the development of hydrogen at the cathode requires a voltage in excess of the value corresponding to the reversible liberation of hydrogen. This excess-voltage, or over-voltage, is nearly zero on spongy platinum, and about three-quarters of a volt on zinc and on pure mercury. Ostwald and Bredig have suggested explanations according to which the over voltage would be due to fundamentally reversible processes immediately accompanying electrolysis proper, while only independent actions, really subsequent to the deposition, are irreversible.

The authors then take up the discussion of the reversibility or irreversibility of electrolytic polarization by the following method. The work spent in forcing a current through an electrolytic cell consists mainly of three parts:

(*a*) The work which appears as heat throughout the electrolyte. The rate at which work is so spent—that is, the rate of generation of heat throughout the electrolyte, is accurately proportional to the square of the current, according to Joule's law.

(*b*) The work which appears as heat at the electrodes on account of the irreversible processes which take place as the ions are deposited. The rate at which work so expended is a function of the current, and it may be represented by F (i).

(*c*) The chemical decomposition of the electrolyte by the current requires work, and the reaction of the liberated ions upon the solvent or upon the electrodes is often a source of energy. The net rate of expenditure of energy in these ways is proportional to the current—it may be either positive or negative. Work gained or spent in an electrolytic cell on account of thermoelectromotive forces or on account of electromotive forces at places where the electrolyte changes in concentration, or in composition may be included in this item (c).

Thus, if a current *i* passes through an electrolytic cell, and if the e. m. f. is E, the power is

$$Ei = Ri^2 + F(i) + e\,i$$

where *e* is the proportionality constant mentioned above in item (*c*). From this follows directly the e. m. f. equation,

$$E = Ri + f\ (i) + e$$

so that the total e. m. f. E may be considered as the sum of three distinct items. The first one, R*i*, is the e. m. f. consumed by ohmic resistance. The sum of the two others (or rather its negative value) is the e. m. f. of polarization. Now, it is at once seen that the polarization consists of two parts: one, the reversible polarization, *e*, which is not a function of the current; the other f (*i*), the irreversible polarization which is a function of the current.

By this method it is, of course, directly possible to separate experimentally the reversible from the irreversible polarization, and to decide the fundamental question, whether there exists an irreversible polarization. This was the object of the experiments of the authors. The method is to cause the current to vary, thus separating R*i* and f (*i*) from *e*, so that f (*i*) becomes known when R*i* has been determined. The difficulty encountered in the realization of this method is that *e* is to a certain extent a function of the coulombs passed, because of local concentration changes of the electrolyte, because of progressive changes in the character of the surface layers of the electrodes, and because of temperature changes due to continued flow of current. However, the authors believe that by their arrangement of the experiments they eliminated these disturbing factors. They experimented with copper sulphate solutions and copper electrodes.

The results were plotted by the authors in form of curves, which were shown by Professor Franklin on Wednesday. The authors determined the irreversible polarization f (*i*), but did not separate it into anode and cathode parts. The results show distinctly the existence of irreversible polarization. For particulars the reader must be referred to the full paper, which will be printed in the Transactions of the Society.

In the discussion which followed, Mr. Carl Hering spoke at length on the use and misuse of the term polarization. He also referred to the development of oxygen and hydrogen by electrolysis of water, and distinguished two processes; first, the disintegration of chemical bonds; second, the expansion of gases against atmospheric pressure. He asked where the energy for the second process comes from, and expressed the opinion that it cannot come from the current.

ALUMINIUM ELECTROLYTIC CONDENSER.

A paper on this subject, presented by Mr. **CLARENCE IRVING ZIMMERMAN**, was a continuation of his former paper on the same subject (our Vol. II, p. 182). The author first explained the action of an aluminium electrolytic condenser by an hydraulic analogy with a combination of membranes and valves. He then gave the results of experiments in which the resistance, dielectric strength and inductivity of the film on the aluminium anode were determined, and found to be as high, or higher, than in ordinary dielectrics. The resistance in one instance is shown to be about 10^{11} ohms per cubic centimeter. A value of about 5,000,000 volts per centimeter is obtained for the dielectric strength. A value of about 14.6 is obtained for the inductivity (specific inductive capacity or dielectric constant).

The author then gave a graphical analysis of the cell losses. They consist of (1) film losses, (2) C^2R losses, and (3) electrolytic decomposition losses, while the film losses (1) consist of (a) leakage through film, and (b) losses proportional to the frequency. The latter are in general larger than the other losses in the cell, and are of a nature not clearly understood. A crude graphical representation of these losses is given in Fig. 1, where there is plotted a curve between watts and frequency. The observations were made upon two cells in which the electrolyte consisted of an aqueous solution of commercial borax. All the losses were increased considerably by this impure electrolyte.

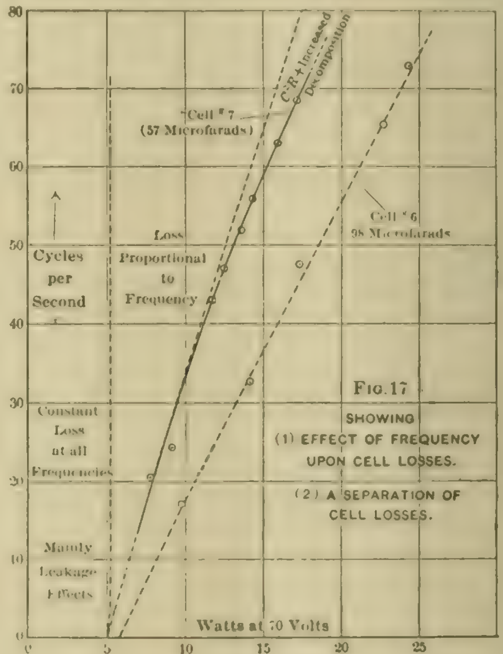

FIG. 1.—LOSSES IN ELECTROLYTIC CONDENSER.

The condenser-charging current varies in proportion to the frequency, when the impressed pressure is kept constant. At low frequencies, however, the watt consumption shows a slight variation from a straight line law, due to the predominating effect of the leakage current.

The theoretical relation of the alternating pressure, *e*, impressed upon the cell terminals to the unidirectional pressure between the electrolyte and a point neutral to the alternating pressure is equal to the impressed pressure *e*, divided by $\sqrt{2}$. The actual values obtained are smaller than these, due chiefly to the imperfect symmetry and partially to the cell resistance, to film leakage, and to polarization pressures.

Fig. 2 shows the relations existing in a cell each of whose plates had a capacity of 114 microfarads. The electrolyte consisted of a saturated solution of borax with a small amount of glycerine added. The method of obtaining these pressure relations is shown in Fig. 3. The alternating-current 60-cycle pressure was varied from 140 volts to a fraction of a volt. The curve is practically a straight line above 10 volts alternating pressure. Below this point, the unidirectional pressures keep close to the zero value, owing to the slight asymmetrical effect of the film at such low values and to the polarization pressures between the aluminium and the carbon. If the straight part of this curve be projected until it crosses the horizontal line representing the zero of alternating pressure, the undirectional value is 4½ volts. Part of this pressure is probably produced by the chain aluminium - electrolyte - carbon. The dotted line in the diagram represents the theoretical relation based upon the assumption that there is perfect symmetry, that there are no polarization pressures, and that the losses in the cell are zero. These pressure relations do not materially vary with any small change in frequency, and since such a cell can be easily set up, this scheme may be used for the purpose of calibrating alternating-pressure instruments where extreme accuracy is not desired. It is not necessary to use the zero method of Fig. 3 for the purpose of calibration. It is merely necessary to connect the alternating-current instruments across the cell terminals and the direct-current instruments between the carbon (electrolyte) and one of the aluminium electrodes.

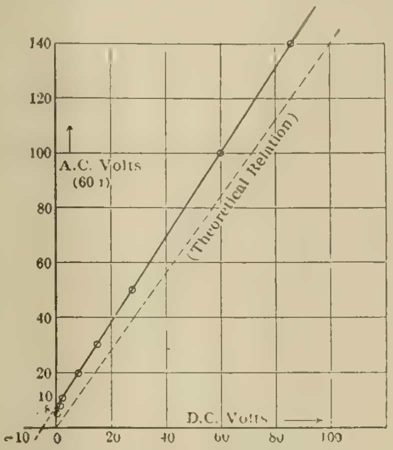

FIG. 2.—RELATION BETWEEN DIRECT AND ALTERNATING-CURRENT VOLTAGE.

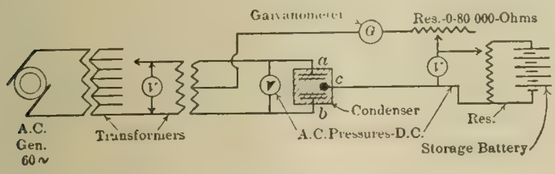

FIG. 3.—ARRANGEMENT OF EXPERIMENTS.

The author then discussed what he calls "unsymmetrical condensers," *i. e.*, aluminium condensers having two electrodes with different electrostatic capacities. Their behavior was fully discussed and analyzed graphically and mathematically. Oscillograph curves were shown of the different pressures. It was shown that the unidirectional pressures across each electrode may be considerably greater than the impressed pressure. The behavior of the cell on alternating-current circuit is, notwithstanding its peculiarities, similar to two tinfoil condensers in series. The "equivalent" capacity is $\frac{c_1 c_2}{c_1 + c_2}$ where c_1 and c_2 are the electrostatic capacities of the films on the two electrodes. Two-phase and three-phase condensers were also discussed, and the three-phase condenser pressures were shown diagrammatically. The paper was illustrated by a number of interesting experiments.

The paper was briefly discussed in a communicated note of Mr. W. R. Mott, who made some complimentary remarks on Zimmerman's work, but pointed out that further and quite difficult investigations are necessary to determine the absolute figures for all the properties of the film.

ECONOMIC TEMPERATURES OF COPPER-REFINING SOLUTIONS.

A paper on this subject, by Professor CHARLES F. BURGESS, was then presented, in the absence of the author, by Dr. M. deK. Thompson. It referred to the investigation of Bancroft (our Vol. I, pp. 484 and 584), who, from experiments made by him, had concluded that a greater economy can be effected if the solutions are run at a higher temperature than has been commonly adopted by copper refineries. In the present paper Professor Burgess shows that Bancroft's conclusion is open to objections.

The pressure between the two electrodes of a copper-refining tank is the sum of two quantities—the polarization pressure and the drop caused by the current flowing through the electrolyte. The former is small and may be considered as practically constant for a given solution, being influenced very slightly by the temperature. The IR drop through the electrolyte varies, however, in a marked degree with change of temperature, and it is this variation which causes the falling off in applied pressure necessary to pass a given current through the cell as the temperature rises.

Professor Bancroft experimented with copper electrodes 1 centimeter apart, in a solution containing 16 per cent copper sulphate crystals and 9 per cent free acid, with a current density of 3.5 amperes per square decimeter, and found that the pressure at 70° is less than half that required at 20° for the same current, and consequently that the cost of the power is less than one-half. He stated that "by expressing voltage in terms of the voltage at 20° the effect of the distance between the plates is eliminated," and that "the percentage variations hold for any set of plates in the same solution."

Professor Burgess objects that this assumption, which served as a basis upon which various conclusions were drawn, would undoubtedly be correct if we are concerned only with the specific resistance of the electrolyte itself, but since it is the virtual resistance between the electrodes that is involved, the assumption is not justified.

The resistance of the cell includes the resistance between the electrode surfaces and the electrolyte, and the resistance of the electrolyte itself. The former is of considerable magnitude in comparison with the latter when the electrodes are close together, and of much less relative magnitude when a considerable column of electrolyte intervenes. The temperature coefficient at the electrode surface resistance is much greater than the temperature coefficient of the electrolyte, and unless these separate resistances and their corresponding temperature coefficients be known, it is impossible to calculate the temperature coefficient of the complete cell with electrodes 5 centimeters apart from measurements on electrodes 1 centimeter apart.

The author gives results of measurements which show clearly that the lowering of voltage is not proportional to the increase of temperature regardless of the distance between the plates, but that on the other hand the temperature effect is far more marked when the electrodes are close together than when they are far apart. In other words, the apparent temperature coefficient is dependent upon the distance between electrodes, being greatest when the electrodes are close together.

We are not, therefore, warranted in saying that the most economical temperature in a cell with the plates 1 centimeter apart is the most economical for a cell with the plates one-half inch or 1½ inches apart, as in the series and multiple refining systems respectively.

The author then discusses what Gore called the "transfer resistance" from electrode to electrolyte, and points out that if there exists a resistance at the electrode surface apart from the resistance of the electrolyte itself, of such nature that it has a great temperature coefficient, we have an explanation of the dependence of the apparent temperature coefficient of an electrolyte upon the distance between the electrodes, the reduction of the "transfer resistance" producing a greater proportion of the total change when the electrodes are close together than when farther separated. Professor Burgess describes a series of experiments which appear to show that in a neutral copper sulphate solution the change of resistance with temperature is due almost entirely to the temperature coefficient of the electrolyte itself, while there are distinct electrode effects in case the electrolyte contains free acid. Further experiments were made by tracing the fall of potential from the anode to the cathode by means of a normal calomel electrode. There is a fall of potential at the two electrodes equal to about 0.2 volt, the cathode contributing to this effect a considerably greater part than the anode. On the other hand, the polarization, as measured by the voltmeter upon interruption of the current, is only 0.02 volt. Against the assumption that this method might not give the total polarization, Professor Burgess remarks that it would be apparently out of the question to assume a polarization of about 0.2 volt.

Neither does the drop of potential appear to be due to changes of composition of the solution as a result of electrolysis, since in this case the effect would be materially decreased by agitation of the electrolyte, while in reality the agitation of the solution had very little influence on this surface resistance. It is also shown that the electrode effect is not a result of electrolytic products, since it also accompanies the flow of alternating current of such small value as to make the accumulation of electrolytic products impossible.

In the discussion which followed, Professor W. S. Franklin said that a genuine polarization of 0.2 volt is not out of question; he found in his experiments an actual polarization of 0.1 volt with quite small current densities. He also showed how some of the results of Burgess may be used to decide the question of reversible and irreversible polarization. Messrs. Lidbury, Hering and Thompson also participated in the discussion. Mr. Lawrence Addicks had communicated a note in which he said that while Professor Burgess' results are very interesting, he has not solved the conundrum; there is no doubt as to the existence of a transfer resistance, but we don't know its nature.

A DIAPHRAGM CELL FOR ELECTROLYSIS OF SODIUM CHLORIDE SOLUTIONS.

An interesting paper, by Mr. **CLINTON PAUL TOWNSEND**, describing his new diaphragm cell (see also page 73 of our March issue) was then presented.

The electrolysis of sodium chloride solutions with insoluble anodes under conditions which permit the liberated sodium to decompose water may readily become a complex operation by reason of the variety of secondary products which may result. If it be desired to isolate with substantial completeness the chlorine which is the primary anode product from the sodium hydroxide, which is the secondary cathode product, it is necessary, first, to prevent the mechanical mingling of anode and cathode solutions, and, second, to prevent the sodium hydroxide formed at the cathode from participating in the electrolysis. The first of these conditions is readily secured by a variety of means, but the second has heretofore proven impossible of complete accomplishment.

As regards the mixing of the solutions, both anolyte and catholyte are necessarily in more or less violent motion, due to gas evolution. It is evident, however, that any device or disposition of parts which will insure a quiescent region of contact between these solutions will avoid their mixing. This is easily accomplished in the various gravity cells by depending partial partitions or bells which provide a quiet lower stratum of electrolyte; it is as readily accomplished by a diaphragm, say, of thin asbestos paper, the interstices of which provide the necessary quiescent layer.

A diaphragm of this character does not for practical purposes check the movement of the caustic by diffusion into the anode chamber, or the migration of the hydroxyl ions toward the anode if the conditions be such that the hydroxid takes part in the electrolysis. The diaphragm cell and the gravity cell may, therefore, be expected to work at ampere efficiencies of the same general order and the published data seem to bear out this conclusion. This ampere efficiency is understood to lie in practice between 75 and 85 per cent. These figures represent current losses which are in themselves serious; the losses are doubly serious in practice because they are largely represented by hypochlorite and chlorate in the anode liquor, in the presence of which graphite anodes rapidly disintegrate, yielding an impalpable carbon dust which will soon put a diaphragm cell out of commission. The use of platinum anodes is a possible alternative, but a costly one for cells equipped at the present time, and, moreover, they do not reduce the power losses; this latter objection holds against the practice of decomposing the hypochlorite in the anode compartment by addition of hydrochloric acid.

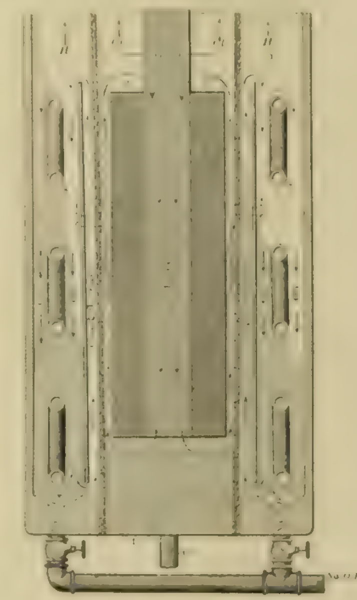

FIG. 4.—DIAPHRAGM CELL.

The statement is frequently made that if the rate of flow toward the cathode be rapid enough to compensate the migration velocity of the hydoxyl ion the ampere efficiency should be 100 per cent; this proposition applies to gravity and diaphragm cells alike, and is the basis of some of the earliest forms. As a matter of fact, however, the solution which flows from anode to cathode carries a very considerable percentage of dissolved chlorine, and practical experience has shown that long before such rate of flow has been reached that hypochlorite and chlorate disappear from the region of the anode, the losses by the action of this chlorine on the cathode products becomes serious. Under such extreme conditions the moving liquid can dissolve all of the chlorine and the caustic efficiency of the cell will fall to zero.

The instantaneous removal of the caustic, as formed from the electrolytic field and from contact with the anode liquor, would mean the avoidance of all difficulties which arise from the presence of hydroxyl ions in the anode compartment. The chlorinated liquor would then remain free from hypochlorite or chlorate; graphite anodes should be substantially permanent, and the evolved chlorine should be pure. This result is nearly and, perhaps, fully accomplished by a cell, the lower portion of which is somewhat diagrammatically illustrated in Fig. 4 in transverse vertical section. The experimental cell has a central brine compartment 1, having cement walls and carrying spaced anodes 2, 2. These anodes, as well as their conductors 2′, are of Acheson graphite. The lateral compartments 3, 3, are of iron, and in operation are filled with a mineral oil which completely submerges one face—the back—of the wire-net cathode 4. The active face of this cathode is in contact with the diaphragm 5, which is, of course, quite permeable to the clorinated anode liquor. The diaphragm is supported on the side toward the anode by a pervious layer, 6, of good mechanical strength; 7, 7, are steel plates adjacent and parallel to the cathodes and submerged in the oil; 8, are pipes for maintaining the temperature of the oil, and thereby of the electrolyte, 9, 10, is the caustic outflow, and 11 the brine inlet.

The chlorine liberated at the anode faces rises through the narrow channels between the anodes and the diaphragms, and acts in a manner similar to an air-lift to induce through this channel a flow of electrolyte in the direction indicated by the arrows. The rate of this flow, the purpose of which is to wash the diaphragm and to prevent any fall of concentration around the anode, depends upon the current density, and under proper conditions attains several feet per second. Uniform concentration is maintained.

As stated, the inactive face of the pervious cathode is immersed in a liquid which is immiscible with the caustic solution and chemically inert toward it. The hydrogen which is liberated at the cathode escapes into the oil, and thence upwardly through the narrow channel afforded by the plate 7, and sets the body of oil in extremely rapid circulation. At high-current densities, in excess of 100 amperes per square foot of electrode surface, the rapidity of this circulation is quite striking.

The oil acts as a seal to prevent carbonating the caustic, as a support for the diaphragm, which may be of extreme thinness, and by its partially balancing hydrostatic pressure it tends to equalize the flow of electrolyte from top to bottom of the diaphragm. But its chief function relates to the quick removal of the caustic from the electrolytic field. Caustic solution does not adhere to surfaces wetted with oil, and, therefore, the caustic does not accumulate in the cathode nor flow down the cathode in proximity to the chlorinated liquor; instead, it is projected by the escaping hydrogen into the oil in the form of minute globules and deposited under the oil at the outlet of the cell.

If this removal of the caustic were instantaneous neither hypochlorite nor chlorate, derived from this source, would be found in the anode liquor. The data following represent the composition of the anode liquor in a cell having 9 square feet of active anode surface. The experimental run was continuous from August 14 to September 7, the anode liquor being sampled daily.

Date	Amp.	Cl Grams per Liter	ClO Grams per Liter	ClO_3 Grams per Liter	
Aug. 14	1000	....	....	....	Start 3.30 p. m.
" 15	"	0.57	0.084	0.424	
" 16	"	0.14	0.154	0.0	Analy'd Aug. 18
" 17	"	0.26	0.0	0.0	Analy'd Aug. 18
" 18	"	0.35	0.105	0.0	
" 19	"	0.64	0.07	0.0	
" 20	"	0.355	0.091	0.0	
" 21	"	0.41	0.091	0.0	Analy'd Aug. 23
" 22	"	0.28	0.028	0.0	Analy'd Aug. 23
" 23	"	0.50	0.133	0.0	
" 24	"	0.53	0.091	0.0	
" 25	"	0.55	0.091	0.0	
" 26 a. m.	1200	1.02	0.154	0.368	
" 26 p. m.	"	1.03	0.21	0.440	
" 27	"	0.50	0.077	0.088	
" 28	"	0.43	0.0	0.0	
" 29	"	0.46	0.133	0.0	
" 30	"	0.57	0.063	0.0	
" 31	"	0.57	0.119	0.0	
Sept. 1	"	0.51	0.119	0.0	
" 2	"	1.065	0.049	0.088	Analy'd Sept. 12
" 3	"	0.19	0.119	0.224	Analy'd Sept. 15
" 4	"	0.46	0.0	0.072	Analy'd Sept. 15
" 5	"	0.28	0.0	0.104	Analy'd Sept. 15
" 6	"	0.28	0.028	0.128	Analy'd Sept. 15
" 7	"	0.38	0.07	0.144	Analy'd Sept. 12

The samples of Sept. 2-7, inclusive, were analyzed after standing for several days in bottles exposed to the light. The traces of chlorate shown in these particular analyses were in all probability produced during this period.

During the run the cement walls of the anode compartment were attacked to a considerable extent, some calcium and magnesium appearing in solution and hydrated silica in suspension in the anode liquor. It is by no means certain that the persistent traces of hypochlorite shown by the analyses, never rising to 2/100 per cent and seldom entirely disappearing, existed as the sodium salt; it is quite probable that the hypochlorite is attributable chiefly to the action of the chlorine on the cement, more particularly because it reaches its maxima at the very beginning of the run, and whenever the wash of the anode liquor is increased by raising the current density.

The above results should correspond to substantially pure chlorine. The samples collected showed 99.7 to 99.8 per cent Cl, subject to a correction of unknown magnitude for the aeration of the anode liquor which, under the particular circumstances, was unavoidable. The effect of this and other errors affecting the analysis would be to increase the above chlorine value.

Caustic efficiency: The anode liquor carried about 0.55 grams per liter of dissolved chlorine. This is the solution which oxidizes the cathodic deposit of sodium, for, as will be apparent, the cell has no distinct cathode electrolyte. The dissolved chlorine in a volume of solution corresponding to the actual outflow through the cathode would correspond to a loss of caustic amounting to about 0.4 per cent of the theoretical yield for 1000 amperes. We should, therefore, expect a caustic efficiency not to exceed 99.6 per cent of theory. All caustic solution produced throughout the run was weighed and analyzed, but the errors incident to sampling and current measurement were sufficient to mask this presumably necessary loss, the apparent caustic efficiency being in slight excess of the expected result.

Mr. Townsend then referred to his article published in Electrochemical Industry for December, 1902, in which he gave some data as to the sodium chloride content of certain electrolytic caustic soda lyes. At that time it was not considered advisable to describe the cell other than by the statement then made that "it was provided with a thin permeable diaphragm, adapted to restrain and equalize the flow of the electrolyte, and a cathode whose form fitted it to serve as a surface for the rapid discharge of the cathode products." This possibly somewhat inadequate description of the present cell gave rise to a misconception, which led to a criticism[1] of the article on the ground that two essential conditions had been overlooked, to wit: the concentration changes which take place around the electrodes when sodium chloride solution is electrolyzed in the anode and cathode compartments of a diaphragm cell, and the migration velocities of the hydroxyl ions when the formed caustic participates in the electrolysis. Mr. Townsend replies that it will now be clear that the reasons for not considering these factors were, first, that the caustic is formed by oxidation of sodium in a homogeneous chloride solution maintained at constant concentration, and, second, that the "rapid discharge of the cathode products" is sufficient to guard against the entry of the caustic into the electrolytic field.

In the discussion, Dr. J. W. Richards asked what prevents the sodium chloride from passing into the caustic, and Mr. Townsend replied that there is no means of preventing that. But the separation of the chloride from the caustic can be easily effected by evaporation until the specific gravity is about 50° Baume.

Conduction in Electrolytes.

A paper on this subject was presented by Professor **Joseph W. Richards**, who spoke especially on the conduction in molten and solid electrolytes. He emphasized that if the conductivity of a fused or solid salt is determined as function of the present state, there is no break in this curve at the melting point, and from the curve alone it would be impossible to say when the electrolyte begins to set. (The conductivity [illegible] only when crystals begin to form.) The author concludes that whatever the nature of conductivity of salts in the liquid state, the conductivity in the solid state must be necessarily of the same nature.

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[1] Trans. Amer. Electrochemical Society, Vol. III, p. 177.

Solid sodium carbonate shows a measurable conductivity at 600° below the melting point. Reference was also made to the recent work of Haber and Tolloczko (our March issue, p. [illegible]).

The author then develops the following hypothesis: Electrolytic conduction in a solid salt seems to come into existence when the possibility of the formation of new phases at the two electrodes is caused by the increase in temperature. The migration of the ions is then the consequence of the formation of new phases. The acid phase produced at the anode, and the basic phase produced at the cathode diffuse against each other; and this diffusion of the new phases accounts for the migration of the ions. Thus the phenomenon of electrolysis is dissolved by the author into the creation of new phases and their diffusion.

In the discussion which followed, Professor H. S. Carhart thought that this theory does not explain how the passage of the current does take place in the first place. Professor W. S. Franklin said that diffusion undoubtedly occurs, but is something subsequent to electrolysis proper. Dr. W. R. Whitney spoke of the danger of confusing the electrolytic reactions at the electrodes with the conduction through the electrolyte. Dr. Theo. W. Richards thought the hypothesis would not account for such cases as electrolysis of silver nitrate between silver electrodes. Dr. Joseph W. Richards replied he did not know just how the electric current passes through a salt, but he expressed the conviction that electric conduction is essentially of the same nature in metals, fused salts and solid salts.

Mr. C. I. Zimmerman gave some very interesting notes on the conduction of Nernst lamp filaments. There is direct evidence that there is electrolysis, to a certain extent, while other phenomena show a distinct analogy with gaseous conduction, particles being thrown off from the negative electrode. Iron, gold, etc. (when used as terminals through which the current is introduced), travel against the direction of the current. There is also strong evidence of thermoelectric effects at the junctions. If platinum is used at the positive terminal, and silver at the negative terminal, the platinum may become white hot, while the silver does not melt, in spite of its much lower melting point; the reason is that at the silver junction heat is absorbed by the thermoelectric effect.

ABSOLUTE POTENTIALS.

Dr. H. M. **Goodwin** then presented a paper on Billitzer's method of determining absolute potentials. It will be remembered that Billitzer has found a value for the absolute potential of the calomel electrode which is different by 0.7 volt from the usual value (0.57 volt) determined by the drop electrode. It was, therefore, of importance to repeat Billitzer's experiments, but this is difficult, since Billitzer, in the description of his experiments, did not state the concentration of the solutions used. Dr. Goodwin repeated, as far as possible, those experiments in which zero deflection of a suspended metallic particle is taken as the criterion for the non-existence of an electric double layer. But he was able to get similar results as Billitzer in a general qualitative way only in some cases. In certain liquids no check whatever could be obtained. Another method of Billitzer, in which the migration of colloids was taken as indicator, was tested by Dr. Blake, who found that he could make the colloids travel at will in either direction, according to the amount of gelatine present. Under these circumstances it would be premature to give as the accepted value of the absolute potential of the calomel electrode.

EDISON STORAGE BATTERY.

Mr. M. deK. **Thompson** presented a paper on an experimental investigation which has been carried out in the Massachusetts Institute of Technology on the Edison nickel-iron storage battery. The results of this investigation were given by lantern slides, and referred mainly to measurements of the single potentials during charging and discharging. Some of these results were very interesting. For instance, if the concentration of the solution is varied, the discharge of the Fe electrode is little affected as long as the concentration is above a certain value. But this is changed for smaller concentrations. Thus, in a 5 per cent solution the discharge curve of the iron electrode is very different from that in a 20 per cent solution. A more detailed report of the paper must be deferred until the paper appears in full in the Transactions of the Society.

TUESDAY EVENING SESSION AND PRESIDENTIAL ADDRESS.

At a special evening session, in Lowell Hall of the Massachusetts Institute, a very fine exhibit of various chemical and electrochemical apparatus, by Messrs. **Eimer & Amend**, of New York City, was shown and explained by Dr. W. H. **Walker**. The exhibit covered the Wanner pyrometer, apparatus made from fused quartz, materials for electric furnaces, and a large collection of electric furnaces for many different kinds of work. Among the latter furnaces, using the new resistance material, Kryptol, were especially well represented. These furnaces were exhibited with the current switched on, and the ease of manipulating them was explained.

Then followed the main feature of the evening session—the address of the retiring president, Dr. **Henry S. Carhart**, who spoke in a most interesting manner on the development which the theories of electrolysis have undergone during the last hundred years. He gave a great many details on the early history (about 1800) of the theory of electrolysis, offered a quite enthusiastic account of Grotthus' researches, and covered successively the work of Clausius, Hittorf, Kohlrausch, Arrhenius and Van't Hoff. The speaker then passed over to J. J. Thomson's recent researches on electric currents through gases, and expressed the opinion that we have not yet the ultimate truth. We will, however, get at it, though "we need not prophesy—we need only to work and wait."

It is impossible to do justice to Dr. Carhart's address in a short abstract, but this address will be most interesting reading when published in full in the Transactions.

WEDNESDAY SESSION.

At this session, which was held in Peirce Hall of Harvard University, the annual reports of the secretary and treasurer were presented and accepted. The reading and discussion of papers followed.

CHROMIUM AND THE ELECTROLYSIS OF CHROMIC ACID.

A paper by Messrs. H. R. **Carveth** and B. E. **Curry** on this subject was then read by Dr. Carveth. The paper is extremely interesting, since the deposition of chromium from a chromic acid solution appeared a priori not very promising. There has been in the past a great diversity of opinion on this subject. The main results of the paper are as follows:

In the study of the cathode reactions at least three, but more probably four or more, different stages of oxidation are met. This means that in the various solutions there are chromium ions with four different valencies.

In regard to the constitution of chromic acid solutions the following facts must be noted: (a) The study of the equilibrium relations point to the existence of free CrO_3—a radical composed of a conductor and a non-conductor of electricity. So far as the authors are aware no compound of such a combination is known which, conducting the current in aqueous solution, is not by the supporters of the theory of electrolytic dissociation supposed to be dissociated into ions. (b) Chromium as anode may dissolve to form CrO_3. If dissolving to form an ion this will be hexavalent. There is a continuous passage from the conditions where it dissolves divalent (and everyone concedes its ionization, because a salt is formed) to the conditions where it dissolves hexavalent (where ionization has not been conceded, because an acid was formed). (c) Chromium may be deposited from chromic acid after a definite decomposition voltage has been reached.

This evidence, therefore, makes it seem very probable that in chromic acid there exists a number of hexavalent chromium cations in equilibrium with many other ions, and that chromium in chromic acid may be considered a reversible electrode.

The main part of the work of Geuther (Liebig's Annalen, Vol. 99, p. 314, 1856) is correct. Metallic chromium is deposited in the electrolysis of solution of chromic acid, but this requires a high-current density, which in this case is synonymous with a high-decomposition voltage. For example, using platinum wires as electrodes, Geuther used currents of from 0.07 to 0.35 amperes—very great densities, probably higher than the authors have used. On comparing with their own results the very great percentage reductions and the high metal depositions which he obtained, they conclude that his acid must have been impure. The oxygen excess which he found at the anode, and which enabled Buff to criticise him severely, the authors have never been able to find; his admission that his tangent galvanometer was wrong probably explains this. After the decomposition point is reached, the efficiency of metal deposition and the reduction of the acid go hand in hand; a chromium chromate is probably formed.

The negative results of Buff are understood when the current density he employed is calculated. Under no conditions have the authors been able to get deposition of the metal at a current density as low as he used; nor have they with any but very impure acids been able to duplicate the results. Like Buff, the other workers, Schick, Cowper-Coles and Ferée, have probably used too low current densities.

Some of the claims of Placet and Bonnet (U. S. Patent 526, 114, Sept. 18, 1894) are sound. It is very probable that solutions of commercial chromic acid and of chromates, to which have been added various substances which aid in the reduction, may not be used in a continuous and economical process for the extraction of chromium, since the efficiency is so rapidly decreased by the formation of the reduction products. By oxidizing these, however, it would be possible to continue the extraction of the metal, making the process continuous. There is nothing to prevent the use of this method in the laboratory for the purpose of making pure chromium.

By selection of the proper impurity (*e. g.*, sulphuric acid), the authors have been able in a continued electrolysis to recover in the metallic form more than half the total chromium present in the solution. Had the reduced products been oxidized, this yield could have been carried to any limits desired.

The criticisms of Le Blanc on the work of Geuther, Placet and Bonnet and Street need radical change. The only justification for such criticism was due to the experimental conditions not having been described in sufficient detail by the various workers.

Electrolytic chromium may occlude as much as 250 times its volume of hydrogen. This may account for its hardness, which may make chromium coatings commercially useful for various purposes. Chromium would constitute in many cases a useful because more resisting, coating on metals which are more easily attacked.

It is possible to reduce chromic acid to the trivalent chromic salts in the presence of an excess of a mineral acid.

The authors regard chromium merely as the type of a large number of elements which, according to the degree of oxidation, are either acid or basic in their properties. The methods which they have used should, they think, be capable of application to numerous other cases, such as the rare earth elements.

Dr. Carveth showed various samples. The paper was discussed by Drs. W. R. Whitney and J. W. Richards. The latter said that on account of the possibility of existing in so many states the deposition of chromium is difficult. He asked what was the voltage of decomposition. This might give us a possibility of calculating approximately some data which are badly needed for the thermochemistry of chromium salts.

Electrolytic Precipitation of Silver.

A paper on this subject, by Mr. **Ralph C. Snowden**, read, in the absence of the author, by Dr. Carveth, reports on a continuation of the researches carried out at Cornell University, in order to test Professor W. D. Bancroft's hypothesis that the precipitation of metals from aqueous solutions, particularly for plating purposes, is as easily controlled and as purely chemical as any chemical process, "the electric current being simply a means to an end and having the nature of any of our power factors." One conclusion from this hypothesis is that since in general chemical practice substances which are precipitated from solutions separate out larger crystals if precipitated slowly, thus, in electrolytic deposition the size of the crystals of the metal decreases with the increase in current density at the cathode unless secondary or extraordinary reactions take place. Rotating the cathode, or stirring vigorously by other means tends to flatten out the crystals, probably by simple skin friction, and it is possible to obtain a bright, polished deposit in many cases by running up the speed of the cathode and using a correspondingly high-current density.

The object of the investigation of the author was to study the effect of current density and speed of rotation of the cathode upon the electro-deposition of silver, and to ascertain the possibility of obtaining a plating deposit of silver from the nitrate solution. The main results are as follows:

A very finely crystalline deposit of silver can be obtained from a silver nitrate solution by rotating the cathode rapidly and keeping the anode and cathode solutions separated. The size of the crystals decreases with the increase in current density and the increase in the rate of stirring at the cathode.

The addition of small amounts of organic "colloids" to the solution makes the deposit very amorphous, and causes the precipitated metal to assume a colloidal state.

The presence of free nitrate acid decreases the size of crystals but slightly.

The paper was discussed by Messrs. Carhart, Sadtler, T. W. Richards and Kern, the latter showing a silver deposit obtained by the process of A. G. Betts, described in this journal, p. 145 of our last issue. Some remarks were also made with respect to the silver coulometer—a term which was very strongly recommended instead of voltameter, since this instrument measures coulombs.

Electrodeposition of Metals on Aluminium.

A paper presented by Mr. **Alexander Lodyguine** described some results of experiments with the electro-deposition of metals on aluminium. The object was to cover very thin aluminium plates with layers of antimonious lead or with antimony for use in light storage batteries; these coatings to be as thin as compatible with the complete protection of the aluminium from the action of the electrolyte. The author thinks that the principal cause for the failure in depositing metals on aluminium lies in the difficulty to keep the aluminium surface free from any compounds of aluminium in contact with it, and that not even the most vigorous scrubbing, brushing and washing, in alkalies, acids and water, suffices for arriving at these results. Any electrolyte that attacks the aluminium must be discarded, however slight and slow the attack may be.

The author did not find a good solution from which he could deposit lead or antimonious lead directly on an aluminium plate. He therefore covers the aluminium plate first with a coating of copper. He uses an anode of pure copper and an electrolyte of pure water, with a few drops of sulphuric acid ment: voltage 0.5, current density 0.0013 ampere per square inch. The plate is taken out after half an hour and vigorously washed and brushed in water, in a solution of hydrochloric acid, in a solution of caustic soda and once more in water, after which it is put again into the electrolyte. The same operation is repeated several times, until the plate is evenly and entirely coated with a fine and adherent coat of copper. After that the plate can be treated as an ordinary copper plate. (It may, for instance, be tinned for many practical uses.)

To deposit antimony on the copper-plated aluminium plate, the author uses a concentrated solution of protosulphide of sodium, Na_2S, with an excess of the protosulphide in the vat. Lumps of trisulphide of antimony, SbS, are placed in a porous cell of very oblong form, around a carbon plate; the cell is filled with the electrolyte and placed in a glass jar filled with the same electrolyte. Two plates of aluminium, coated with copper, are used as cathodes. The antimony adheres firmly to the plate, is of very fine structure and can be polished. The deposit can be continued as long as the monosulphide of sodium is not transformed into polysulphides. Voltage of deposition 1 to 1.5, current density 0.0013 amperes per square inch. The amount of antimony deposited per ampere-hour was found to be 0.833 grams (the theoretical value being 1.49).

Reduction of Titaniferous Ores.

A second paper by Mr. **Alexander Lodyguine** gave the results of experiments with the reduction of titaniferous ores. While the magnetites, hematites, etc., now available for the large iron and steel plants will, according to the author, sooner or later be things of the past, this country has inexhaustible deposits of titaniferous ores. Table I is a list of the United States Geological Survey of the more prominent localities in the United States where titaniferous ores are found, with the brief analyses of a number of representative ones:

Table I.

	Iron Oxides.	Titanic Oxides.
Iron Mountain, Col..................	62.00	10.00
Iron Mountain, Col..................	70.50	14.00
Mayhew Lake, Minn..................	80.78	12.09
Church Mine, Hunterdon Co., N. J....	69.40	10.52
Naughtright Mine, Morris Co., N. J....	89.39	7.50
Split Rock Mine, Essex Co., N. Y......	61.83	14.70
Iron Mountain, Essex Co., N. Y........	55.82	16.37
Tunnel Hill, Essex Co., N. Y..........	49.17	16.45
Little Pond, Essex Co., N. Y..........	53.15	5.21
Lake Sandford, Essex Co., N. Y........	87.20	10.73
Lake Sandford, Essex Co., N. Y........	55.64	15.77
Rockingham Co., N. C.................	66.56	13.71
Rockingham Co., N. C.................	79.78	12.08
Cumberland Hill, R. I................	40.00	15.30
Cumberland Hill, R. I................	45.62	9.93
Cumberland Hill, R. I................	58.50	3.66
Near Laramie, Wyo...................	61.99	23.40
Near Laramie, Wyo...................	73.52	23.18
Near Laramie, Wyo.....	73.01	22.43

The most important deposits of titaniferous magnetite, so far as size and accessibility are concerned, occur in the Adirondack Mountains, near Westport, Elizabethtown, and Lake Sandford, in Essex County, N Y.

And yet the ironmasters of to-day will not allow over 1 per cent of titanium in the ores they use, and many will not touch a titaniferous ore with tongs. The difficulty is that these ores smelt hard in the now existing types of furnaces, besides choking the blast furnaces. New types of furnaces and new methods of smelting are needed for the purpose.

The author referred to the splendid pioneer work done by Mr. Auguste J. Rossi, and then gave a brief account of some experiments made by himself with a Canadian ore in an electric furnace of small capacity (2 kg.). No particulars of the process are given, but the author states that from 2,000 grammes of ore he obtained, as an average, about 840 grammes of metal. The amount of iron in 2,000 grammes of ore was 1,006 grammes, and the amount of titanium 197 grammes, so that the loss in iron is equal to 16.5 per cent, and in metal to 30 per cent. The analysis of three different samples of iron obtained from his furnace and by his methods from the same ore are as follows: Fe 95.57, 84.70, 80.34; Si 0.40, 0.40, 1.40; S no, 0.60, 0.20; P no, 0.08, 0.09; Ti traces, 13.79, 17.22; Al traces; C "some;" total of chemically pure metal 95.57, 98.49, 97.56. The method of treatment was, of course, different in the three cases.

The analysis of the slag left after smelting the first sample showed: Fe 22.46%, SiO_2 14.77%, Al_2O_3 14.55%, CoO 8.27%, TiO_2 17.07%. From this slag ferrotitanium, very rich in titanium, can profitably be extracted.

He then gives some notes on the amount of power used, and concludes that for the reduction of one short ton of steel, or of ferrotitanium in his furnace and by his method, about 1,570 kw-hours are needed. Some figures are given on estimated cost, in comparison with the blast furnaces and with the electric furnaces reported on by the Canadian Commission.

The author thinks that in the high value of the by-products lies the real advantage of titaniferous ores. The by-products that are obtainable from reduction of titaniferous ore into steel are many, and they are all valuable.

Ferro-titanium is one of them. When it contains a small amount of titanium it is excellent for the wheels of railroad cars. It stands tear and wear a great deal better than any other steel. When it contains more titanium it is so hard that it can be used for cutting glass, etc. Titanium carbide is another valuable by-product. It "is a great deal harder than carborundum, and cannot only stand its ground in competition with it, but can even conquer a large market from its rival." It can be used in very large quantities in the metallurgy of iron and steel. The mordants and pigments of titanium are of great value and can also have a large market. The silicide, the boride and the nitride of titanium are nearly as hard as diamond, and can polish and cut any precious stone, including the diamond.

These two papers of Mr. Lodyguine were discussed together. Concerning electroplating upon aluminium, Dr. J. W. Richards called attention to the commercial value of nickel-plated or chromium-plated aluminium ware to give its surface the resistance against corrosion, which is required for many purposes. He regretted that in the second paper no details are given on the methods by which one may vary the titanium in the final product. The gist is to keep the titanium exactly in check. Mr. A. J. Rossi replied that in his experiments he never encountered any difficulty; the treatment is quite similar to that of silica.

A paper, by Mr. **Ernest Fahrig**, on the treatment of low-grade ores and tailings by electrolysis was then read by title. Since no advance copies were printed the report on it must be deferred until the publication in the Transactions of the Society.

Specific Inductive Capacity of Oleic Acid and its Salts.

Dr. **Louis Kahlenberg** gives in this paper the results of a determination of the dielectric constant of free oleic acid and of a number of its typical salts. The dielectric constants of three samples of oleic acid at 20° C. were found to be 2.50, 2.57 and 2.60 respectively. Aluminium oleate has a dielectric constant of 2.40 at 20° C., ferric oleate 2.68, copper oleate 2.80, etc. A fine sample of cotton-seed oil had a dielectric constant of 2.30; eight samples of pure California olive oil 2.60, 2.55, 2.70, 2.45, 2.60, 2.60, 2.53, 2.70.

It thus appears that oleic acid has about the same dielectric constant as the oils which represent its esters with glycerine as the base. Moreover, the dielectric constant of oleic acid must be regarded as quite low if one considers that the compound is unsaturated and that it contains the carboxyl group. With the exception of aluminium oleate, the metallic oleates measured have a slightly higher dielectric constant than free oleic acid, the results obtained apparently indicating an increase of dielectric constant with increase of the chemical equivalent of the metal used. It is of special interest that the specific inductive capacity of oleic acid is so slightly changed by the introduction of heavy metals into the compound. It would seem as though the characteristic electrical properties of the metals were almost entirely overcome by the union with the large, fatty radical of the oleic acid. It is further noteworthy that the oleates have nearly the same dielectric constant, whether in the solid or liquid state.

The Rotating Diaphragm.

A paper on this subject was presented by Professor **Wilder D. Bancroft**. In the absence of the author it was read by Dr. Carveth. If an acidified copper sulphate solution is electrolyzed between copper electrodes, and if a cylindrical graphite rod is interposed between the electrodes as a partial diaphragm, copper will be deposited on one side and oxygen developed on the other side of the graphite rod, if the impressed voltage is high enough. If we now reverse the position of the graphite diaphragm, by rotating it through 180°, the precipitated copper side will now face the cathode, and the copper will dissolve, while copper will precipitate on the graphite surface facing the copper anode. Until the copper on the surface facing the copper cathode has been completely removed, the polarizations at the two faces will balance each other and the diaphragm will be temporarily non-polarizable. The gist of Professor Bancroft's paper is that the same result may be obtained if instead of reversing the graphite diaphragm every so often, it is rotated rapidly. It will then behave like an ideally-reversible electrode. No visible amount of copper precipitates on such a rotating electrode, though a considerable proportion of the total current flows through the electrode. These facts were established with a graphite cylinder filling nearly the whole cross-section of the tank and revolving with a speed of 200 revolutions per minute.

Professor Bancroft considers the relation of the rotating diaphragm to the mercury diaphragm. In the Castner process there is a metallic diaphragm reversible with regard to sodium, and the sodium is carried from one mercury surface to the other, chiefly by a surface flow due to rocking. In the Rhodin cell the same result is obtained by centrifugal action. With a solid diaphragm there can be no diffusion and no surface flow, as the depolarization must be effected by rotation. The rotating diaphragm is therefore the analogue of the Rhodin mercury diaphragm, the difference between the two being the necessary result of one being solid and the other liquid.

Since polarization can thus be eliminated by rotation, Professor Bancroft asks whether this process is reversible and whether polarization can be made to cause rotation. Experiments made by him to detect a directional effect of a small difference of potential on a floating or suspended electrode showed, however, that this directional effect is small, and can easily be masked by surface tension phenomena and convection currents.

In the discussion Mr. Sperry pointed out that it would be interesting to determine experimentally the minimum speed for which the rotating diaphragm is reversible.

Heat of Solution of Aluminium Bromide in Ethyl Bromide.

A paper on this subject was presented by Dr. **H. E. Patten**, and was read, in the absence of the author, by Dr. J. W. Richards. The main results of the investigation of the author are as follows: The heat of solution of aluminium bromide in ethyl bromide was determined near 20° C. at six concentrations: 1.497 per cent, 23,097 calories; 2.23 per cent, 20,354 calories; 10.2 per cent, 17,750 calories; 25.5 per cent, 11,540 calories; 37.1 per cent, 7,357 calories; 41.8 per cent, 6,077 calories.

The voltage required to decompose a solution of aluminium bromide in ethyl bromide, liberating aluminium and bromine, may be roughly calculated from the thermal data of the heat of formation of anhydrous $AlBr_3$ plus its heat of solution in ethyl bromide for dilute solution. The difference between this calculated decomposition voltage and that experimentally obtained is 0.13 volt, or 23.5 per cent of the total voltage attributable to the heat of solution. But if we take the total voltage of aluminium against platinum in this solution, 2.235, instead of the decomposition voltage (2.30 volts), the agreement is better. The difference then is 0.065 volt, which is 10.8 per cent of 0.552 volt, the total voltage to be calculated from the heat of solution. The presence of a secondary action at the cathode explains this discrepancy. "These results may be taken as indicating that with proper conditions, agreement between decomposition voltage calculated from thermal data and that determined experimentally by electrolysis will be found, provided the same chemical reaction obtain in the thermal and in the electrolytic experiments."

"On the other hand, one may take the ground that the experimentally-obtained heat of solution (for 1.5 per cent solution) is 38 per cent less than that which was to be expected from the experimental determination of the decomposition point of this solution, and on this basis may maintain that thermal data do not lend themselves to the calculation of decomposition voltage."

The heat of solution when plotted as a function of dilution has much the same form as the heat of solution curve for sulphuric acid in water.

The above results show that the solution of aluminium bromide in ethyl bromide is subject to the same energy considerations as obtain in aqueous solution, although its molecular conductivity curve is radically abnormal. At the same time the molecular formula of aluminium bromide, 11 per cent in etyl bromide, was shown to be very nearly $AlBr_3$ by the freezing point method. And this concentration lies in the region of rapid deviation of molecular conductivity from the trend of curve required by the theory of electrolytic dissociation.

In the discussion Dr. Theo. W. Richards objected to Patten's statement that he has applied Helmholtz' fromula while he used apparently Thomson's rule, thus using the total energy instead of the free energy. A complete treatment of the case would involve far more experimental work.

Atomic Weights and Electrochemical Equivalents.

A paper on the interdependence of the atomic weights and the electrochemical equivalents was presented by Professor **Arvid Reuterdahl**. In the absence of the author it was read in abstract by Dr. J. W. Richards. The paper is of a highly theoretical nature.

The investigation is essentially based on the relation $r\ t = c$, where r is the number of ampere-hours per gram per unit of valence, t the atomic weight, c a constant value. This relation is represented by the author graphically by means of polar coördinates, r being taken as radius vector and t as the corresponding vectorial angle. This relation is then made use of by the author for calculations in molecular physics. The author assumes the atom as a spherical shell, capable of expanding and contracting with pressure; the volume of this spherical shell is, at all times, proportional to the mass of the atom. Some of the results desired by the author are: The calculation of the electric charge of a corpuscle, the figure for the mass of the negative gaseous ion being used for this calculation. The author concludes that the number of ions, whether gaseous or liquid, liberated by a given quntity of electricity, is a constant, independent of the kind and mass of the ion; also, the charge carried by a free ion, whether gaseous is equal to a constant quantity independent of the kind and [illegible].

[illegible] and electrical mass may then be regarded as the two [illegible] in which mass manifests itself to us. On the one hand we have attractions, kinetic energy, absorption and integration; on the other hand we have repulsions, potential energy, self-conservation and differentiation; yet these are merely the two modes in which actual mass interactions take place. The conservation of mass and, therefore, the conservation of energy follows as a direct consequence from the previous considerations."

Electric Reduction of Different Oxides of Lead

A paper by Mr. A. Lodyguine describes experiments which were undertaken to try to electrically extract lead from the lead peroxide of old storage batteries, instead of selling this peroxide to the junk dealers. He used a mixture of the following composition:

Peroxide of lead, PbO^2	200	grs.	
Chloride of sodium, NaCl	100	"	
Sulphuric acid monohydr., H^2SO^4	131,5	"	Solution of
Water, H^2O	168,5-2	"	H^2SO^4 at 30% B.

A glass jar, with lead electrodes, was prepared to receive the mixture.

In Fig. 5, *a* is the negative electrode; *b* is the positive electrode; *c* and *d* are the insulated conductors connecting the electrodes with the battery giving the current; *e* is the above-mentioned mixture, and *f* is some of the liquid part of that mixture; *g* is a cover, with a pipe, *h*, by which the gas escapes.

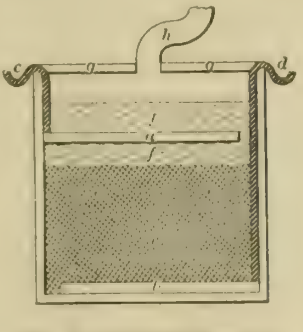

FIG. 5.—ARRANGEMENT OF EXPERIMENT

When the mixture was introduced into the jar, the liquid part poured on, and the whole mixed with a glass stirrer, the chlorine gas began to escape by pipe *h*. The setting free of chlorine soon stopped, because hydrochloric acid could not continue to be replaced by sulphuric acid in the chloride of sodium without the application of heat or of electric current.

The instant that current passed through the mixture a violent setting free of chlorine occurred, and there appeared a formation of chloride of lead in the solid mixture. After a certain time, the length of which depended on the density of the current, a formation of some sulphate of lead was noticed. After another period of time some litharge was formed: finally, at a certain stage of the process, the mixture in the jar presented the appearance shown in Fig. 6. *a* is positive electrode; *b*-PbO^2, *c*-$PbCl^2$, *n*-$PbSO^4$, *d*-Litharge; *e*-Suboxide of lead, Pb^2O; *f*-spongeous lead; g-negative electrode; *h*-liquid part of mixture. The lower the current density the higher is the efficiency.

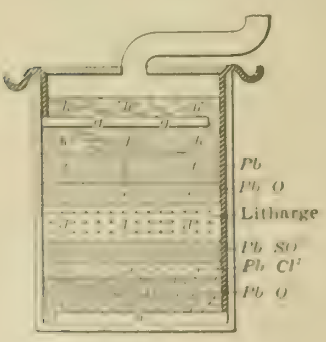

FIG. 6.—[illegible] STAGES OF [illegible] PbO

To the end of the experiment one could follow the increase and decrease of the different strata and their transformation into one another. At last only spongeous lead remained in the jar, all of the mixture having been transformed into it.

In these experiments the amount of lead deposited per watt hour was 2.1 grams, and per ampere hour 3 grams. The extraction of one short ton of pure lead from peroxide would, therefore, require about 435 kw.-hours, costing 87 cents, at a rate of $15 per kw.-year.

Utilization of Blast-Furnace Gases in Connection with the Electric Smelting of Iron.

An interesting paper on this subject was then presented by Mr. Auguste J. Rossi. After some general remarks on the importance of this problem for electrometallurgy, and especially for the manufacture of special steels in the electric furnace, the author passes over to the main object of his paper, *i. e.*, to show that whatever improvements have been already realized, or can legitimately be expected in future blast-furnace practice, tending to reduce to a minimum the amount of fuel per ton of iron, yet the amount of gases and, consequently, the available surplus, is not necessarily reduced in direct proportion to the fuel consumed. With special reference to Mr. Gayley's celebrated dry-blast run, the author emphasizes that it would be a mistake to conclude a priori that a smaller amount of coke per ton of iron necessarily gives a smaller weight of gases.

On the basis of the figures given in Mr. Gayley's report, Mr. Rossi calculates a total of 11,284 pounds gases per ton of metal, with 1,706 pounds of coke; the same amount of gases is practically given by L. Bell for the Edgar Thomson furnace, with moist air, for a consumption of 1,881 pounds of coke. This calculation shows that the amount of coke is not necessarily a criterion of the amount of gases, though it certainly enters into the calculation as an element, but that the composition of the ore smelted has a great deal of influence.

According to the latter, more or less limestone will be required giving out CO_2 in the gases, and we cannot conceive a blast furnace smelting chemically pure oxide of iron. If the ore is richer and the percentage of gangue is low, it will contain more oxide of iron, more oxygen, and will require more carbon for the reduction. As the heat necessary to reduce 1 kg. of iron from its oxides, to impregnate it with carbon, silicon and phosphorus, and to melt the product represents *several* times the number of calories required for the gangue of the ores and charges to be transformed in slag and melted and for its volatile matters to be gasified, for these reasons, with such *richest ores* as we can practically assume would be smelted, it is not likely that this amount of gases could fall to less than some 8,000 pounds to 9,000 pounds per ton of iron, even were the consumption of coke to fall as low as 1,500 pounds or below.

The figure of 9,000,000 thermal units per ton, assumed by some writers as *the total heat of the gases* for a consumption of 1,700 pounds of coke, would correspond at a heat value of these gases at 1,280 thermal units per pound to only 7,000 pounds of gases per ton of iron, a figure difficult to establish in the ordinary condition of ores; at any rate, for Mr. Gayley's remarkable run, even with an amount of coke of only 1,706 pounds per ton, as calculated from his charges, we have still over 11,000 pounds of gases. But let this figure of 9,000,000 thermal units, as total heat of the gases per ton of iron, stand as it is, and let us utilize as available, as the same writer says, only 52 per cent of it, or 468,000 thermal units. At 10,000 thermal units per effective horse-power-hour in gas engines we come to his figure, 468-horse-power-hour, as available surplus per ton, or, for 2,240-ton hour, to over 1,000,000-horse-power-hours surplus. The author considers himself justified, in conclusion, to assume that this figure is the minimum possible in the best provisions of the blast-furnace practice of the future as to consumption of coke per ton of iron and best condition of ores and steel. It is one which, if only but partly utilized, opens new and great possibilities for electric smelting.

This paper elicited a very lively discussion. Mr. E. R. Taylor agreed with the author of the paper that here is a source of power available for profitable development. Mr. H. S. Blackmore mentioned another example of the utilization of what heretofore had been wasted. On the other hand, the gas-engine proposition in the whole was strongly attacked by Mr. Herbert H. Dow (Dow Chemical Co., Midland, Mich.), who emphasized its complication and the high investment involved. He is entirely in favor of steam engines. He thinks the term "waste gases" of blast furnaces is a misnomer, since they are used under boilers, etc. Mr. E. A. Sperry seconded him, and also emphasized that capital is expensive in this country, and that the interest factor and the high labor charge speak against gas power; coal and water power are cheaper. Dr. Joseph W. Richards, in a detailed reply to these statements, agreed that the commercial conditions are different in Europe and in the United States, and this fact accounts for the slow development of the utilization of blast-furnace gases in this country. Nevertheless, the development is going on, since the method has its undeniable commercial advantages. It is true, we have now cheap fuel, but we must be economical. Not so long as fifty years ago one could see blast furnaces, with open top, lighting at night time the country in the neighborhood; hence the name, waste gases, and this name has remained, although these gases are now used under boilers, etc. Dr. Richards stated that the Pittsburg plants will be equipped with the new system within three or four years. Plans and drawings have already been prepared for the most important plants of the United States Steel Corporation. The speaker then explained the importance of this development for electric-furnace steel, and predicted that for making the finer qualities of steel the crucible process will be replaced by the electric furnace within the next five years.

Thursday Session.

This session was again held in Lowell Hall of the Massachusetts Institute of Technology. Most of the papers presented at this session were illustrated by experiments, lantern slides or exhibits. Mr. A. J. Rossi exhibited a very interesting and full selection of samples, which he has produced in the electric furnace, especially in connection with the problem of smelting titaniferous ores.

Electrostatic Method for Separating and Concentrating Ores

A paper by Professor Lucien Ira Blake described his and L. N. Moscher's electrostatic separation process. In the absence of the author the paper was read by Mr. C. E. Locke. The process depends on the difference in the electric conductivity of the constituents of a mechanical mixture of various materials, and on the fact that, according to the conductivity, the electrostatic charge required by different materials from the same source of electric charge in the same time is different. Thus, if the mixture consists of a finely-crushed non-conductor, and of a finely-crushed conductor, and is brought in contact with a charged surface, the latter will receive a charge, while the former remain uncharged. Now, electrostatic repulsion of the charged particles from the charged surface will result, while the uncharged particles are not repelled. In this way the charged particles are separated from the uncharged ones, and this fact is made use of in the electrostatic separator.

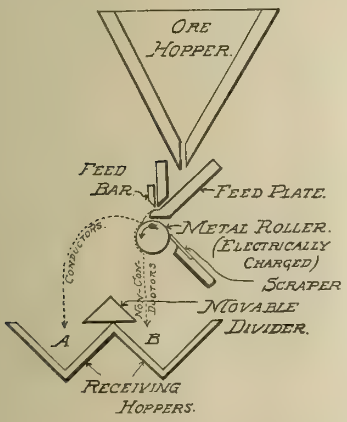

FIG. 7.—ELECTROSTATIC SEPARATOR.

The author first gave some mathematical formulas, which show that the process depends for a given separator with a given potential, on the electric resistance of the particles and on the electrostatic capacity of the particles in the position of contact. The resistance again is the sum of the internal resistance and surface resistance. The time in which a charge is acquired depends upon both the resistance and the capacity.

There is no possible way of predicting the action of the process on any given ore from a description or even from an analysis, but an actual trial must be made with a sample of ore. It is, however, possible to classify the minerals roughly, and the author gave the following list as a guide to their usual behavior:

Conductors.	Non-conductors.
Native metals.	Quartz.
Pyrite.	Calcite.
Chalcopyrite.	Limestone.
Galena.	Porphyries.
Graphite.	Slates.
Molybdenite.	Sandstones.
Copper glance.	Garnet.
Silver glance.	Spinel.
Gray copper.	Zinc blende.
Most sulphides.	Zinc carbonate.
Most copper minerals.	Barite (heavy spar).
Most iron minerals.	Gypsum.
Most silver minerals.	Granite.
Most manganese minerals.	Fluor-spar.
Tellurides.	Most silicates.
Black sands.	Most gangue rocks.

In a general way most sulphide minerals and most copper, iron, etc., minerals are conductors, while the gangue, waste minerals, quartz, etc., are non-conductors. Blende (zinc sulphide) is an exception to this general rule, being a non-conductor; and this is important for practical applications.

The process in practice requires, first, crushing; second, drying; third, suitable feeding to the charged surface; fourth, collection and removal of the separated products.

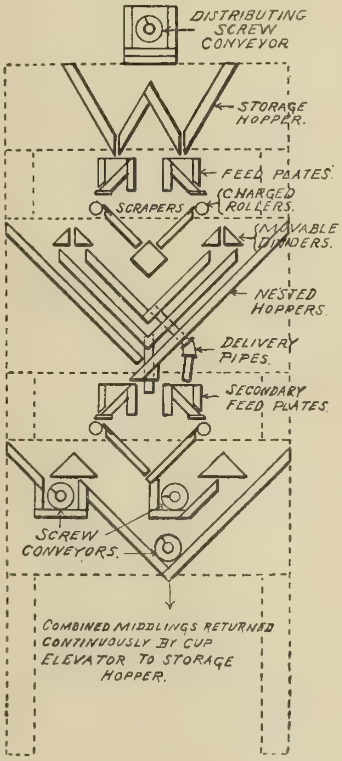

FIG. 8.—ELECTROSTATIC SEPARATOR.

Coarse crushing is preferable, but it must be fine enough to liberate the crystals, one from another. All materials to be delivered to the separator must be absolutely dry. This almost invariably means the use of heat, and the use of a direct-heat rotary dryer with boiler setting has been most satisfactory.

The construction of the separator itself is clearly shown diagrammatically in Figs. 7 and 8, while Fig. 9 gives the exterior view of an electrostatic separator. All these pictures are self-explanatory.

The standard type of separator, as built at present, has a nominal capacity of 12 tons in 24 hours. This varies, of course, within wide limits, depending upon the size of crushing, dust present, percentage of products, etc. The machine is chiefly built of wood in two similar sections, and is really two machines, standing side by side or connected in tandem, to fill a long, narrow space.

The size of the machine is 12 feet by 12 feet by 7 feet (with an elevator 5 feet higher, or 12 feet over all).

The voltage required is 350,000, and is furnished by a special induction machine. The leakage is considerable, but the total power expanded per machine is only 1 horse-power.

Most of the electrostatic separators now in practical use are used for separating pyrite or marcasite from zinc blende. This is done without roasting.

Not only does the process remove the iron minerals from zinc blende, but also lead, copper and other impurities. This means that when complex sulphide ores are concentrated on jigs or tables as a preliminary step to separating the middlings, it is not necessary to make a low-grade lead concentrate, carrying a high percentage of zinc and iron, in order to get a middling product free from lead. Lead can be freely sent over to the middling product if desired, since it will be removed, [illegible]

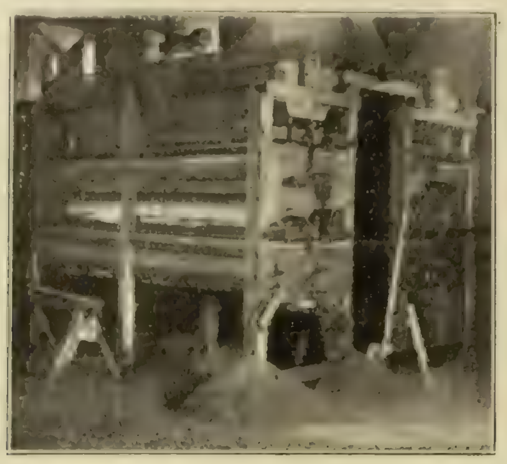

FIG. 9.—EXTERNAL VIEW OF ELECTROSTATIC ORE SEPARATOR

TABLE II

Description	Products	% Zinc.	% Lead	% Iron	Remarks
Leadville, Col., Zinc middlings [illegible]	Orig. Iron. Zinc.	30.48 9.04 51.20	3.21 8.00 0.62	21.60 35.11 6.40	20 mesh. (Combined iron 4.87%)*
[illegible]	Orig. Iron Zinc.	49.53 2.92 62.31	2.52 9.92 trace	10.78 37.14 1.57	10 mesh. Lead practically all removed
[illegible] W., Zinc middlings [illegible]	Orig. Iron. Zinc.	29.15 5.07 58.40	5.16 8.40 0.23	20.90 38.88 2.16	6 mesh. Iron sold to acid burners
Silverton, Col., Zinc middlings [illegible]	Orig. Iron. Zinc.	23.12 9.04 49.73	8.00 15.56 2.08	23.87 30.35 4.77	24 mesh*
Idaho lead and [illegible]	Orig. Lead. Zinc.	25.87 4.45 45.11	20.40 50.58 3.81	12.02 16.14 7.40	High silver*
		Copper			
[illegible] in garnet	Orig. Conc Tlgs	3.20 12.94 0.40			12 mesh. Saving 81% of the copper. Crude ore; no dust removed
[illegible]	Orig. Conc Tlgs	4.60 [illegible] 0.79			20 [illegible] of the copper. [illegible]
[illegible]	Orig. Conc Tlgs	2.68 38.70 0.27			30 mesh. Saving 76% of the copper
[illegible] Chalcopyrite [illegible]	Orig. Conc Tlgs	[illegible] [illegible] [illegible]			[illegible] copper
			[illegible]	[illegible]	
[illegible]	[illegible] [illegible] Tlgs	1.14 4.20 0.44	[illegible] [illegible] 1.27	[illegible] [illegible] 0.14	[illegible]

Those lots marked * also contained considerable gold [illegible] largely followed the lead and iron.

Some of the results obtained in commercial practice on various kinds of ore are given in Table II.

The paper was briefly discussed by Messrs. J. W. Richards and C. P. Townsend.

COLLOIDS.

A lecture on this subject, with a series of very interesting experiments, was held by Dr. W. R. WHITNEY. To bring an interesting and still little understood problem "before a new crowd" was the object of the lecturer, who pointed out that we have here to do really with an electrochemical problem, since we have here migration of material under the influence of an electric current, precipitation by electrolytes, etc. Dr. Whitney gave a general summary of what we know at present on colloids.

Colloids are apparently solutions or suspensions, but they are characterized by the feature that they do not exert a measurable osmotic pressure. Further, two classes of colloids may be distinguished: positive and negative colloids. A positive colloid precipitates a negative colloid and vice versa, but a positive colloid does not precipitate a positive colloid, and a negative colloid does not precipitate a negative colloid. Under the influence of an electric current the two classes of colloids migrate in opposite directions.

In most cases we have to do with particles in suspension, perceptible by the microscope. They are then seen to be in perpetual movement, like the picture we have formed of kinetic motion of molecules.

Colloids were shown to migrate under the influence of a current in one or the other direction. The lecturer described the precipitation of colloids by electrolytes, and gave a summary of various methods of preparing colloids. For instance, by volatilization of a metal under water; thus, with two platinum electrodes slightly touched under water, without fusing the terminals together, a colloidal solution of platinum in water is obtained. Further, production of a colloidal solution by putting a powdered substance into water. Another method is indicated by the reaction $Hg\,Cy_2 + H_2S = Hg\,S + 2\,H\,Cy$; in this method it is important that no electrolyte is present, as otherwise the colloid would be precipitated. All these explanations were illustrated by experiments.

The author then discussed the biological applications of colloids (blood, saps of trees) and the application for dyeing of textile fabrics; here it is important to produce the colloid inside the fabrics, then it cannot get out, since it does not exert an osmotic pressure and has no diffusing power.

The paper was discussed by Messrs. Franklin, Richards and Carveth.

AN ELECTROLYTIC SWITCH.

A paper on this subject, with experiments, was presented by Mr. WILLIAM SMITH HORRY.

The various properties of the aluminium cell are, besides its symmetric peculiarity, that it presents a high resistance to the passage of an electric current, it possesses capacity and it leaks. These qualities may be turned to account for the purpose of preventing those arcs that occur when a current, possessing inductance, is interrupted.

As all continuous currents possess more or less inductance, the voltage rises across the break when a switch carrying such a current is opened. This not only causes an inconvenient flash at the switch, but endangers the insulation of the circuit.

But if a suitable electrolytic cell be inserted across the break and the circuit then broken, there is no flash at the switch, and the increased voltage will force a current through the cell instead. The insulation of the cell then breaks down, but quickly re-establishes itself. After that the mains may be entirely disconnected without any spark.

To obtain the best results in this way an electrolyte for the cell should be chosen that substantially shuts off all current at the working voltage but that allows considerable leakage to occur with any increased pressure, a cell that has a critical voltage a little above the line voltage. The most simple diagram of the circuits is that shown in Fig. 10. The circuit is represented as a line containing an electromagnet; there is a double-pole switch for the line, a rheostat contact and an aluminium cell. If the switch is closed no current will flow except that which leaks through the cell, but if the rheostat arm is moved to the left then the full current will pass. To shut off the current the rheostat is returned to the position shown, then the switch is opened.

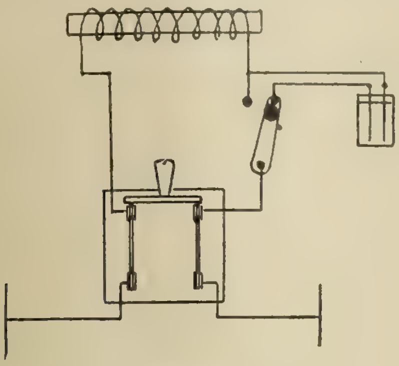

FIG. 10.—ELECTROLYTIC SWITCH.

But the cell possesses capacity, and, if it be a large one intended to suppress a powerful arc, there will be a spark at the arm contacts when the terminals of the cell are connected. To make the circuit sparklessly the rheostat must be moved before the switch is closed. If the arm be moved first, both in opening and closing the circuit, there will be no spark at make, due to capacity, and no arcing at break, due to self-induction. This may be done with one movement of the switch handle, if there are provided three knives to it, the middle one having a more extended movement than the other two, so that it operates always in advance of the outer blades. Fig. 11 is a diagram of the circuits.

The construction of various cells of different sizes for different purposes was described. If such a cell be left unused its insulation slowly diminishes, but if used once a day that tendency can hardly be noticed. On the other hand, the cell insulation increases with frequent use up to the critical point. For long-continued and rapid use a larger cell is necessary than for infrequent use, so that the heat generated by the continued arcing through the cell shall be radiated more rapidly.

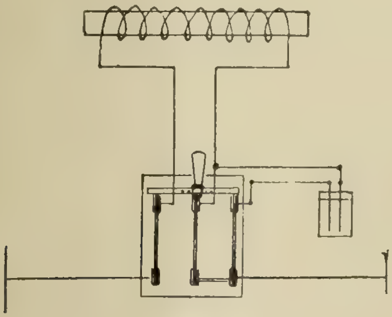

FIG. 11.—ELECTROLYTIC SWITCH.

The paper was briefly discussed by Messrs. Richards, Smith and Dow; the latter stated that he patented a similar device several years ago.

SILICON.

An interesting paper on this subject was presented by Mr. F. J. TONE. He began to describe the variety of products which may be obtained from a reaction between silica and carbon. To get a certain product alone it is all-important to provide a method of very closely regulating the temperature. He then described his own process for making silicon, which is operated at the works of the Carborundum Co., of Niagara Falls.

He mixes 60 parts of silica and 24 parts of carbon, according to the reaction:

$$SiO_2 + 2C = Si + 2CO.$$

The mixture must be made in very exact proportion. The furnace to be used must be specially designed in view of the fact that the temperatures of reduction and volatilization of silicon are very near together. For details of the construction of Mr. Tone's furnace the reader may be referred to the description and illustration on page 111 of our volume II.

The author then discussed the properties of silicon, samples of which were exhibited. The melting point is 1,430° C. In the neighborhood of the melting point it oxidizes in air, while at lower temperatures this is not the case. Silicon has a specific gravity of 2.34, and is between 6 and 7 in the scale of hardness.

Metallic silicon may become very important in the metallurgy of iron and steel, and is already very largely used in these industries in form of ferrosilicon and silicon-copper. Silicon may also be used to a certain extent as a substitute for aluminium in Goldschmidt's thermit reaction. Silicon is also suitable as a resistance material.

In the discussion Professor Elihu Thomson spoke of the great importance of the subject. It may find many uses. Silicon is an excellent resistance material when granulated and placed in a cylinder; by simply varying the pressure it is possible to change the resistance from 300 to 500 to 1. When the current density gets too high, however, the silicon particles are fused together. The resistance should, therefore, be artificially cooled. The specific resistance of silicon is two or three times that of carbon. Professor Thomson also spoke on casting silicon.

Mr. H. S. Potter stated that Mr. A. B. Albro has succeeded in developing a satisfactory process for substituting the cheaper silicon for the expensive aluminium in the thermit reaction of Goldschmidt. No further details were given in view of pending patents. Mr. Potter also mentioned that at high temperatures silicon is easily attacked by carbon dioxide.

MICROSTRUCTURE OF SILICON AND ALLOYS CONTAINING SILICON.

A paper on this subject was presented by Mr. A. B. ALBRO. The object was "not to propose or to defend any theories, but to show photographically recorded facts." Accordingly, the paper was a discussion of some 46 photomicrographs, which were shown by lantern slides, and which were extremely interesting. For this reason, however, it is impossible to abstract the paper satisfactorily without the photographs. We hope to be able to publish the paper in full with the photographs in a future issue. We will therefore simply state here that the author dealt successively with the microstructure of silicon, ferrosilicon, coppersilicon, showing in each case a marked characteristic influence of silicon, which can easily be detected microscopically. Two specially interesting statements were incidentally made; one to the effect that "a novel silicon process has been devised for Mr. George Westinghouse, by Mr. Henry Noel Potter," the process being not yet disclosed; the other statement referring to the observation made by Mr. Potter, that when silicon and silicon carbide are saturated with each other, the resulting material, called "carbosilicon," exhibits remarkable properties. Analysis proves this to contain the element and its carbide in proportions approximating very closely the formula $SiC + Si =$ (perhaps) Si_2C. The material is non-porous, harder than corundum, and so tough that a three-inch cube withstood heavy blows from a sledge on all its faces without fracture. Another interesting disclosure was the proof that fluid silicon, at temperatures near its melting point, does not dissolve carbon or react therewith to produce silicon carbide.

The paper was briefly discussed by Drs. Carveth and Richards.

The Mercury Arc.

In a paper on this subject, illustrated by numerous interesting experimental demonstrations, Dr. E. **Weintraub** gave a review of the work done by him in the Research Laboratory of the General Electric Co. As the author pointed out, the subject is of special interest to the electrochemist, "whose science can be in the main properly defined as the science of conductivity of a particular kind of matter, namely, of solutions and molten salts. The ionic theory of conductivity that had its birth in electrochemistry found afterwards fruitful applications in the theories of conductivity in gases. Any increase of our knowledge in this last named branch is in its turn apt to react on the general structure of that theory, and eventually influence the electrochemical ideas themselves."

The author first discussed the starting of an arc in metallic vapors. The distinctive feature of the conductivity of gases and metallic vapors, in contradistinction to that of metals and electrolytes, consists in that under ordinary conditions, in absence of exterior ionizing agents, the current itself has to create the material which is to carry it from one electrode to the other. As this material does not exist at the beginning, some special means must be used to start the discharge, and the author shows that the discharge must derive its carriers, at least in the immediate neighborhood of the cathode, from the material of that cathode. The application of a moderate electromotive force is of itself insufficient to start the process by which these carriers are formed out of the material of the cathode. The new fact discovered was that this formation of carriers (the "ionization" process) is started and the arc discharge made possible if a spark or a small arc is produced at the surface of the cathode. Several designs, based on this fact and using an auxiliary cathode for starting, were described. A consequence of the above fact is that Ohm's law does not hold in vapor arcs. The space separating the electrodes has an important influence on the velocity with which the starting takes place, but this influence is mainly of a negative character. The space must present as little hindrance as possible to the flow of ions starting from the cathode surface. Accordingly, the degree of vacuum must be the highest obtainable, and the more careful the exhaustion the more instantaneous is the starting.

In contradistinction to the cathode the anode plays no role whatever in the starting process. It receives the carriers of the current without any previous excitation.

The author then passed over to a discussion of the conditions of stability of the arc. Under ordinary circumstances there is a bright spot on the surface of the mercury cathode, which spot is continually wandering about on that surface. The wandering of this spot has not found as yet any very satisfactory explanation. The spot appears to be the place where the production of ions takes place, and the continuous motion of that spot means a disturbance of the ionization process and a danger to the existence of the arc. The wandering of the spot can be avoided in two different ways: either by making the surface of the cathode small, of the same magnitude as that of the spot, or by having a wire of iron or platinum project above the surface of the mercury. In this latter case the cathode spot is centered on the mercury surface right around the metallic wire.

The conductivity of the arc depends on the relative amounts of ionized and inert mercury vapor. If a sufficient condensing [illegible] is provided, so that the pressure of the ordinary mercury vapor volatilized from the cathode is kept down to a certain value, the conductivity of the arc is almost exactly proportional to the current. (The current creates its own ions.) [illegible] is accordingly almost independent of the current.

[illegible] the ionization process at the cathode must be continuously maintained, the difficulties with an alternating current are evident, since with every change of polarity the ionization process at the cathode dies out, and moderate e. m. f's are not sufficient to start it again. If an alternating e. m. f. of moderate value is to maintain an arc in metallic vapors, one metallic electrode must permanently keep its negative sign, notwithstanding the perpetual change of polarity. Two arrangements for this purpose were described by the author.

In extending his researches to the arc discharge in vapors of alkali metals the author found that the regularities found for mercury vapors hold good for all other metallic vapors.

In the concluding part of his paper the author discussed the mechanism of the ionization process at the cathode. Disintegration is characteristic of the cathode of a metallic arc in an exhausted space. A transfer of matter either does not take place at all, or is so small as to be of the order of magnitude that would be required by the value of the electronic mass accepted in the modern theories. The cathode is the only space where the current-bearing material is formed. The temperature of the cathode is not the cause of ionization; it is probably in some way connected with the boiling point of its material, and is a result of the ionization process rather than the cause of it.

The author finally offers the following working hypothesis: He assumes that the mechanism of the conduction in the arc is similar to that in metals, and that an arc is, electrically considered, in the main a flow of negative electrons directed from the cathode toward the anode. While in the metal the number of electrons is a given quantity and the conductivity accordingly constant, the number of electrons crossing the surface of the cathode and entering into the arc must be proportional to the current. Accordingly, the conductivity of the arc is proportional to the current. The process of starting an arc serves to initiate the process by which the electrons are driven out from the metal into the space. The starting is probably produced by purely electrical causes. If this process, by means of which the negative electrons are projected out of the metallic mass, is started, it goes on of itself so long as a sufficient amount of energy is applied. The disintegration of the cathode is a mechanical accompaniment of this electrical process. The light emitted by the arc is not due directly to the radiation of the electrons. These negative electrons are probably the same in all arcs, and the light emitted changes according to the molecular material through which these electrons move.

Electric Arcs.

A lecture by Dr. W. R. **Whitney** on this subject, with experimental demonstrations, in which different arcs were projected on a screen, formed the able conclusion of this highly interesting session.

Dr. Whitney first referred to the unsuccessful attempts of applying something like Faraday's law to the arc. The arc starts at the cathode, and material is driven off from the cathode, but it does not go to the anode. He then showed various arcs; two solid carbon electrodes, plain carbon arc; electrodes, impregnated with salts (calcium fluoride and chloride); magnetite and carbon arc; iron and copper arc. The fact that it is the cathode which furnishes the carriers of the current was made evident by reversing the current between two unequal electrodes.

A vote of thanks was accorded to all who had shown courtesies to the Society and had contributed to the great success of the meeting. Then the meeting adjourned. The following three papers, which had been read by title, are abstracted from advance sheets:

A Low Voltage Standard Cell.

A paper, by Dr. **Geo. A. Hulett**, deals with the following combination:

Cadmium, cadmium sulphate, cadmium amalgam, 5 to 13 per cent. It is based on the experimental fact that the electromotive force of cadmium amalgams, with concentrations within the above limits, has a constant value. Observations, began in May, 1904, by the author on this combination indicate that its electromotive force remains very constant, while cells set up at different times and with materials from different sources agree remarkably well, considering that one electrode is a metal in a solid state. The construction of the cell is described in detail and results of tests are given.

The e. m. f. of this cell is 0.05175 international volts at 20° C. The temperature coefficient is negative, = — 0.000244 volt per degree, "and practically linear." The temperature coefficient is higher than could be wished, but the cell in the construction of the author takes up its true value in a remarkably short time, when brought from one bath to another, generally in less than 15 minutes. In a bucket of water the cell follows any ordinary temperature change as readily as a mercury thermometer.

The small e. m. f. is an objection for a primary standard, but there is a very considerable amount of work done in which small e. m. f's are to be accurately measured—for instance, temperature measurements with a thermocouple.

The fact that the e. m. f. and temperature coefficient are not only independent of the concentration of the electrolyte but even of the presence or absence of the solid salt, make the cell particularly interesting. The cell is strictly reversible at both electrodes, but it does polarize and is only to be used in the Poggendorf compensation method. The internal resistance is very small.

Electrolytic Precipitation of Nickel on Nickel.

In a paper on this subject Mr. **Ralph C. Snowden** refers to the well-known difficulty experienced in technical electroplating with replating nickel-plated work. It is often claimed that it must be entirely freed from the previous deposit of nickel, or the subsequent nickel will not adhere. The reason is that the old nickel coating has become "passive" and is coated with a film of some oxide. It is not possible to remove this film by ordinary means, but this may be accomplished by making the passive metal the cathode in an acid solution and passing a fairly large current for a short time. This will cause a heavy evolution of nascent hydrogen, part of which will be taken up by the metal, and the oxide will be reduced. Quick washing and transferring to the plating bath gives an active metallic surface, on which an adherent deposit can then be successfully plated. This was found to be correct by experiment.

For the preliminary treatment of the nickel cathodes a 3-normal solution of hydrochloric acid was prepared, and a current density used of 8 amperes per square decimeter. The electrodes were both of nickel and were placed about 4.25 cm. apart. The run lasted 4 minutes. Hydrogen was evolved copiously at the cathode. As soon as sufficient treatment had been given the cathode it was washed in clean water and plunged into the regular plating solution, which was made up:

Nickel ammonium sulphate, 6 aq...........	80 g.
Water	1 liter.

The other conditions: Area cathode, ¼ dm^2; current density, 2.0 amp./dm^2; voltage, 3.8 volts; temperature, 18°; time, ½ hour.

In case a rolled nickel anode is used, the solution must be kept neutral or only very slightly acid by means of ammonium hydroxide. Too great acidity causes the precipitation of a spongy deposit. In the experiment of the author the deposit came out perfectly smooth and so adherent that it could be burnished and scratch-brushed with impunity. The "active" state of nickel is very unstable.

Diffusion in Electrolytes.

A paper, by Mr. **Carl Hambuechen**, describes an optical method for observing diffusion in electrolytes. The author first corrects an erroneous conception sometimes held, that is, that in an electrolytic vat there is a gradual change of density from the anode to the cathode. As a matter of fact, whatever differences of density there may be at the immediate surface (or in the pores) of the electrodes, there is almost no variation in density on a horizontal line extending across the cell. In all active electrolytic cells there is a continual systematic movement of the electrolyte aside from that caused by gases which may be liberated.

For instance, if a solution of iron sulphate is electrolyzed between iron electrodes, the solution becomes more concentrated immediately at the surface of the anode, and this heavy solution sinks to the bottom of the cell. Simultaneously the solution at the surface of the cathode becomes more dilute and rises in a steady stream to the top. The whole solution then becomes stratified by horizontal layers of different concentration, laying above each other. They may be shown by the following optical method:

The cell is made of such size that it can be placed as a slide in a stereopticon lantern, and, upon focusing, a good image of the cell is thrown upon the screen. Even if the boundary lines between liquid layers of different concentration cannot be seen by the naked eye within the cell, they are clearly shown on the screen, on account of differences in the refraction of the light as it passes through the solution of non-uniform density.

By studying the stratification of an electrolyte produced in this manner a number of the difficulties encountered in the electro-deposition of metals can be explained. The rapid rising of the cathode solution explains the corrugated appearance which metallic deposits sometimes assume, and the non-uniformity in the density of the electrolyte explains the differences in the appearance of the deposits at different parts of the cathode.

By tracing out in a cell of this sort the paths taken by the solution, methods can be easily devised for altering such paths so as to meet practical requirements by means of suitable baffle plates or by an arrangement of electrodes.

In the following we give an alphabetical list of those members and guests who registered at the meeting, but may say that quite a number of those present did not register:

A. B. Albro, N. L. Amster, Henry S. Blackmore, Wm. Hand Browne, Jr., John S. Browning, Herbert Buckley, Henry S. Carhart, H. R. Carveth, Alan A. Claflin, T. J. Clexton, G. W. Coggeshall, Geneva B. Davidson, Gustavus Drobegg, Frank D. Easterbrooks, R. Fleming, W. S. Franklin, A. H. Gill, H. M. Goodwin, M. L. Griffin, Mrs. M. L. Griffin, H. E. Heath, Carl Hering, Wm. Smith Horry, E. J. Hutchinson, Alois von Isakovics, W. McA. Johnson, Yogoro Kato, A. E. Kennelly, Edward F. Kern, W. H. Kimball, Chas. A. Kraus, F. Austin Lidbury, A. Lodyguine, H. T. Matthew, A. C. Melcher, Henry Noel Potter, Jos. W. Richards, Mrs. Jos. W. Richards, E F. Roeber, Auguste J. Rossi, Philip B. Sadtler, S. S. Sadtler, F. E. Schmitt, Fred'k F. Schuetz, Edmund T. Smith, H. M. Smith, W. O. Snelling, C. C. Speiden, E. A. Sperry, H. P. Talbot, Edward R. Taylor, Mrs. Edward R. Taylor, M. DeK. Thompson, Jr., Elihu Thomson, F. J. Tone, Clinton Paul Townsend, Wm. H. Walker, Olive M. Wallace, Ray Hill White, S. E. Whiting, W. R. Whitney, H. W. Wiley, C. Irving Zimmerman.

Nickel Free from Carbon.—We have received from Mr. E. Stütz, general manager of the Goldschmidt Thermit Co., of New York City, the first regulus of pure nickel free from carbon, produced in this country, with the aluminothermic reaction from United States materials. This carbon-free nickel is mainly applied to convey to cast iron certain mechanical properties highly desirable in cast-iron vats used in various chemical industries, especially for the evaporation of caustic alkalies.

Electric Furnace Steel.—We have received from Mr. E. A. Colby and Dr. Leonard Waldo part of the first steel made in the electric induction furnace in the United States. Mr. Colby's original patents, taken out in 1890, for this type of furnace, were referred to on page 134 of our last issue under correspondence.

Metallurgical Calculations. III.

By J. W. Richards, Ph.D.
Professor of Metallurgy in Lehigh University.

The Use of the Thermochemical Data.

Simple Combinations.

If the problem is the simple calculation of how much heat is evolved in the combination of a given weight of one element with another, the factors needed, the heat evolved per unit weight of substance combining, are obtained by a simple division from the thermochemical data given. Thus, suppose the question to be the total heat evolved in Problem 3, in the oxidation in a Bessemer converter of

100 kilos. of carbon to carbonic oxide.
200 kilos. of carbon to carbonous oxide.
50 kilos. of manganese to MnO.
150 kilos. of silicon to SiO^2.
500 kilos. of iron to FeO.

The solution would be

$$100 \times \frac{29,160}{12} = 100 \times 2430 = 243,000 \text{ Calories}$$
$$200 \times \frac{97,200}{12} = 200 \times 8100 = 1,620,000 \quad ''$$
$$50 \times \frac{90,900}{55} = 50 \times 1653 = 82,650 \quad ''$$
$$150 \times \frac{180,000}{28} = 150 \times 6429 = 964,350 \quad ''$$
$$500 \times \frac{65,700}{56} = 500 \times 1173 = 586,500 \quad ''$$
$$\text{Total. } 3,496,500 \quad ''$$

Complex Combinations.

If the problem is a step more complex, that is, includes the combination of compounds with each other to form a more complex compound, the molecular weights are still our guide, together with the thermochemical data given. If, for instance, the question above solved is complicated by the further requirement, to add the heat evolved by the formation of the slag, that is, of the MnO and FeO with SiO^2 to form silicate, we may calculate the heat of union of FeO and MnO with SiO^2 as follows:

$$(Mn, Si, O^3) = 276,300 \text{ Calories.}$$
$$\text{But} \quad (Mn, O) = 90,900 \quad ''$$
$$\text{and} \quad (Si, O^2) = 180,000 \quad ''$$
$$\text{therefore} \quad (MnO, SiO^2) = 5,400 \quad ''$$
$$\text{or, per kilo. of MnO} = \frac{5,400}{71} = 76 \text{ Calories.}$$

Similarly

$$(Fe, Si, O^3) = 256,300 \text{ Calories.}$$
$$(Fe, O) = 65,700$$
$$(Si, O^2) = 180,000$$
$$(FeO, SiO^2) = 10,600$$
$$\text{or, per kilo. of FeO} = \frac{10,600}{72} = 147 \text{ Calories.}$$

Therefore the additional heat of formation of the slag would be

Wt. MnO = 64.5 kilos × 76 = 4,902 Calories.
Wt. FeO = 643 " × 147 = 94,521 "

Sum = 99,423 "

Similar principles of calculation apply to all the oxygen containing salts. Thus, if from the heat of formation of any sulphate we subtract the heat of formation of SO^3, and also of the metallic oxide present, the residue is the heat of combination of the metallic oxide with SO^3,—the weights involved being always those represented by the formulæ. Thus, calling MO any metallic oxide, we may express the principle as follows:

$$(MO, SiO^2) = (M, Si, O^3) - (M, O) - (Si, O^2)$$
$$(MO, SO^3) = (M, S, O^4) - (M, O) - (S, O^3)$$
$$(MO, CO^2) = (M, C, O^3) - (M, O) - (C, O^2)$$
$$(3MO, P^2O^5) = (M^3, P^2, O^8) - 3(M, O) - (P^2, O^5)$$
$$\text{etc.} \qquad \text{etc.}$$

Example:—What is the heat required to calcine limestone?

$$(CaO, CO^2) = (Ca, C, O^3) - (Ca, O) - (C, O^2)$$
$$= 273,850 - 131,500 - 97,200$$
$$= 45,150 \text{ Calories.}$$

This is the heat required to split up $CaCO^3$ (100 parts) into CaO (56) and CO^2 (44); the heat required is therefore

45,150 ÷ 100 = 451.5 Calories per kilo. of $CaCO^3$ decomposed
45,150 ÷ 44 = 1026. " " " CO^2 driven off.
45,150 ÷ 56 = 806. " " " CaO remaining.

And either of these quantities may be used, according to convenience in working the problem.

Double Decompositions.

If the thermochemical problem involves the simultaneous decomposition of one or more substances and formation of one or more others, then the chemical equation should be written, and a thermochemical interpretation given of all the energy involved in the passage from starting compounds to products. Every chemical equation can be thus interpreted, if the heats of formation of all the compounds represented in it are known. We obtain the net energy of the reaction by assuming that all the substances used are resolved into their elements, and that all the products are formed from their elements; the first item is therefore to add together the heats of formation of all the substances used, starting with their elements, and by changing the algebraic sign of this sum we have the heat necessary to decompose the substances used into their elements; the second item is similarly found by adding together the heats of formation of all the substances formed; the difference between these two items is the net energy of the reaction. In making these summations, regard must, of course, be paid to the number of molecules of each substance concerned, as shown in the reaction; because the tabulated data are the heats of formation of one molecule only; and to the algebraic sign of the heats of formation.

Examples: (a) What heat is evolved when dry ferric oxide is reduced by aluminium, in the Goldschmidt "Thermite" process:

$$Fe^2O^3 + 2Al = Al^2O^3 + 2Fe$$
$$-195,600 \qquad +392,600$$
$$+197,000 \text{ Calories.}$$

(b) Write the equation showing the reaction of metallic sodium on water in excess

$$2H^2O + 2Na = 2NaOH + aq \text{ (excess water)}$$
$$-2(69,000) \qquad +2(112,500)$$
$$-138,000 \qquad +225,000$$
$$+87,000 \text{ Calories.}$$

(c) What heat is evolved in the reaction of water on calcium carbide to form acetylene gas?

$$CaC^2 + 2H^2O = CaO^2H^2 + C^2H^2$$
$$(-6250) - 138,000 \qquad +215,600 + (-54,750)$$
$$-131,750 \qquad +160,850$$
$$+29,100$$

The last case leaves out the very small heat of solution of the calcium hydrate which would normally go into solution. If such a large excess of water was used that all the calcium hydrate formed could dissolve, the heat of formation of the hydrate in dilute solution, 219,500 Calories, would be used, and the final result would be 33,000 Calories. The case is also very instructive, because it contains two compounds which are endothermic, that is, their heat of formation is negative, and therefore, as in the case of CaC^2, heat is given out when it decomposes, while in the case of C^2H^2 heat is absorbed when it is formed.

(d) What are the heats of combustion of the gaseous hydrocarbons which form constituents of common fuels, expressed per cubic meter of gas burning, and with the water formed assumed remaining as vapor, uncondensed? To calculate these we write the equations of combustion, with water as gas, and find the sum total of heat evolved; since every molecule of gas burning represents 22.22 m³ of gas, we can, by a simple division, obtain the value sought.

Methane:

$$\begin{array}{l|l} CH^4 + 2O^2 = & CO^2 + 2H^2O \\ -22{,}250 & +97{,}200 + 2(58{,}060) \\ \hline \multicolumn{2}{l}{+191{,}070 \text{ Calories.} \qquad = 8{,}598 \text{ per m}^3 \text{ CH}^4.} \end{array}$$

Ethane:

$$\begin{array}{l|l} 2C^2H^6 + 7O^2 = & 4CO^2 + 6H^2O \\ -2(26{,}650) & +4(97{,}200) + 6(58{,}060) \\ \hline \multicolumn{2}{l}{+683{,}860 \text{ Calories.} \qquad = 15{,}387 \text{ per m}^3 \text{ C}^2\text{H}^6.} \end{array}$$

Propane:

$$\begin{array}{l|l} C^3H^8 + 5O^2 = & 3CO^2 + 4H^2O \\ -33{,}850 & +3(97{,}200) + 4(58{,}060) \\ \hline \multicolumn{2}{l}{+489{,}990 \text{ Calories.} \qquad = 22{,}050 \text{ per m}^3 \text{ C}^3\text{H}^8.} \end{array}$$

By applying these principles to all the hydrocarbons whose heat of formation has been given in the tables, we obtain the following useful table, water being considered as uncondensed and cold:

Hydrocarbon.	*Molecular Heat of Combustion.* Calories.	*Heat of Combustion of* 1 m³ (Calories)	1 ft³ (B.T.U.)
Methane (gas)	191,070	8,598	966
Ethane (gas	341,930	15,387	1728
Propane (gas	489,900	22,050	2477
Ethylene (gas)	321,770	14,480	1627
Propylene (gas)	471,830	21,232	2385
Toluene (liquid)	906,990	.....	....
Benzene liquid	758,130	.....	....
Benzene gas	765,330	34,440	3869
Turpentine liquid	1,428,930	.....	....
Turpentine gas	1,438,330	64,725	7271
Naphthaline solid	1,223,690	.....	....
Naphthaline liquid	1,228,290	55,273	6209
Anthracine (solid)	1,690,150	.....	....
Acetylene (gas)	365,270	16,437	1846
Methyl alcohol liquid	148,270	.....	....
Methyl alcohol gas	156,670	7,050	799
Ethyl alcohol liquid	295,330	.....	....
Ethyl alcohol gas	305,430	13,744	1544
Acetone liquid	396,130	.....	....
Acetone gas	403,630	18,163	2040

(To the above we will add data for three other common gaseous fuels).

Carbonous oxide (gas)	68,040	3,062	344
Hydrogen (gas)	58,060	2,613	293.5
Hydrogen sulphide	122,520	5,513	619

Calorific Power of Fuels.

By using the principles explained we can calculate the calorific power of any combustible, with the aid of one or two simple assumptions. Regarding the water formed, we will consider in all ordinary metallurgical calculations that it remains uncondensed, thus putting it on the same footing as the other products of combustion. This amounts virtually to assuming that the latent heat of condensation of the vapor formed has not been generated in the furnace, an assumption which is quite justified if, as is almost always the case, the water formed inevitably escapes as vapor. If, in any special case, the products of combustion are in reality cooled down so far that the water does condense, then it would be proper to assume the higher calorific powers for the combustion of the hydrogen-containing materials, and thus a more accurate heat-balance of the furnace operation could be constructed. An instance of the latter might be the case of the partial combustion of a fuel in a gas producer, where the fuel gas is afterwards cooled before using, in order to condense from it ammonia water, etc. (Mond's producer). In this case, it is possible that some water formed by the partial combustion would be afterwards condensed, and to obtain a correct heat balance-sheet of the producer it would be necessary to assume that this heat had also been generated. Such cases occur very rarely in metallurgical practice, and it is therefore recommended to always use the lower calorific values, assuming water vapor uncondensed and its latent heat of condensation not to have been generated—unless the conditions of the problem point plainly to the opposite course as correct. It is no more correct to charge against a furnace the latent heat of condensation of the water vapor formed, than it would be to charge against it the latent heat of condensation of the carbonic oxide formed, if the conditions of practice are such that it is impossible to utilize in any useful manner either one of them. Those who persist in assuming that the latent heat of vaporization of the water formed by combustion is generated in and chargeable against the furnace, and then, of necessity, charge the same quantity against the heat carried off in the products of combustion, are in the great majority of cases adding to both sides of the balance sheet a quantity which cannot possibly be utilized, and are therefore distorting all the factors of heat generation and distribution. By thus abnormally increasing the sensible heat in the waste gases, they abnormally *decrease* the proportionate value of other items of heat distribution and utilization. I have dwelt on this matter at length, because it is of importance in furnaces using fuel, such as gas, rich in hydrogen; where, in some cases, the counting of the latent heat of condensation of the water vapor formed, in the escaping gases, would perhaps *double* the apparent chimney loss, and give distorted values to the whole heat balance sheet. What we are after, in every case, are the real facts and figures as to the working of a furnace or operation, and we must therefore judge the operation by a theoretically perfect standard, it is true, but not by a theoretically impossible standard; *i. e.,* our standard must be the theoretically possible one under the practical conditions necessarily prevailing. A radical reform is very much needed in just this respect,—in the treatment of the latent heat of vaporization of the water formed, by writers on metallurgical processes and problems of combustion.

Problem 4.

A natural gas from Kokomo, Ind., contained by analysis: CH^4 94.16, H^2 1.42, C^2H^4 0.30, CO^2 0.27, CO 0.55, O^2 0.32, N^2 2.80, H^2S 0.18 per cent.

Required:

(1) What is its metallurgical calorific power per cubic meter and per cubic foot?

(2) Comparing it with coal having a calorific power of 8000 Calories, how much gas gives the same generation of heat as a metric ton of coal?

(3) If the natural gas costs $0.15 per 1000 cubic feet, at what price per metric ton would the coal furnish heating power at the same cost for fuel.

Solution:—Requirement (1) Heat of combustion of 1 cubic meter:

CH^4 [illegible] × 8,108 = [illegible] Calories

H^2 [illegible] × [illegible] = 37.1

C^2H^4 [illegible] × 14,480 = 43.4

CO [illegible] × 3,062 = 16.8

H^2S [illegible] × 5,513 = 9.9

Total = 8203.1 "

In British thermal units per cubic foot we have

$$8{,}203.1 \times 3.967 \div 35.314 =$$
$$8{,}203.1 \times 0.11233 = 921.5 \quad \text{B. T. U. per ft}^3 \ (1)$$

Requirement (2):

$$\frac{8000 \times 1000}{8{,}203.1} = 975 \text{ m}^3$$
$$= 34{,}450 \text{ ft}^3$$

If the ton of coal be taken as 2240 pounds, instead of the metric ton of 2204 pounds, the equivalent volume is

$$34{,}450 \times \frac{2240}{2204} = 35{,}013 \text{ ft}^3.$$

Requirement (3):

$$\frac{34{,}450}{1000} \times 0.15 = \$5.16\tfrac{3}{4} \text{ per metric ton}$$

or per ton of 2240 pounds,

$$\$5.16\tfrac{3}{4} \times \frac{2204}{2240} = \$5.08\tfrac{1}{2} \text{ per long ton.}$$

DULONG AND PETIT'S LAW.

When a solid or liquid carbonaceous fuel is to be burned, its calorific power may either be determined directly in a calorimeter (which is the best way, if carefully done), or may be calculated from its analysis. In case it is determined calorimetrically, the weight of water produced per unit of fuel should be determined either in the same or by a separate experiment, and the latent heat of vaporization of this water subtracted from the calorimetric value of the fuel, in order to get its metallurgical, or practical, calorific power.

Example.—A coal gave 9215 calories per gram in the calorimeter, and its products of combustion gave up 0.45 grams of condensed water. What is its practical calorific power?

Taking the products cold, the heat of condensation per gram of water vapor may be taken as 606.5 calories (Regnault), and we therefore have obtained in the calorimeter

$$0.45 \times 606.5 = 273 \text{ Calories}$$

more than if the products were cold and water uncondensed. The practical calorific power is therefore

$$9215 - 273 = 8942 \text{ Calories per kilogram.}$$

Dulong and Petit's law, or method of calculation, is the statement that the calorific power of a fuel can be calculated from the amount of carbon, hydrogen and oxygen it contains by assuming all the carbon free to burn, the oxygen to be all combined with hydrogen in the proportions of O^2 to $2H^2$ (32 to 4), and that the rest of the hydrogen is free to burn.

Example.—Taking the bituminous coal of Problem 1, containing carbon 73.60, hydrogen 5.30, nitrogen 1.70, sulphur 0.75, oxygen 10.00, moisture 0.60, ash 8.05, its calculated calorific power is per kilogram:

Carbon 0.7360 × 8100 = 5962 Calories.

Hydrogen 0.0530

[illegible]

0.0405 × 34,500 = 1397 " (water liquid)

Sulphur [illegible] × [illegible] = 17

Total [illegible] "

This calculation is made, however, on the assumption that all the water in the products is condensed to liquid, *i. e.*, not only the water formed by combustion of the free hydrogen, but also the already-formed H^2O containing the oxygen of the coal, as well as the moisture present to start with. To obtain the practical calorific power by calculation we must subtract from the above-generated heat the latent heat of *all* the vapor of water in the products, as follows:

Moisture in coal	0.0060 kilos.
Moisture formed by combined hydrogen.	0.1125 "
Moisture formed by free hydrogen......	0.3645 "
Total	0.4830 "

Latent heat of evaporation = 606.5 × 0.4830 = 293 Calories.

Practical calorific power, water all as vapor = 7083 Calories.

The values thus calculated are found to agree satisfactorily with the laboratory and the practical calorific powers of the fuels.

THE THEORETICAL TEMPERATURE OF COMBUSTION.

If we start with a cold fuel and cold air the maximum temperature which can be obtained by the combustion is simply that temperature to which the heat generated can raise the products of combustion, assuming that all that heat resides primarily in the products as sensible heat. This is then a problem in physics, in which we have a known amount of heat available, and the question is to what temperature it will raise the products of combustion. The problem is at once soluble when we know the mean specific heats of the products of combustion and their respective quantities.

Until a few years ago these specific heats at high temperatures were unknown, and very false results were obtained by assuming constant specific heats from ordinary temperatures up, and values were thus calculated which were known to be hundreds, and in some cases thousands, of degrees too high. The most satisfactory values for the specific heats of the fixed gases and water vapor and carbonic oxide are those determined by Mallard and Le Chatelier, and their proper use has entirely solved the question of theoretical temperatures of combustion, and removed a positively disgraceful discrepancy between theory and practice. The specific heats of these gases increase with temperature, so that the actual specific heat of a cubic meter (measured at standard conditions) is, at the temperature t,—

for air N^2, O^2, H^2, CO — S = 0.303 + 0.000054t

" " CO^2 — S = 0.37 + 0.00054t

" " (vapor) H^2O — S = 0.34 + 0.00030t

For calculating the quantity of heat needed to raise the gas from 0° to t°, however, we need the mean specific heats, Sm, between 0° and t°. These are, of course, the above constants plus half the increase, viz.:

for N^2, O^2, H^2, CO — Sm = 0.303 + 0.000027t

" CO^2 — Sm = 0.37 + 0.00027t

" (vapor) H^2O — Sm = 0.34 + 0.00015t

and the quantity of heat needed to raise 1 cubic meter (at standard conditions) from 0° to t° is

for N^2, O^2, H^2, CO — Q (o-t) = $0.303t + 0.000027t^2$

CO^2 — Q (o-t) = $0.37t + 0.00027t^2$

H^2O — Q (o-t) = $0.34t + 0.00015t^2$

or, finally, the quantity of heat needed to raise 1 cubic meter (measured at standard conditions) from t to t' is

for N^2, O^2, H^2, CO — Q (t'-t) =
$$0.303\ (t'-t) + 0.000027\ (t'^2-t^2)$$

CO^2 — Q (t'-t) =
$$0.37\ (t'-t) + 0.00027\ (t'^2-t^2)$$

H^2O — Q (t'-t) =
$$0.34\ (t'-t) + 0.00015\ (t'^2-t^2)$$

And the mean specific heat, Sm, between any two temperatures is

for N^2, O^2, H^2, CO — Sm (t'-t) =
$$0.303 + 0.000027\ (t' + t)$$

CO^2 — Sm (t'-t) =
$$0.37 + 0.00027\ (t' + t)$$

H^2O — Sm (t'-t) =
$$0.34 + 0.00015\ (t' + t)$$

Examples:

(1) What is the temperature of the hottest part of the oxy-hydrogen blowpipe flame?

According to the equation $2H^2 + O^2 = 2H^2O$, the hydrogen gas forms an equal volume of water vapor. (Equal numbers of molecules.) The heat of combustion of one cubic meter of hydrogen is 2613 Calories. The question is, therefore: "To what temperature will 2613 Calories raise one cubic meter of water vapor?" The answer is, using the data above,

$$Q\ (0\text{-}t) = 0.34t + 0.00015t^2 = 2613$$

Whence $t = 3191°$

(2) What is the maximum temperature of the hydrogen flame burning in dry air?

The heat evolved is, as before, 2613 Calories, and there is formed also one cubic meter of water vapor, but the products will contain also the nitrogen which accompanied the 0.5 cubic meter of oxygen necessary for combustion, viz.: $\frac{0.5}{0.208} - 0.5 = 1.9$ cubic meters, and this is heated as well as the water vapor. Therefore,

$$Q\ (0\text{-}t) = (0.34t + 0.00015t^2) + 1.9(0.303t + 0.000027t^2) = 2613$$

Whence $0.916t + 0.0002013t^2 = 2613$

And $t = 2010°$

(3) What is the temperature of the air-hydrogen blowpipe, if 25 per cent excess of air is used above that required?

All the data are the same as above, except that to the products will be added $0.25\ (0.5 + 1.9) = 0.6$ cubic meters of unused air, which has the same specific heat as nitrogen, and therefore the equation becomes

$$Q\ (0\text{-}t) = (0.34t + 0.00015t^2) + 2.5\ (0.303t + 0.000027t^2) = 2613$$

Whence $t = 1764°$

This calculation brings out very clearly the uselessness and ineffectiveness of using more air than is theoretically necessary; any excess simply passes unused into the products of combustion, and thus reduces their maximum possible temperature. The obtaining of the maximum possible temperature depends upon accurately proportioning the supply of oxygen or air to the quantity of gas burned; an excess or a deficiency will result in a lower temperature.

COMBUSTION WITH HEATED FUEL OR AIR.

If the fuel itself or the air which burns it is preheated, the sensible heat in either one or in both is simply added to the heat generated by the combustion, to give the total amount of heat which must be present as sensible heat in the products of combustion. The effect is exactly the same as if the heat developed by combustion had been increased by the sensible heat in the fuel or air used.

What is the calorific intensity (theoretical maximum temperature) obtained by burning carbonous oxide gas? (a) Cold, with cold air; (b) cold, with hot air at 700° C.; (c) hot, with hot air, both at 700° C.

(a) Take one cubic meter of carbonous oxide. Calorific power 3,062 Calories; product one cubic meter of carbonic oxide, and 1.904 cubic meters of nitrogen.

Let t be the temperature attained; then

Heat in the 1 m^3 carbonic oxide $= 0.37t + 0.00027t^2$
" " 1.904 m^3 nitrogen gas $= 0.577t + 0.0000514t^2$

" " products $= 3{,}062$ Cal's. $= 0.947t + 0.00032114t^2$

Whence $t = 1947°$

(b) If the 2.404 cubic meters of air needed are preheated to 700° C., they will bring in as sensible heat

$$Q\ (0\text{-}700) = 2.404\ [0.303\ (700) + 0.000027\ (700)^2]$$
$$= 552 \text{ Calories.}$$

The total heat in the products will be $3062 + 552 = 3614$ Calories, and therefore $0.947t + 0.0003214t^2 = 3614$

Whence $t = 2189°$

(c) If the gas itself is also preheated it brings in

$$Q\ (0\text{-}700) = 0.303\ (700) + 0.000027\ (700)^2$$
$$= 225 \text{ Calories.}$$

The total heat in the products will be $3614 + 225 = 3839$ Calories, and therefore $0.947t + 0.0003214t^2 = 3839$

Whence $t = 2284°$

Heating both gas and air to 700° before they burn thus raises the theoretical temperature from 1947 to 2284, or 337°.

Problem 5.

Statement.—The natural gas of Kokomo, Ind., contains by analysis: Methane (marsh gas) 94.16, ethylene (olefiant gas) 0.30, hydrogen 1.42, carbonous oxide 0.55, carbonic oxide 0.27, oxygen 0.32, nitrogen 2.80, hydrogen sulphide 0.18,—in percentages, by volume.

Required:

(1) The maximum flame temperature, if burned cold with the theoretical amount of cold, dry air necessary.

(2) The calorific intensity, if burned cold, with the requisite air preheated to 1000° C.

(3) The calorific intensity, if burned cold, with 25 per cent. more air than theoretically necessary, preheated to 1000°.

Solution:

[The practical calorific power of this gas has been already calculated in Problem 4 as 8203 Calories per cubic meter. The gas itself is always burned cold, because, if preheated, it decomposes and deposits carbon in the regenerators.]

The products of combustion of the gas, say per cubic meter, must first be found, using the formulæ for combustion.

Requirement (1):

	1 Cubic Meter of Gas.	Oxygen Needed. m^3	Products. CO^2	H^2O	SO^2	N^2
CH^4	0.9416	1.8832	0.9416	1.8832	.....	....
C^2H^4	0.0030	0.0090	0.0060	0.0060	.....	.....
H^2	0.0142	0.0071	.....	0.0142	.....	.....
CO	0.0055	0.00275	0.0055	.....	.....	.....
CO^2	0.0027	.....	0.0027	.....	.....	.....
O^2	0.0032	0.0032	.....	.....	.....	.....
N^2	0.0280	.....	.....	.....	.....	0.0280
H^2S	0.0018	0.0027	.....	0.0018	0.0018	.....
		1.90155	0.9558	1.9052	0.0018	0.0280
Air required		9.14		(carrying in N^2) =		7.238
					Total N^2	7.266

The heat generated will exist as sensible heat in the CO^2, H^2O, SO^2 and N^2 of the products. The mean specific heat of SO^2 per cubic meter is 0.444 at ordinary temperatures; what it is at high temperature has not been determined; we will give it the same index of increase as the analogous gas CO^2, and take for it $Sm\ (0\text{-}t) = 0.444 + 0.00027t$, or $Q\ (0\text{-}t) = 0.444t + 0.00027t^2$.

At the temperature attained by combustion, t, the products will contain the following amounts of heat:

$$N^2 = 7.266\ (0.303t + 0.000027t^2)$$
$$H^2O = 1.9052\ (0.34t + 0.00015t^2)$$
$$CO^2 = 0.9558\ (0.37t + 0.00027t^2)$$
$$SO^2 = 0.0018\ (0.444t + 0.00027t^2)$$

$$\text{Total} = 3.2044t + 0.00074057t^2 = 8203 \text{ Calories.}$$

From which $t = 1806°$ (1)

Requirement (2):

Heat in 1 m^3 of air at $1000° = 0.303\ (1000) + 0.000027\ (1000)^2$
$= 330$ Calories.
" 9.14 m^3 " $1000° = 3016$ "

Therefore: $3.2044t + 0.00074057t^2 = 8203 + 3016$

Whence $t = 2288°$

Requirement (3).

Excess air [illegible] $= 11.14 \times 0.21 = 2.285$ m

Heat in [illegible] at 1000° = 11425 + 330 = 3770 Calories

The heat capacity of the excess air must be added to the heat in the products at temperature t, viz.:

$$\text{Excess air} = 2.285\ (0.303t + 0.000027t^2)$$

making the total heat in the products (adding to previous expression.)

$$8.068t + 0.00080271t^2 = 8203 + 3770$$

Whence $$t = 2134^\circ$$

[The next instalment of these "Metallurgical Calculations" will discuss the heat necessary to produce reactions at high temperatures.]

Electric Smelting and Blast Furnace Gases.

By Auguste J. Rossi.

(Concluded from page 150.)

What has been said of blast furnace gases is equally true of the waste gases of the coke ovens which recuperate ammonia and tar products. They have been utilized in gas engines for generating power to operate the machinery or electricity for lighting purposes, and, in certain collieries in Germany, some 2500 hp. have thus been created last year.

The first experiments with these gases were made as early as 1895 in Scotland. The blast furnaces of this district using soft coal and being already provided with installations to collect the by-products, tar, ammonia, etc., the attention was naturally called to the value of these gases. Since then, and simultaneously in England, Belgium and Germany, papers have been published on this subject, and we give in a note at the end of this article references to the documents to be consulted. At Hoerde, in Germany, a 600-hp. machine was in operation as early as 1898. But in 1898, Mr. A. Greiner, general director of the Cockerill Iron Works, Seraing, Belgium, made an elaborate study of the question, both as to the heating capacity of these gases and their applications to gas engine study, which has appeared in a paper[3] read before the Iron and Steel Institute at their annual meeting, and, as the data he gives are based on numerous and authentical tests made under his direction, and continued for a period of some years, and that, furthermore, they are corroborated by other tests made in Europe since, and agree very well, as we will see, with figures furnished by Mr. Uehling; we will extract from this paper interesting data bearing on this question and compare them with those given by Mr. Thomson.

We will find that the discrepancies are not what they appear to be at first sight. Indeed, the amount of gases from a blast furnace varies necessarily with the amount of coke consumed per ton of iron, so that, for a consumption of 1700 pounds per ton of iron, instead of 2240 pounds, the two amounts may differ considerably, and if a comparison is to be made, it must be on the assumption of the same duty of coke and conditions of ore in both cases. The amount of gases used to heat the blast varies also necessarily according to the temperature at which the air is blown in the furnace, and, for instance, blast heated to 593° C. (Pittsburg district, 1890) will require *less* gases for the same volume than blast heated to 704° C. (Clarence Works, England). So, again, two figures which may appear wide apart may well be [illegible] by the temperature of blast. As a consequence, that part of the gases which is supplied to the boilers, from the total amount generated in the blast furnace, may well be found to vary in these different conditions. Again, as to horse-power required per ton for blowing purposes, it will be so much the greater as the pressure of this blast, which may vary from 7 pounds to 10 pounds and much more, will be greater, and as this horse power has to be subtracted, with the auxiliary power for hoists, etc., from the amount which the gases could supply in a gas engine, the *available surplus* may well differ, unless similar conditions are compared.

These blast furnace gases consist, in a general manner, of about 58 per cent to 65 per cent of nitrogen, with a balance of CO and CO_2, with a small quantity of hydrogen. The ratio of CO_2 to CO varies according to the furnace. Such a ratio as 0.70, as is obtained at the Clarence Works in England, is considered as a good one, though a higher ratio nearer to 1 has been realized. The higher this ratio the most economically as to fuel is the furnace working for a given ore; the higher it is the greater also is the specific gravity of the gases. Were it that of air, 1 cubic meter would weigh: 1.294 kilogram. As it is, we may assume in the following calculations 1.30 kilogram per cubic meter of gases as approximately sufficient.

Experiments were made at Seraing on the heating capacity of these gases. Specimens were collected *every day* during fifteen consecutive days, from 6 a. m. to 6 a. m., in vessels of 10 liters capacity, and tests made by Mr. A. M. Witz gave about 997 calories per cubic meter as the average result. This gives, at 1.30 kilogram per cubic meter: 1314 thermal units per pound (assuming 950 calories); the figure adopted by Mr. Greiner was 1296 thermal units per pound; Mr. Uehling takes the figure, 1,283.9 thermal units per pound, which is very close. Mr. Thomson, by assuming, as builders of engines do, 10,000 thermal units per horse-power-hour at 25 per cent efficiency, and amount of gas required would come to the same figure, very nearly.

As to the amount of gas per ton of iron smelted, Mr. Greiner gives: 13,365 pounds per ton of iron; Lowthian Bell gives for the Pittsburg district, 11,200 pounds (blast 593° C., coke 1882 pounds); for the Clarence Works, 13,347 pounds per ton of iron (blast 704° C., coke 2240 pounds[4]); Mr. Uehling gives, 10,589.62 pounds, which is very close to the figure of the Pittsburg district, and a more conservative one.[4]

Hence, per ton of iron, we have, adopting Mr. Uehling's figures, 10,589.62 pounds × 1283.09 thermal units = 13,587,330 thermal units per ton of iron: as this corresponds to the lowest figure for weight of gases per ton of iron we will adopt it. If we take 13,347 pounds gas per ton (of the Clarence Works), and 1283.09 thermal units per pound, we come to 17,122,201 thermal units, and this corresponds to a consumption of coke of 2240 pounds per ton of iron (ton for ton) and a temperature of blast of 704° C. If we take 11,200 pounds gas per ton of iron, and again 1283.09 thermal units per pound, we come to: 14,369,600 thermal units per ton of iron, and this corresponds to a consumption of 1881 pounds of coke per ton of iron.

Mr. Thomson says: "The average practice in coke consumption is probably very close to 2240 pounds per ton, yielding 14,000,000 B. T. U. in waste gases." This is more than the figures adopted by Mr. Uehling, and nearly that corresponding to a consumption of 1881 pounds of coke only per ton of iron. It is true Mr. Thomson adds that a figure as low as 1700 pounds of coke per ton of iron has been obtained (with dry air by Mr. Gailey) in which case he assumes that the amount of gases would be considerably smaller, and consequently the total number of thermal units, which does not necessarily follow. But this smallest figure for coke is so far exceptional, and will be so, at least for some time to come, it being due to modifications in the working of blast furnaces, which necessitate particular additional machinery, a practice not generalized yet. Until it has become so the average practice in coke consumption may be retained at one ton of coke per ton of iron, and even 1881 pounds (0.82 ton), and still the heat contained in the gases will be found to approximate 14,000,000 B. T. U., or the figure of Mr. Uehling, 13,587,330 B. T. U.

We will remark here in passing, that if we adopt 11,200 pounds of gases per ton of iron, and the 14,000,000 B. T. U. of Mr. Thomson, corresponding to 2240 pounds of coke, we come to a heat capacity of 1270 B. T. U. per pound of gas, that is practically a figure already mentioned.

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[3] [illegible] A. Greiner, Iron and Steel Institute meeting, London, [illegible]

[4] Special volume Iron and Steel Institute in American in 1890, page 171.

Hence, in short, and so far as the present good ordinary practice of blast furnace goes, we can count on an amount of at least 13,587,330 B. T. U. in the gases per ton of iron, and on a heat capacity of 1283 B. T. U. per pound (Uehling, corroborated by other tests). As 1 hp.-hour = 2545 B. T. U., and that a gas engine is now guaranteed to give 25 per cent efficiency, this corresponds to, say, 10,180 B. T. U., 10,000 B. T. U. in round numbers, per effective hp.-hour, so that with such gases we would require 10,180 ÷ 1283 = 7.92 pounds gas per hp.-hour, or, say, 8 pounds in round numbers. At Seraing direct experiments made with a 200 hp.-motor showed a consumption of 4 kgs. (8.80 pounds) of gas per effective hp., with the prospect that with a large motor (said Mr. Greiner) this figure would be much lowered; this makes it close to the 8 pounds mentioned above.

In the volume of the Iron and Steel Institute (special volume 1890) calculations of the horse-power required per ton of iron per hour for blowing purposes, show for the Isabella, Edgaro, Thomson, Lucy and other furnaces, such figures as 150 hp. to 160 hp and 187 hp., and as low as 130 hp. if we assume 10 per cent (Mr. Thomson's figure) for "auxiliaries," pumps, hoists, etc., we come to 175 hp. to 200 hp., and if we call it 200 hp., as Mr. Uehling does, this will cover the cases of higher pressures (higher than 10 pounds) now resorted to in certain furnaces. If, again, we add 50 hp., as Mr. Uehling proposes, for all auxiliary power, "as the tendency is to replace by machinery the hand labor," we come to an ample allowance of 250 hp. per ton of iron per hour; with an efficiency for gas engines of 8 pounds of gases (7.92 pounds) per effective hp.-hour; these 250 hp. represent 250 × 8 = 2000 pounds of gas for this purpose. The total amount of gases per ton being 10,589.62 pounds (Uehling), this corresponds, in round numbers, to practically 19 per cent of the volume of the gases for blowing in and compression. Mr. Thomson (gas engines) takes the figure of 20 to 25 per cent, counting the "auxiliaries." This is quite close, and probably 22 to 23 per cent ought to cover all possible cases.

Mr. Uehling gives for *heating* the blast, 1,884.38 pounds of gases. This is, again, nearly 19 per cent of the total weight. Mr. Thomson assumes 18 to 33 per cent; obviously this depends on the temperature at which the blast is blown in the furnace. At the Clarence Works (England) the temperature was 704° C., while at the Pittsburg works it was only 593° C., and, in some furnaces it is even lower, so that one figure may be just as correct as the other. If we take the average we come to 25 per cent of the total gases to heat the blast.

Twenty-three per cent to compress the blast, and 25 per cent to heat it, give us a total of 48 per cent, or, say, in round numbers, 50 per cent, and we could count thus on 50 per cent of the total amount of gases as likely to be available for other purposes than the requirements of the blast furnace. Mr. Thomson[5] gives 52 to 42 per cent as available surplus, an average of 47 per cent, as the figures are given; and we can see that, to all purposes, starting from different points of view, we come very nearly to the same results.

What will make the calculations based on the preceding data differ widely, is that Mr. Thomson, assuming 52 per cent, a figure based on Table I, page 96, as available surplus, applies it to the amount of gases corresponding to an expected normal future average consumption of coke per ton of iron of 1700 pounds (0.773 ton), while he states that the average normal consumption is now 2240 pounds, ton for ton, or thereabout. We do not contest the possibilities of the future, but we believe that we are authorized to say that, for the present at least, and for some time to come, the figures based on these anticipated results, legitimate as they may be, are ultra-conservative now. But, far from taking exception from them, as we have said, taking them such as they are, they show, even in the best possible conditions of blast furnace practice in the future, what could be expected of this utilization of about half the heat of the blast furnace gases in gas engines.

We will take such of the figures of Mr. Uehling, which, as it has been seen above, are corroborated and agree with those of Mr. Thomson for an assumption of 1 ton of coke per ton of iron, and not too high a temperature of blast, viz.:

10,589.03 pounds of gases per ton of iron.

1,283.09 B. T. U. per pound of gas giving 13,587,330 B. T. U. per ton.

1 hp.-hour = 2545 B. T. U., and an efficiency of 25 per cent for gas engines.

That is to 7.93 pounds of gas per effective hp.-hour. Mr. Uehling adopts 1,884.38 pounds of gas to heat the blast per ton, corresponding to about 19 per cent of the whole amount of gases. This leaves 8,705.25 pounds of gas available, corresponding to 1,097.76 hp. per ton of iron per hour. Subtracting 250 hp. required for blowing in and compressing the blast and auxiliaries, corresponding to about 19 per cent of gas, Mr. Uehling arrives at "847.76 hp. for sale or available for other purposes for every ton of iron produced per hour."

If we assume, as explained above, as an average of Mr. Thomson's figures, 25 per cent of the total amount of gases for heating the blast, and some 23 per cent, as an average, for the blast engine and auxiliary power, this gives us some 48 per cent of gas as necessary for the blast furnace plant with gas engines, leaving a surplus of 52 per cent. 0.52 × 13,587,330 B. T. U. = 7,065,411 B. T. U., and since we can assume with gas engines 10,000 per guaranteed hp.-hour, this gives us 706.54 hp. available as surplus per ton of iron per hour.

Mr. Thomson, for a consumption (as an average practice now) of 2240 pounds of coke per ton of iron, assumes as total amount of heat contained in the gases per ton, 14,000,000 B. T. U., and adds in the article quoted that improvements in gas stoves and gas engines permit the use of the larger figure given in his Table I, page 96, that is, as given, 52 per cent as surplus of gas available:

0.52 × 14,000,000 B. T. U. = 7,280,000 B. T. U., and

7,280,000 B. T. U. ÷ 10,000 = 728 hp. as available per ton of iron per hour. But if, anticipating the future based on some results recently obtained in particular conditions in one case, he assume 1700 pounds of coke per ton of iron as a normal figure, then the heat contained in the gases, "*he says*," is reduced to 9,000,000 B. T. U. per ton, which gives him, adopting as a surplus his figure of 52 per cent "*the larger figure*" as given in his table:

9,000,000 × 0.52 = 4,680,000 B. T. U., corresponding to 468 hp. per ton of iron made per hour, as available surplus.

Mr. Thomson adds, however (page 97) that "with poorer ores and in other unfavorable conditions the coke per ton of iron may reach as high as 2800 pounds, in which cases the heat contained in the gases per ton will reach 18,000,000 B. T. U." This is almost the figure at the Clarence Works, as quoted by Lowthian Bell in the volume of the transactions of the Iron and Steel Institute, based on the weight of gases given.

But here we may be allowed to call the attention to an oversight of Mr. Thomson, which changes the results he gives to much better figures, whatever he may assume for the total heat of the gases per ton, let it be 14,000,000 B. T. U. or 9,000,000 B. T. U. Indeed, referring to his Table I, page 96, of the article quoted, we read:

	Gas Engines.	
	%	%
Heating the blast	18 to	33
Compressing the blast	16.5 to	20
Auxiliaries	3.5 to	5
Surplus available	52.0 to	42
	100	100

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[5] Table I, page 96 (gas engines) Electrochemical and Metallurgical Industry, Vol. III, No. 3.

[illegible] "heating" and "compressing the [illegible] as quoted by him, represent only 38 per cent [illegible] that the real surplus available from his calculations is not 52 per cent to 42 per cent, but 62 per cent to 42 per cent (an average of 52 per cent) and should read so. Considerations which he explains further (page 97) "may permit," he says, "the use of 'the larger figure' given in Table I as the percentage of surplus gas available in gas engines." We have thus to adopt this corrected figure, 62 per cent, instead of [illegible] per cent, which gives for 14,000,000 B. T. U. heat of the [illegible] 868,000 B. T. U., corresponding to 868 hp., instead of 728 hp., for 2240 pounds coke per ton of iron, and for 9,000,000 B. T. U., based on a hoped-for normal reduced consumption of 1700 pounds.

9,000,000 B. T. U. × 0.62 = 548,000 B. T. U., corresponding to an available surplus of 558 hp. per ton-hour, instead of 468 hp.

The production of pig iron in the United States for this years is estimated to 20,000,000 tons, or thereabout, which gives 2280 tons per hour.

At a surplus of 847.76 hp. per ton-hour (Uehling) we come to a total per hour for 2280 tons of 1,932,892 hp. At our figure of 700 hp. surplus, to 1,596,000 hp. At Thomson's figure for 14,000,000 B. T. U. in the gas, viz: 728 hp. to 1,659,840 hp. per hour, and for 868 hp.-hour per ton (the above figure of 728 hp. corrected for 62 per cent surplus of gases, instead of 52 per cent) we come to 1,959,040 hp. At Mr. Thomson's figure of 468 hp. per ton-hour (corresponding to 9,000,000 B. T. U., consumption of coke 1700 pounds) we come to 1,067,040 hp. per hour. At the same figure, 468 hp. corrected to 558 (for surplus of 62 per cent of gases, instead of 52 per cent), we come to 1,272,240 hp.-hour.

Hence, we can legitimately count, taking Mr. Thomson's figures as they stand, in the best possible future conditions of blast furnace practice *and of ores*, on at least one million of horse power per hour, and, with his own figures corrected on 1,272,240 hp., as available power to generate electricity, and for some time to come on at least 50 per cent more, if not double this amount in many furnaces.

In fact, such calculations should be made *for "each particular"* furnace, according to the coke consumption per ton of iron, temperature of blast and amount of gases resulting per ton, factors which necessarily vary with the quality of the ores smelted. As a minimum possible, we can depend on at least 1,000,000 hp. per hour of surplus available power.

The importance of this utilization of blast furnace gases for generating power available for electrometallurgical purposes is such that it may not be considered foreign to our subject to examine briefly the objections which have been raised against it, and the answers, based on practice, which have been made by eminent metallurgists.

As to the heat capacity of these gases, and their adaptability to be used in gas engines, they are not any longer contested:

First, one of the drawbacks brought forward has been the dust they carry with them.

The gases at Seraing are certainly not cleaner than at any other works; the contrary is true, owing to the kind of ores smelted. Calculations made by Mr. Greiner, based on repeated experiments, show that the gases supplied to the gas engine (200 hp. motor), freed from such dust, as they are in all cases when supplied to the boilers at these works, did not contain more than 2 grams of impalpable powder per cubic meter. At this figure of 4 cubic meters of gases per hp., a 200-hp motor would absorb 40 kgs. of dust per day. This dust, says Mr. Greiner, was given out almost completely as a light white cloud, as proved by the fact that, after [illegible] found necessary to [illegible] the cylinder of the motor, and Mr. Greiner adds, where are the boilers, heated by blast furnace gases, which can run four [illegible]?

Second. As to the destructive action on metal of these gases, owing to the sulphuric anhydride, chlorine, phosphoric acid, the dust may contain, analyses show such an amount of alkalies, lime, magnesia, alkalies soda and potash that this corrosive action was certainly counteracted. At any rate, by a simple washing not requiring necessarily a large amount of water, they were able at Seraing to run a motor (200 hp.) for two years, without any observable corrosion of the cylinder.

Third. Objections to the irregularities in the composition of these gases have not been found to be serious, as it might have appeared a priori; first, because gas engines accommodate themselves to gases of different composition, and experience shows that such cases occur when the furnace is working badly, and a furnace must be working exceedingly badly when the gases, in these conditions, cannot be lighted. At any rate, if the furnace is working so badly as to produce non-inflammable gases, the same trouble will certainly prevail with boilers heated by these gases, as with the gas engine, and in such cases in the same manner as coal or coke is burnt on grates under the boilers during such temporary derangement, as Mr. Greiner remarks, an emergency gas producer could feed the gas engine. M. Thomson has very wisely remarked that this objection, if it exists, could be easily remedied by grouping together two, three or more furnaces without even the use of a gas producer.

Fourth. As to the speed of the gas engines operating the blowing cylinder, says Mr. Greiner, if it were an object when the gas motors were small, and their speed high, the large motors of 2000, 3000 hp. and more are running now at such speed as cannot be objectionable with our modern blast engine. In Mr. Gailey's run, the blowing engine was running at 100 to 114 revolutions per minute.

The advantages of gas as a source of power are obvious. Beside the economy of running of the machine as compared to that of engines supplied with steam, gas can be transmitted at any distance by piping, almost without condensation in the flues, as is the case with steam, without any appreciable loss of head, without any pressure, and such piping is comparatively simple and cheap. The use of gases supresses the boilers and their attendant drawbacks, and suppresses cumbersome masonry. It may be added that a gas motor can, at will, supply heat, light or power.

Mr. Thomson has examined the details of the cost of an installation of such a gas motor plant, utilizing the blast furnace gases, and has given estimates which we have not to consider in this article, his competency on such matters being well known. We will retain only his very conservative estimate of surplus, assumed, as we have said, in the most favorable conditions possible of work of the blast furnace, and realized only so far, but exceptionally, the figure 468 hp. per ton-hour in order to show the possible gain were this power to be applied even to the direct electric smelting of iron from its ores.

We have seen that 200 hp., as the power required for the electric smelting of 1 ton of pig iron per day, is not a too favorable figure; this corresponds to 4800 hp. per ton-hour. Hence the surplus of power over all the requirements of the blast furnace represents the possibility of smelting electrically an additional amount of pig iron per hour of about 10 per cent of the present production. Were we to take Mr. Uehling's figures, we would come to an increased production in metal of 20 per cent. These figures speak for themselves. Applied to the manufacture of electrical steel by the open-hearth process, to that of special steels, etc., this wasted surplus of energy, even utilized but in part, opens possibilities of which we desired in this article to show the importance.

We give below a table resuming the calculations which we have examined above.

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Hubert, Ingenieuer au corps des Mines de Belgique. De l'utilization de gaz des haut fourneaux pour la production de

	Heat Value of Gases, Thermal Units, per lb.	Weight in lbs. of Gases per Ton of Iron	Total Amount of Thermal Units in Gases, 1 lb. 1,283 B. T. U.	Percentage of Total Amount of Gases Used.				Net Horse-power Surplus With Gas Engines Over All Requirements of the Blast Furnace.	
				For Compressing Blast and Auxiliaries	For Heating the Blast.	Totals, Per Cent	Surplus, Per Cent.	Per Ton Hour.	Per Hour for 2,280 Tons Estimated Production of Pig Iron in 1905.
	Th. U.	Lbs.	Th. U.	%	%	%	%	H.P.	H.P. per hr.
Luermann	1,220	....	....	......	......	......	....	......	....
Greiner (Seraing) Coke 2,240 lbs., per ton of iron..........	1,314	13,365	16,858,620	.....	......	......	....	800	1,824,000
Uehling	1,283	10,589	13,587,330	19°	19	38	62	847.96	1,932,392
Lowthian Bell (Clarence Works) Coke 2,240 lbs, Blast 704° C...	......	13,347	17,122,201	......	......	......	....	......	....
Lowthian Bell (Pittsburgh Works), Coke 1882 lbs, Blast 593°C.	......	11,200	14,309,600	.... .	......	......	....	......	... .
Thomson on a basis of 2,240 lbs. Coke, assuming fig. Table I, Surplus 52%........	......	......	14,000,000	20 to 25	18 to 33	38 to 58	52 to 42 as given	728	1,659,840
Thomson, same basis, 2,240 lbs. Coke, assuming corrected fig of Table I, viz., 62%.	......	......	14,000,000	20 to 25	18 to 33	38 to 58	62 to 42 as corrected	868	1,959,040
Thomson, Coke 1,700 lbs., assuming fig. of Table I, 52% Surplus	......	......	9 000,000	......	......	.....	52% as given	468	1,067,000
Thomson, Coke 1,700 lbs., assuming corrected fig. of Table I, 62%..............	.....	......	9,000,000	......	......	......	62% as corrected	558	1,272,240
Assuming 2,240 lbs. Coke, Average of 50% of Surplus.....	.. ...	......	14,000,000	......	......	......	50%	700	1,596,000
Calculated from Mr. Gailey's data in his dry air run. Blast 465° C., Gas 191°C Coke 1,726 lbs. per ton of iron, average.	......	11,284	14,477,372	......	......	......	50%	724	1,610,720

la force motrice.—*Annales des Mines de Belgique*, 1897, tome II.

Lüermann, Stahl und Eisen, March 15, 1898, page 158. Heat value of blast furnace gases. Paper read at the general meeting Assembly of German Metallurgists, Düsseldorf, February 27, 1898.

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Stahl unde Eisen, March 15, 1898.

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Lowthian Bell, etc., the *Iron and Steel Institute* in America. Special volume, 1890.

A. Pourcel, Génie Civil, XXXIII, No. 3, 1898. On the Utilization of Blast Furnace Gases in Gas Engines.

A. Dutreux, Utilisation directe des gaz des Haute fourneaux dans les Moteurs a explosion. *Génie Civil*, 1898.

Prof. Watkinson, West of Scotland, *Iron and Steel Institute*, March, 1895.

Engineering, May 13, 1898.

F. duP. Thomson, Electrochemical and Metallurgical Industry, March, 1905.

Calcium Carbide Duty Free in Chili.—By a recent law calcium carbide has been placed on the free list in Chili. The intention of this law is to so cheapen the cost of the production of acetylene gas as to make it a practicable illuminant for the numerous estates and small centers of population for which illuminating gas and electricity are not feasible.

RECENT METALLURGICAL PATENTS.

Roasting.

T. Huntington and F. Heberlein (786,814, April 11) divide the roasting of sulphide ores into two portions. First, a preliminary roast with access of air, until the proportion of sulphur is reduced to 12 per cent. Then the temperature of the ore is suddenly cooled by means of water; the temperature is thereby reduced to that of atmosphere, some sulphatization is effected, and the anhydrous sulphates of the metals combine with their water of crystallization, so that $ZnSO_4$ becomes $ZnSO_4$ 7Ag; $CuSO_4$ becomes $CuSO_4$ 5Ag, etc. Then follows the second roasting process, whereby the water-cooled material is placed in a receptacle in which it is reheated and through which a current of air is caused to pass. In order to start the necessary combustion, a layer of hot fuel or of hot ore containing sulphur is placed in the bottom of the receptacle. The current of air assists and augments the heat thus furnished, with the effect that the sulphur is oxidized and sulphurous-acid gas is given off until the desulphurization of the ore is complete. The material is then ready for treatment in the blast furnace.

H. M. Heath, G. Morrice and R. L. J. Gruss (785,067, March 14) patent a modification of the process of roasting auriferous ores containing sulphur or arsenic ("sulphurets"), preliminary to the chlorination process for the extraction of the gold. If the sulphurets are roasted with access of air, a temperature of 1,200° F. is required, and the treatment requires considerable time. The inventors propose to carry out the roasting process not in presence of air but of oxygen.

[illegible] [illegible] is maintained a temperature of 600° F. only is [illegible] and the time of treatment is considerably reduced. The inventor uses a roasting furnace with a vertical flue of [illegible] [illegible] form, through which the auriferous ore passes downwards, while oxygen is passed upwards from an oxygen-generator connected to the lower terminal of the flue

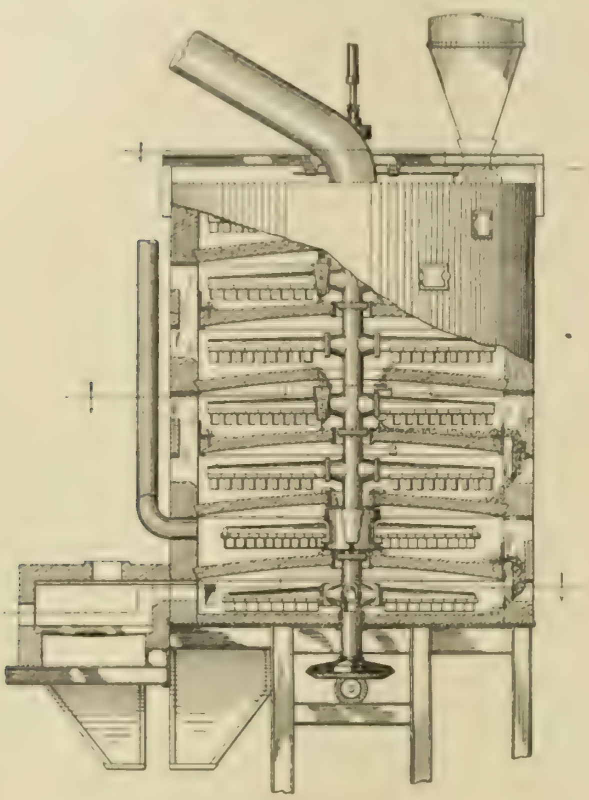


FIG. 1.—ROASTING FURNACE.

C. H. Repath and F. E. Marcy (patent 785,437, March 21, assigned to Frank Klepetko) patent a modification of the McDougall furnace for roasting zinc blende. The furnace is shown in Fig. 1. The object is twofold: to oxidize all the ore without loss, and to save the sulphurous anhydride formed during the operation. The loss of ore in ordinary furnaces is mainly due to the upward draft of gases which carries the fine particles of ore with it. To prevent this, the inventors cause the ore to slide from one hearth to the next below, instead of letting it drop through space and encounter the upward draft of the gases. The sliding of the ore assists the action of the rakes. The upper four hearths are provided with two rabble-arms each, the sixth and seventh with three rabble-arms, in order to increase the stirring and discharge rather on one side of the furnace. The construction of the furnace is described in detail.

WET PROCESSES.

An agitator for use in the cyanide process is patented by E. Stevens (784,598, March 14). Its construction is shown in Fig. 2, which is self-explanatory. The gold-carrying materials are placed within the bottom of the funnel at the lower end of the apparatus, and the cyanide solution is added. By revolving the shaft 13, which carries a screw, 14, the materials are raised inside the cylinder which surrounds the screw. Arriving at the top, 21, of the cylinder, the materials spread out and glide [illegible] the inclined upper surface, 20, of the cone around the [illegible]er. The materials then fall to the bottom, and are again [illegible]d upward inside the cylinder, etc. While being spread [illegible] the conical surface at the top, the pulp is exposed [illegible] the air. "Where potassic cyanide is employed, as [illegible] done, the cyanogen appears to be set free in its nascent state upon the upper surface of the spreader, which is, of course, always above the level of the solution. The cyanogen thus being set free, and having intense chemical activity, owing to its nascent state, unites with the gold, with the result that a much larger proportion of the metal is saved, or that with a given ore a weaker cyanide solution may be employed." It has, of course, long been known that the presence of air is essential for dissolving gold in cyanide solution.

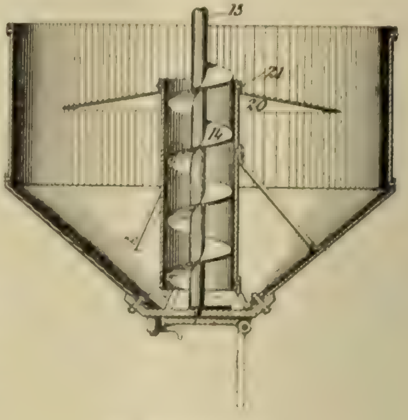


FIG. 2.—AGITATOR.

G. A. Duncan (787,878, April 18) patents a treatment of slimes by filtering. The essential point is that the slimes are accumulated on the exterior of a filtering cell by suction from the interior of the cell while the latter is immersed in the slime-bearing liquid. The supply of this liquid is then cut off and a metal-solvent liquor is introduced, and is sucked through the slimes, etc.

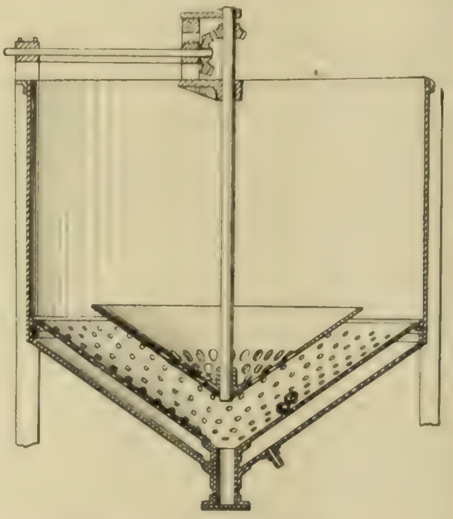


FIG. 3.—AGITATOR AND FILTER.

V. A. Robinson (785,522, March 21) patents the ore agitator and filter shown in Fig. 3, in which 8 is a foraminous supplemental bottom, which acts as a filtering medium; above the same is a perforated inverted-cone-shaped agitator. By revolving it a continuous systematic agitation is produced, whereby the filtering medium, 8, is kept free from clogging. The material in the inner inverted-cone-shaped agitator is continuously carried upwards, flows over and passes downwards between the outer surface of the agitator and the inner surface of the filtering surface. Through the perforations in the surface of the agitator it enters again into the latter and so on, until the process is completed.

B. Hunt (786,441, April 4) patents a gravity filter for extracting liquids from granular or comminuted material. This material is arranged in pyramidal form, of sufficient height to cause its compression by its own gravity. The inventor provides means for removing from the base of the column, by mechanical appliances, a quantity equal to that supplied at the top, so that the head and pressure will remain approximately uniform. He also provides a series of inwardly-inclined surfaces, or arresting devices, attached to the inside walls of the retaining chamber, whereby the filtering material is retained, and water exuding from the mass is directed away from the same and through the filtering medium.

On account of limitations of space a large number of patents had to be held back, which will be covered in our next issue.

Notes on Electrochemistry and Electrometallurgy in Great Britain.

(From Our Special Correspondent.)

Metallurgical Papers of the Month—Metallurgy of Gold.

The February meeting of the Institution of Mining and Metallurgy by no means exhausted the subject of the paper by Messrs. A. Jarman and E. Le G. Brereton of the use of ammonia compounds in cyaniding cupriferous ores. At the March meeting a supplementary paper on this subject entitled "Note on the Ammonia-Copper-Cyanide Process" was presented by Mr. H. Livingstone Sulman. In no sense did Mr. Sulman treat the previous paper in a hostile spirit, but tendered his suggestions with the object of rendering it even more complete.

In investigating the solubility of gold in various cyanide and other solutions, in 1893, the author discovered a number of solvents of greater or less power; amongst the most rapid and efficient of these were solutions which contained an alkaline cyanide mixed with certain equivalents of ammonia, and of a cupric salt.

Such a solution when freshly prepared formed a gold solvent of much greater rapidity than ordinary potassium cyanide, indeed approaching that of bromo-cyanide; but its application was accompanied with a comparatively rapid loss of solvent power, and with the deposition of sparingly soluble crystals, containing ammonium, copper and cyanogen. Mr. Sulman endeavored to trace the reactions involved, but, not then recognizing their applicability to the treatment of oxidized cupriferous gold ores, he abandoned the ammonia-copper-cyanide type of solvent for simple gold ores by reason of its instability. The present note cannot, therefore, in any way detract from the priority of the several metallurgists who have since made the ammonia-copper-cyanide process for the extraction of gold a more or less successful one.

After describing in detail the various reaction, and discussing the effect of the instability of cupric cyanide and the varying compositions of the hydrated cupric-cuprois-cyanide, the author surmises his view of the process in the following terms.

"Owing to the complexity of the reactions, it is difficult, if not impossible, to formulate any simple chain of reactions as is held to explain the solution of gold by simple alkali-cyanide in presence of oxygen; it is still further complicated by the probable formation of other metallic cupro-cupric-cyanides, such as those of iron, etc. But it seems evident that the broad explanation of the ammonia-copper-cyanide process is dependent not on the solvent action upon gold of potassium cyanide, nor of ammonium cyanide, whether combined with copper cyanide or not, plus oxygen, but upon the formation of ammonium-cuproso-cupric cyanides of the cuprammonium type, which, gradually yielding up some portions of their cyanogen contents, act as powerful solvents for gold until their limits of stability and solubility are reached, without the necessary intervention of oxygen. When they reach this limit they cease to act as solvents ,and tend to separate out in the crystalline form noticed by the authors.

"These compounds, either in solution or as crystals, are broken up on heating with lime or other fixed alkali yielding ammonia and a solution of alkaline or of alkaline-earth cyanide, whilst cuproso-cupric hydrates, contaminated with more or less cuprous cyanide, are left as precipitate. The form in which the gold is dissolved in the above reactions, therefore, also differs from that of the ordinary cyanide process, as there is no uncombined alkaline cyanide present necessary to produce an alkaline aurocyanide. It is probable that the gold is converted by nascent cyanogen into simple aurous cyanide, which is readily soluble in any slight excess of ammonia and ammoniacal salts.

Messrs. Jarman and Brereton's conclusion is "that an ammoniacal cupriferous cyanide is produced of low solubility, having the property in common with copper potassium cyanide, that the copper in it is replaceable by gold—(i. e., in part.)" I think, however, it will be clear that whilst an ammonium cupriferous cyanide is responsible for the solvent action, it is not of the copper-potassium cyanide type referred to by the authors, but a cuprammanium cyanide containing no cyanogen combined with alkali metal. Further, it appears that the gold is not dissolved by replacing copper, but that this takes place by virtue of nascent cyanogen derived from the cupric cyanide of the cuprammonium compound. That copper is deposited whilst the dissolution of the gold proceeds is undoubted, but this is due not to replacement of copper by gold, but the formation of increasingly insoluble cuprammonium cyanides from the break-down of the cupric cyanide radicle with loss of cyanogen. Though this cyanogen is the cause of gold dissolution, it might equally well be used up in other reactions, with the continued deposition of the green ammonium dicuproso-cupric cyanide crystals. Messrs, Jarman & Brereton have hitherto dealt only with the question of oxidized copper in gold ores; the question of copper in pyritic or unoxidized gold ores is a problem of even greater commercial importance, although, apparently, correspondingly complex. The formation of cuprous sulphocyanide, which, becoming more insoluble as the alkaline cyanide contents of the liquors fall, is thus precipitated in the sands, is, I believe, to a considerable extent, the cause of cyanide losses in the treatment of ores containing copper sulphides."

While Mr. Sulman's paper called for such extensive quotations as has been accorded to it, the other paper by Mr. Gustav Köller on the Kedabeg Copper Mines, although much longer, does not lend itself to this treatment. It is not easy to abridge concisely some 38 8-vo. pages which are devoted to an historical account of the undertaking, and to the geological and metallurgical summary of the mines. As a contribution to the bibliography of copper mining, the paper as a whole is of great value and much interest. It may be mentioned, however, only about one-sixth of the copper is refined electrolytically.

The Metallurgical Side of the Liege Exhibition.

A modern exhibition without some meeting of scientific societies will only bear comparison to the eating of an egg without salt, or to another standard for things that are inane, of which one of Mr. Kipling's characters speaks startlingly. An international congress of mining, metallurgy, mechanics and applied geology is to be held at Liege from the 26th of June to the 1st of July. An influential organization has been appointed. In addition to the inaugural meeting, the congress will comprise: (1) General meetings; (2) section meetings, devoted to the special discussion of questions relating to mining, metallurgy, mechanics, and applied geology; (3) lectures; (4) visits to the exhibition, to various scientific or industrial establishments, and applied geology excursions. The provisional programme of the second or metallurgical section comprises papers on the following subjects: (1) Utilization of Non-Caking Coals for Making Coke; (2) Study of the Blast Furnace; (3) Influence of Foreign Substances on Pig Irons and steels; (4) Methods for Intercepting Dust from Blast Furnace Gases, with a View to Their Utilization; (5) Slag Bricks and Cements; (6) Utilization of Poor Gases, with a View to Producing Power for Driving Roll-Trains; (7) New Methods for Making Open-Hearth Steel; (8) Special Steels; (9) Forging by the Press and Steam Hammer—Hardening and Annealing; (10) Electro-Metallurgy; (11) Metallography.

The Institution of Mechanical Engineers is also holding a meeting at Liege shortly before the congress mentioned. The list of papers to be read has not yet been announced, but it is probable that several will be of a metallurgical nature.

Market Quotations.

In the chemical trade prices have been generally stable, particularly in the case of the alkaline products. Shellac seems to have ceased falling and to have taken an upward turn, the quotations now standing 147s. per cwt., as against 140s. a month ago. Copper sulphate has fallen 15s. to £22 per ton, while cyanides (98 per cent. minimum) are ½d. per pound cheaper at 7½d.

Para rubber is still in good demand, with firmer prices. Platinum is unchanged. Cleveland pig iron which fetched over 50s. at the beginning of the month has closed at 49s. 2d. Copper has also fallen since the middle of the month (when it was just under £69 per ton) to £67.17.6. Tin has advanced in a very striking manner to £137.10.0, chiefly due to a report that the Banca production would not come up to anticipations. A rise of £6.5.0 is indeed unusual. Lead remains at practically last month's closing price, although a rise is expected. Quicksilver is unchanged.

ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS.

By George P. Scholl, Ph. D.

Electric Furnaces and Furnace Products.

Electric Furnace.—W. C. Arsem, Schenectady, Assignor to General Electric Co. Patent 785,535, March 21, 1905. Application filed Aug. 12, 1904.

The furnace is intended for heating materials to a high temperature in a vacuum or an atmosphere of any gas. It consists essentially of a vacuum chamber of gun metal, coated with tin to prevent leakage through the pores of the metal. A cylinder of refractory but conducting material, such as graphite, is suspended from the cover of the gun-metal vessel by U-shaped tubular conductors. This graphite cylinder is brought into the form of a tubular helix by cutting a helical slot into it. It is provided at both ends with suitable graphite terminals, and a carbon standard is placed in its interior, which is intended to carry the crucibles or other articles to be heated. A box of refractory material is placed around the heating tube, the space between them being filled with pulverized graphite. The conductors which lead to the tube are made hollow for the purpose of circulating water through them, so as to keep them cool. Means for exhausting the air from the chamber or introducing any desired gas are, of course, provided. Furthermore, the whole vacuum chamber is surrounded by a water jacket. It is stated that with this apparatus any temperature up to the vaporization point of carbon can be obtained and constantly maintained for any length of time at a pressure of 0.2 mm. or less. The apparatus may be used for the fusion of refractory substances, the reduction of metals from their compounds in an atmosphere free from oxygen, the preparation of alloys in vacuo or inert gas, etc.

Electrolytic Production of Metals and Compounds.

Apparatus for Effecting Electrolysis.—H. Philipp, Wyandotte, Mich. Patent 784,592, March 14, 1905. Application filed Aug. 10, 1903.

The apparatus is intended for the electrolysis of alkali chlorides, and belongs to the type of the mercury cells. As shown in central vertical section in Fig. 1, it depends upon the flow of the mercury by gravity. It consists of a cylindrical vessel *A*, preferably made of earthenware, which is provided with a spirally-wound trough *B*, an inlet pipe c^1 and outlet pipe c^2. The mercury enters through pipe c^4 and flows out through pipe c^6. The cylindrical vessel is provided with a cover *c*, which rests in a water seal *d*. Introduced through an aperture in its top is the cylindrical carbon anode *D*, composed of carbon staves held together by rubber bands b^2, electrical connection being made with the terminal of the current by means of an iron ring *b* and rivets b^3. The cylindrical carbon [illegible] filled with an alkali chloride, holes *c* being arranged [illegible] for a free exchange of the solution in both compart- [illegible] wires in conjunction with an electromagnet [illegible]ged in the mercury inlet pipe so that the electromagnet will lift the valve *P*, and will only allow a certain quantity of metal to flow in a constant stream through the spiral trough so as to cover the bottom of the latter to the necessary depth. Conducting wires *a*, placed at various points along the path of the mercury, are connected to the negative terminal of a source of current, and thus constitute the flowing sheet of mercury, the cathode of the apparatus. The gutter is so inclined horizontally that the centrifugal action of the mercury is counteracted, and an exact and even depth of the moving cathode is assured. The gases generated during electrolysis pass off at the outlet *c*, while the amalgam escapes through a pipe c^6, furnished with a tripping valve *S*, which will withstand the action of the solution, but will lift when the specifically heavier mercury strikes it. One of the advantages claimed for the apparatus is that the flow of mercury is automatically interrupted as soon as the current is cut off, and the apparatus then empties itself of mercury, and the deleterious reactions upon the mercury, which are liable to happen in horizontal apparatus when the current is cut off, are thus prevented.

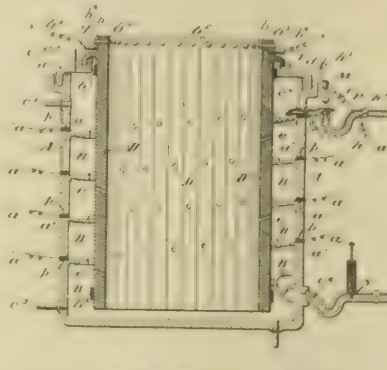


FIG. 1.—ELECTROLYSIS OF SODIUM CHLORIDE.

Method of Treating Alkali Metal Amalgam.—C. E. Baker and A. W. Burwell, Cleveland. Patent 782,893, February 21, 1905. Application filed April 15, 1904.

The method depends essentially upon heating the amalgam in an atmosphere containing hydrogen for the purpose of forming a hydride of the alkali metal. Such amalgam, derived, for instance, from an electrolytic cell with a mercury cathode, is drawn into a suitable retort and heated in a hydrogen atmosphere until the mercury distills off, while the hydride of the alkali metal is substantially non-volatile at the temperature of vaporization of the mercury, and is left behind as a residue. The vapors of mercury may then be condensed and returned to the electrolytic cell. It is stated that the amalgam may be heated to a variable temperature, provided it keeps between the vaporization point of the mercury and the decomposition point of the sodium hydride. The inventors have found that the mercury distills off more freely by alternately raising and lowering the temperature of the amalgam between these limits. The mercury may be distilled in a current of hydrogen, if desired. The residual alkali-metal hydride is then heated to a temperature sufficient to decompose it with liberation of hydrogen, the resulting alkali metal being recovered. It is proposed to employ a number of interconnected retorts, so as to operate the process without substantial loss of hydrogen, by absorbing in one retort the hydrogen produced by distilling the sodium hydride in another one.

Electrolytic Apparatus.—A. Wright, Brighton, England. Patent 782,308, February 14, 1905. Application filed March 23, 1903.

The invention applies specially to electrolytic apparatus, such as mercury electrolytic meters in which mercury is used as anode, and is intended to prevent the formation of deposits on the surface of the mercury or at the edges, where the mercury and the containing vessel meet. This formation is explained as being due to the stagnation of the mercury and the impossibility of realizing a washing action of the electrolyte in the ordinary type of apparatus. In order to get over this difficulty, it is proposed in the present apparatus to support the mercury constituting the anode, not by a solid support, but by a so-called "surface tension grid;" that is, a support having orifices and formed of material which is not wetted by mercury, so that the well known surface tension of the mercury is utilized. Such a grid may be made of platinum or platinum-iridium gauze or perforated platinum or platinum-iridium sheet or foil, or if spirally or otherwise bent platinum or platinum-iridium wire. They may also be made of spun-glass fabric. Platinum gauze with a hundred meshes to the linear inch is stated to give good results in practice, in the case of electrolytic meters. In other cases the general consideration of the gauze to be employed is dependent upon the fact that the greater the depth of mercury on the grid, the shallower must be the orifices, and the greater the vibration to which the mercury is exposed, the smaller must be the orifices. When a grid is made of spun-glass gauze, its strength as compared with the weight of mercury to be supported must be carefully considered. The action of such grids is described as follows: When acting as a support for a globule or other mass of mercury, the grid is not wetted and the surface tension of the mercury is such as to prevent its flowing through the interstices of the grid. The effect is that instead of the upper surface of the mercury alone being washed by the currents set up in the electrolyte, the under surfaces and sides are also washed by reason of the electrolyte having access to the mercury at the orifices or interstices of the grid. Several constructions of mercury electrolytic meters embodying the above principles are described.

Process of Detinning.—F. von Kuegelgen and G. O. Seward, Holcomb's Rock, Va., assignors to Union Carbide Co. Patent 783,726, February 28, 1905. Application filed August 1, 1904.

The process depends upon the use of a solution of dry chlorine in a liquid which contains no moisture, based on the fact that dry chlorine, in marked contrast to wet chlorine, between certain limits of temperature acts on tin, but not on iron. The inventors have discovered that a solution of the dry chlorine in stannic chloride or other suitable anhydrous liquid removes the tin completely and rapidly from tin scrap dropped into it, while the iron is not attacked at all. Even if the iron were attacked, in the case of a stannic chloride electrolyte, it would form chloride of iron, which is insoluble in anhydrous stannic chloride. This advantage would also hold good if other metal were present in the tin scrap, as the anhydrous stannic chloride does not dissolve other chlorides. The stannic chloride containing the dry chlorine gas, by action upon tin scrap, reforms a new quantity of stannic chloride. The process is carried out by placing tin scrap in a closed vessel, so as to prevent moisture reaching the materials, and there subjecting it to a solution of dry chlorine in anhydrous stannic chloride. An excess of the chlorine should always be present. After the chlorination has taken place, a suitable quantity of dry chlorine is added and the quantity of new stannic chloride formed taken away. The removed stannic chloride may then be treated for the recovery of the tin by electrolysis or by chemical means. Though stannic chloride is preferable, any other liquid may be used which is capable of dissolving dry chlorine, and which contains no moisture.

Electroplating Device.—G. W. Clough, Cleveland. Patent 784,034, March 7, 1905. Application filed January 30, 1904.

The object aimed at in the invention is the construction of an apparatus for continuous electroplating of small articles. The apparatus consists of a rectangular vat, in which is situated, nearly half submerged, a rotating drum. Within this drum there is arranged a spiral carrier, which rotates with the drum and the turns of which form a continuous channel in the manner of an Archimedean screw, so that the articles to be plated, and which are introduced at one end, are gradually carried forward until they emerge at the other end. A spiral wire conductor is wound within the spiral turns of the carrier, so as to ensure continuous electrical contacts of the articles being plated with the cathode terminal of a source of current. Anodes, suspended from the shaft around which the drum rotates, supply plating metal to the fluid in the usual manner. In operation, the speed of the apparatus is regulated according to the thickness of deposit the articles are to receive, so that they are finished when they emerge at the discharge end of the cylinder. The vessel in which the electrolyte is contained is made of wood or other insulating material, and the drum is also preferably composed of wood or rubber.

Electroplating Apparatus.—L. Potthoff, Flushing, N. Y. Patent 786,776, April 4, 1905. Application filed Dec. 1, 1904.

The apparatus is intended for continuously galvanizing articles such as bars, pipes, etc., in large quantities. Upon the electroplating tank there is mounted a framework, which carries hangers at opposite ends of the tank, through which pass two shafts, having a sprocket wheel mounted on each of them. Chains or belts run around these sprocket wheels, upon which are mounted, at suitable intervals, wooden pins, of such shape as to cause round work, such as tubes, to roll when engaged by them. The work is rolled by the pins over longitudinal conducting bars, which form a sort of a track extending through the tank and which act as cathodes. The anodes are placed below the cathode bars. As the work is rolled by the pins over the conductor bars, the points of contact constantly change, so that all parts of the work receive a deposit.

Galvanic Elements.

Galvanic Battery Cell.—A. T. Sanden, New York. Patent 784,125, March 7, 1905. Application filed December 15, 1904.

The battery belongs to the type of small cell used for being carried by persons by means of a belt or the like. It is compact, and the zinc element is made of a much thicker metal than used heretofore, and is corrugated, preferably longitudinally. A porous fabric divides it from the outer element, which consists of a plate or sheet of copper so folded as to bring its edges together and give it an ovoid form.

Miscellaneous.

Apparatus for Hygienic and Therapeutical Purposes.—J. M. A. Lacomme, New York. Patent 782,400, February 14, 1905. Application filed September 3, 1903.

The apparatus is intended for the electrolysis of water or chemical reparations, so as to use the resultant gases for medical purposes in hospitals, schools, etc., for changing the atmosphere. It consists essentially of a glass cup, the lower part of which is filled with the liquid intended to be electrolyzed, while the upper part is perforated. Electrodes, the whole of which, with the exception of a small part at the lower end, is covered with a glass tube, dip into the liquid. One or more of these cups may be provided, each being a duplicate of the other. The electrodes are adjustable.

Composition of Matter and Process of Making It.—B. F. Gardner, Chicago.

The composition is intended to be used as a resistance material for use in heaters, rheostats, etc. It is made up of about 50 per cent graphite, 25 per cent carborundum, 20 per cent rubber and [illegible] per cent sulphur, these proportions, however, being variable, according to the amount of electrical resistance the material is required to have. The carborundum, graphite and sulphur are mixed in a dry state and are then thrown onto the rubber, which has previously been reduced to a plastic condition by being passed through heated rolls. The rolling is continued until the mixture is thoroughly united with the rubber. The mixture is then calendered or molded into any suitable form and vulcanized, after which it is carbonized. This is done by burying it in sand and applying heat until all the sulphur is driven off and the rubber is carbonized. The carbonized material is very hard, dense and non-fusible, and retains the shape in which it has been molded.

Asymmetric Shunt for Direct Current Electric Circuits of High Inductance.—W. S. Horry, Niagara Falls. Patent 782,134, February 7, 1905. Application filed October 29, 1904.

The arrangement consists of a main circuit, which contains an inductance, for example, an electric motor, which circuit can be opened or closed by a switch. A shunt circuit connects two points, one at each side of the inductance, with each other, which shunt circuit contains an asymmetric electrolytic cell, in order to allow the passage of the current carried by the main circuit, but to permit the passage of current in the opposite direction. The asymmetric cell consists of an electrode of aluminium, and one of lead in an electrolyte composed of an aqueous solution of potassium phosphate or citric or sulphuric acid. A modified arrangement is provided with a switch having an arc-shaped contact, by which, when it is in its normal position, the main circuit is closed, while the shunt circuit is open. When the switch is opened it first closes the shunt circuit and then opens the main circuit. This arrangement is desirable when the inductance must remain in circuit for considerable periods, as the asymmetric cell is normally out of circuit, and there occurs no loss of current or injury to the cell, which might be occasioned by slow or accidental leakage through it.

On acount of limitations of space a number of patents had to be held over for our next issue.

SYNOPSIS OF PERIODICAL LITERATURE.

A Summary of Articles Appearing in American and Foreign Periodicals.

INDUSTRIAL APPLICATIONS

Precipitation of Gold from Cyanide Solutions.—A paper read by W. J. Sharwood last December, before the California Miners' Association, reviews the various processes by which precious metals may be precipitated from a cyanide solution, and deals especially with precipitation by electrolysis, by zinc shavings and by zinc dust. The zinc dust process is thought to be as well adapted as the electrolytic to the precipitation of low-grade slime solution. Some experiments were made on a solution carrying about $2.50 gold and $0.20 silver per ton, together with 0.015 per cent copper and 0.05 per cent free cyanide. One portion was treated with hydrogen sulphide until saturated; another was made distinctly acid and then allowed to stand until the recipitate of silver and copper cyanide settled, when it was filtered. Other portions were electrolyzed. It was found that precipitation was far from complete unless the current was allowed to act for a long time and unless the surface of the lead plates used was large in proportion to the volume of solution. When this was done all traces of the gold and silver and copper were removed. Other portions were agitated with zinc dust at the rate of one-quarter pound per ton, without filter-pressing and others filter-pressed. The results were as follows:

Method of Precipitation.	*Percentage of Contents Precipitated.* *Gold.*	*Silver.*	*Copper*
Saturation with hydrogen sulphide......	2	97	3
Acidulation	65	90	75
Electrolysis for 24 hours...............	100	100	100
Agitation for 15 minutes with zinc dust..	25	60	3
Agitation with zinc dust and filter-pressing	94	97	4

Six Months' Experience with Similar Solutions, Using Zinc Dust and Filter Pressing.

	Gold.	Silver.	Copper
KCN. averaging 0.04 per cent...	98 10 99.	Over 96	2 to 4
KCN. from .08 to 0.1 per cent...	98 to 99.	Over 96	0.5 to 1

These figures bring out the fact that silver is, as a rule, easier to precipitate than gold, just as it is slower to dissolve in cyanide solutions. The author concludes that it impossible to draw any general comparison between the merits of the different zinc and electrolytic methods of precipitation, but gives the following figures as based on practical results, with material of moderately low value, all being calculated to a standard, or 100 tons solution precipitated per day: Electrolysis; precipitating tank of 1,200 cubic feet capacity, say a box 10 by 24 feet, having about 3000 square feet each of anode and cathode surface. This would require five or six horse-power for electrolysis, in addition to the power for pumping. The actual precipitation obtained at two plants averages very close to 90 per cent of the value in solution. Zinc shavings, 100 cubic feet of capacity for the zinc, or about 175 gross cubic feet of zinc boxes, say, two boxes 2 by 2 feet by 9 inches by 16 feet, using 25 to 30 pounds of zinc per day, the only power required being for pumping the solution and turning the zinc. The actual precipitation obtained may be taken at 90 per cent under favorable circumstances. Zinc dust; one filter press with 16 frames,-about 2 feet square, working not over 15 or 16 hours per day, would require about 2 horse-power during that time (neglecting any additional friction in pumps) in excess of the power required for pumping solution from pumps in the last case. Some power is also needed for stirring or agitating solution when adding zinc, but this is only for a few minutes at a time. The percentage of precious metals precipitated at two plants ranges from 95 to over 98 per cent, from 20 to 30 pounds zinc dust being used per 100 tons. Owing to the greater cost of the machinery required for the electrolytic and zinc-dust processes, it is evident that the zinc-shavings system is by far the best adapted to all small or temporary plants; but with installations intended to treat 100 tons or more per day, or likely to be in use for several years, the merits of the other systems should be considered.

De-tinning of Scrap.—Dr. H. Mennicke continues in the March number of the *Elektrochemische Zeitschrift* his study of the advances in this art since 1902. This last instalment treats of the non-electrical methods. Bergmann, in a Swedish patent (No. 14,156 of June 6, 1901), treats the scrap in an iron vessel with caustic soda solution, using suspended cupric oxide as a depolarizer. The supric oxide forms voltaic couples with the tin, and the latter goes into solution as sodium stannate. The solution is filtered, concentrated and precipitated by a stream of carbon dioxide. The precipitate is filtered out, dissolved in acid, and the tin precipitated electrolytically. Dr. Mennicke maintains that it would be cheaper to treat the stannate solution direct electrolytically; it appears to the reviewer that since precipitation by carbon dioxide is a cheap process, that it might be equally cheap to precipitate, filter, and then reduce the tin oxide by carbon in a shaft furnace. In any event, either of these suggested variations would be cheaper than the original proposition of Bergmann.

Th. Twynam, of Leeds, England, has devised a method which appears to Dr. Mennicke as being very simple, cheap and direct. The scrap is dipped into a thick mush of concentrated salt solution and powdered coke, taken out at once, and then let stand some time in the air. The tin is oxidized to tin oxide and falls off the scrap, or is washed off with the unchanged salt, caustic and coke, leaving the iron as clean black iron. The wash water is filtered, thus recovering part of the tin as oxide and basic chloride, with coke powder; which mixture, after drying, is reduced in a reverberatory furnace. Some tin goes into solution as stannate, and accumulates until in sufficient amount to be electrolytically precipitated out. Twynam speaks of the oxidation in the air being complete in an hour, but investigations of Mennicke have shown that a longer time is necessary; also that iron rust on the scrap will be mixed with the tin oxide obtained, thus showing that the process will require large working space for the heaps exposed to the weather, and that only clean, bright tin scrap can be advantageously treated. The treatment of the subject by Dr. Mennicke is to be continued in subsequent issues.

METALLURGY.

Dry Air in Blast Furnaces.—O. Boudouard rushes into the breach, in the February number of the *Revue de Metallurgie* with a new theory to explain the importance of drying the blast. He has made experiments with carbonic oxide on iron oxides, and finds that if moisture is present with the reducing gas that reduction is slightly less active at temperatures below 1000° C., and thence concludes that the removal of this moisture accounts for the greater economy by increasing the reducing power of the gas. The weak point of these views is that the differences found were not large or very clearly marked, and that, even if they are true experimentally, there is no doubt that they do not apply to the case under discussion, because the moisture of the blast is decomposed to hydrogen (and carbonous oxide) at the tuyeres, and it is the action of a mixture of hydrogen with carbonic oxide on the reduction of the ores which M. Boudouard should have studied, instead of the influence of moisture.

Of much greater value, and, indeed, worthy of attention, is an address on the same subject made by the well-known German chemical engineer Dr. C. von Linde, before the *Verein Deutscher Eisenhüttenleute* at the recent yearly meeting in Düsseldorf, on December 4, as reported in *Stahl und Eisen* for January 1. Dr. Linde starts out very happily by remarking that "the 'Eismann' has come to the help of the 'Eisen hüttenmann.'" His paper discussed the cost of cooling the blast. To cool one cubic meter of air 27° C. absorbs about 8 calories, and to condense 9 grams of moisture 6 calories, making a total refrigerating requirement of 14 calories. Since a horse-power-hour can exert a refrigerating effect of 3600 calories, one horse-power should refrigerate 250 cubic meters of air, 27° C. It is interesting to compare this with the power needed to compress the blast; this requires on an average one horse-power-hour for every 25 cubic meters compressed, showing that the refrigerating requires about one-tenth of the power needed by the compressing engine. A plant to cool 1000 cubic meters of air per minute requiring about 240 horse-power, would cost, approximately, $50,000. Extensive experience with air refrigerating plants has shown that such cooling plants easily work twenty years without great deterioration, so that a yearly charge of 5 per cent for deterioration, or $2,500 is quite sufficient; interest on average capital at 5 per cent would be $1,250; labor can be put at $2,000; and general expenses, including repairs, $2,000, making the total yearly cost of the operation $7,750 ($21.11 daily, or less than $0.10 per ton of iron made). An important point must, however, not be overlooked. Six out of the 14 calories required for cooling the air is absorbed as latent heat in the ice formed, and is therefore in such a form as to be partially recoverable; that is, part of it can be utilized for cooling down the incoming air. This utilization may not be perfect, but if only 20 per cent of it is thus utilized, the heat to be absorbed per cubic meter of air cooled is reduced 1.2 calories out of 14, or by 9 per cent, making the cost of the plant necessary only some $45,000, in place of $50,000, and reducing expenses in about the same proportion. The figures above given are not theoretical data, or mere estimates, but rest on many years' practical experience with refrigerating plants.

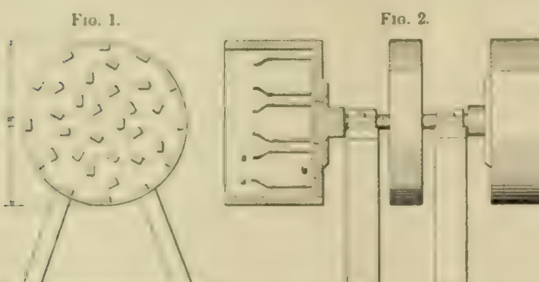
In the discussion of Dr. Linde's paper, Dr. Weiskopf, of Hanover, referred to the results obtained by Mr. Gayley (increased output of furnace, 24 per cent, decreased consumption of coke 20 per cent), and after remarking that these results were totally unexplainable by the mere saving of the heat necessary to decompose the ordinary moisture of the blast, advanced the suggestion (somewhat similar to Boudouard's) that the absence of hydrogen in the furnace gases must in some mysterious way be the cause of the improvements noted, but gives no satisfactory reasons as to the why or wherefore of this influence. If the equations of reduction of iron oxide by carbonous oxide and hydrogen are, in fact, carefully studied, it will be found that the presence of extra carbonous oxide and hydrogen coming from the decomposition of moisture should *facilitate* reduction, and therefore their removal or absence should make the reduction poorer. The argument that the heat required to decompose the moisture of the blast is regained in the upper part of the furnace by the reoxidation of the hydrogen, and that therefore the removal of the moisture does not assist the furnace by reason of the avoidance of heat absorption in decomposing it, is a statement hardly needing refutation. Everyone who knows how a blast furnace works knows that, like a human being, it works best with warm feet and a cool head—hot below and cool above.

Herr Direktor Haedicke, of Siegen, mixed up the matter still worse. He maintained that the question was one of the physical condition of the moisture in the air; that such cool moisture was really liquid water in suspension; that it was not decomposable by glowing carbon like real high temperature steam, and that the only action of the blast furnace on such cold moisture was to truly vaporize it. Since the vaporization of the ordinary amount of moisture in the air would require but a very small amount of heat, the conclusion was evident that, if Mr. Gayley's statements were reliable, the removal of the moisture from the blast was not the real cause of the improvement. We are sorry that *Stahl und Eisen* was complaisant enough to print such gibberish.

Engineer Dr. Lürman, of Berlin, made the most serious attempt to cut this siderological "Gordian knot." In his contribution to the discussion he made the following points: (1) The volumes of air blown into the furnace without dried blast and with dried blast do not correspond with the dimensions and rate of running of the engine given by Mr. Gayley, and are inconsistent with the amount of coke burned per day in the furnace in each case. As to the first statement, it is true that there is only a rough correspondence between the piston displacement and the volumes given by Mr. Gayley, but Dr. Lürman has forgotten that the air blown into the furnace is not equal to the piston displacement, and that Mr. Gayley estimated an average net deliverey of 90 per cent. This was really the only thing Mr. Gayley could do, but he failed to state in so many words that he had done so; when this fact is recognized, the apparent inconsistency disappears. On the next count, Dr. Lürman is again wrong, for he forgets that the same proportion of carbon need not always be burnt at the tuyeres; that varying proportions of it may be consumed above the tuyeres in direct reduction, and that the air received by the furnace in the two cases is not necessarily proportional to the amount of coke put into the furnace in each case. In really a much smaller proportion of the coke is burned at the tuyeres in the case of dried blast than with moist blast, and Dr. Lürman's criticism loses its point because of failure to notice that fact. (2) The decomposition of the moisture blown in could only absorb 3.64 per cent of the heating power of the coke burned, and the actual result obtained by Mr. Gayley being a saving of 19.5 per cent, the removal of the moisture cannot account for the larger part of the saving. The blast was, however, warmer in the case of dried blast, but this can only account for a saving of 8.5 per cent of the calorific power of the coke. The two together are therefore unable to account for the saving, and Dr. Lürman's final effort at a solution is the opinion that the furnace was receiving too much coke to start with; that it was, so to speak, overfed with coke, and, like an overfed human being, it improved in working capacity and economized at the same time by being put on a more restricted diet. *Next!*

Chlorination and Bromination of Gold.—An interesting paper, presented before the American Institute of Mining Engineers by H. O. Hofman and M. G. Magnuson, is published in the March issue of the *Bi-Monthly Bulletin* of the Institute. The aim of the research was to find the dissolving power of chlorine-water and of bromine-water upon gold and upon a series of alloys of gold and silver, the operations being carried out in revolving vessels. The dissolving tests were made in pint fruit jars of glass, 3 inches in diameter and 6 inches high, the covers being held in place by screw-clamps, and the joints made tight by rubber gaskets. The jars were rotated in the Richards revolving apparatus shown in Figs. 1 and 2. This is seen to consist of a horizontal shaft with pulley, having at either end a cylindrical box holding seven jars. A box has a wooden bottom, twelve staves connected by a stout wire, and sides of 2.5-pound lead. A jar is held loosely in a horizontal position by three sheet-steel springs screwed to the wooden bottom. A circular felt pad, *b*, tacked down between three springs protects the bottle against jarring. The barrel makes 4.3 revolutions per minute. The results are given in tables and diagrams. One diagram is reproduced in Fig. 3, the constant for each curve being the quantity of active reagent in 150 c. c. water; the percentage of extraction is given as ordinate, and the ratios of gold and silver in the alloy as abscissas. A supersaturated solution of chlorine in water acts more strongly than one that is merely saturated; a supersaturated solution can extract a satisfactory percentage of gold from a gold-silver alloy containing as much as 20 per cent of silver, while with the decrease of chlorine flow a certain amount (2.1 g.) and the increase of silver above 10 per cent, the extraction of gold falls off quickly. Bromine is also an efficient solvent for gold, giving extractions of 99.8, 98.2, 96.6, 93.2 and 92.2 per cent, with solutions of 8.97, 5.98, 4.48, 2.90 and 2.60 g. of bromine in 150 c. c. of water; when the bromine present sinks to 2.1 g. the yield in gold falls quickly to 34.2 per cent, and then diminishes more gradually. With the concentrated solutions (2.8 g. of Cl equivalent to 6.0 g. of Br.), chlorine is a slightly stronger solvent than bromine. For concentrations below 1.58 gram chlorine (equivalent to 3.4 gram of bromine) bromine is a better solvent than chlorine. The curves in Fig. 3 bring out forcibly the result which the per cents of large proportions of silver may have upon the extraction of gold.

Fig. 1. Fig. 2.



FIGS. 1 AND 2.—REVOLVING APPARATUS FOR CHLORINATION.

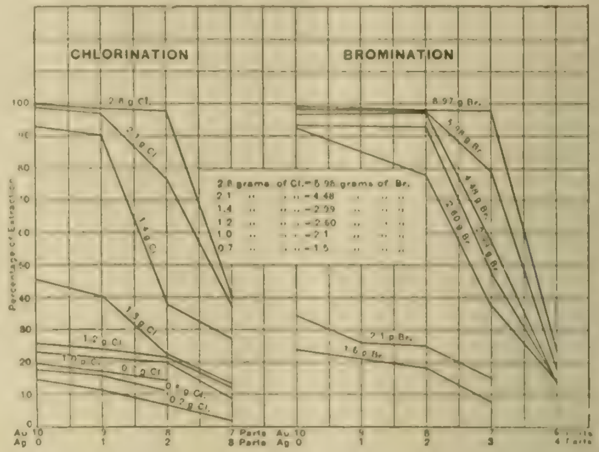


FIG. 3.—RESULTS OF CHLORINATION AND BROMINATION TESTS.

On account of limitations of space a number of abstracts had to be reserved for our next issue.

BOOK REVIEWS.

Electricity in Every-Day Life. By Edwin J. Houston, Ph. D. New York: Collier's Weekly. Three volumes, 584, 566 and 610 pages, respectively. Illustrated. Price, $4.50.

The contents of these books are best indicated by the headings of the different sections; viz., Electricity of High Electromotive Force (13 chapters), Magnetism (8 chapters), The Voltaic Cell (4 chapters), Other Sources of Electricity (4 chapters), Production of Electricity from Magnetism (8 chapters), Electric Lighting (21 chapters), Electric Power (10 chapters), Electrochemistry (5 chapters), The Telephone (6 chapters), The Telegraph (12 chapters), Annunciators and Alarms (3 chapters), Heating (4 chapters), Electro-Therapeutics (3 chapters).

Under each heading, the successive chapters take up first the early history, with considerable reference to the personality of the inventors, and then follow descriptions of the development of the subject, right up to date. The reviewer was unable to think of any recent development in any of these lines which he did not find indexed and described. The completeness of the work is therefore very meritorious, considering the great range of subjects treated.

Two or three marginal annotations on each page facilitate reference to the sub-divisions of the topics treated, and unusually complete indexes add still to the value of the work for reference. The type is large and clear, the paper, however, poor. It is a pity that the publishers could not have afforded a quality of paper good enough to prevent print showing through so plainly. The binding is neat and attractive; the numbers of the volumes should have been printed on the backs more conspicuously.

Aside from the paper, the work is in every respect the opposite of the ordinary "cheap" book. It is logically arranged, comprehensive and right up-to-date, and written by an electrician whose name is almost a household word, whose skill as a teacher has been derived from a twenty-five years' professorship of physics, and who has written these books in his clearest and happiest style. No books could be better suited for giving to school boys and girls in their teens, to the apprentice or artisan, to the business man or farmer, to the seamstress or the clerk, the minister or lawyer, a clear and reliable account of "Electricity in Every-Day Life." Neither will the scientific man find it beneath his attention, for the chronological arrangement and complete index will make the work valuable for reference.

Plant of the Shawinigan Water & Power Company.

The power development of this Canadian company is located at Shawinigan Falls in the Province of Quebec, on the St. Maurice River, about 85 miles from Montreal.

The company has just finished the installation of the largest water-wheel in the world, which was built by the I. P. Morris Co., of Philadelphia. This monster turbine is of 10,500 hp. capacity, is 30 feet from base to top, is 22 feet wide over all and weighs 182 tons. There is direct connected to this turbine a Westinghouse generator of 10,000 hp., and with this installation, the company has now a power house and machinery with a capacity of 25,000 hp.

In addition to this, 15,000 hp. is sold in the form of hydraulic power to the Northern Aluminum Co., and the Belgo Pulp & Paper Co. Other users of power at Shawinigan Falls are the Shawinigan Carbide Co., manufacturing calcium carbide; the Electro Manganese Co., manufacturing ferro-manganese, and lighting companies.

In addition to selling power at Shawinigan Falls, the company furnishes power in the city of Montreal for street railway and industrial purposes, transmitting at 50,000 volts over an equipment of two transmission lines parallel to each other and comprising the longest transmission line of such high voltage in the world.

At Montreal, the company has a terminal station where the current is stepped down and the frequency changed. In this station, during the month of March, there was put into successful operation the largest frequency changer ever built. It was manufactured by the Bullock Electric Manufacturing Co., of Cincinnati. The current from Shawinigan Falls is transmitted at 30 cycles, and is changed to 60 cycles for the use of the city lighting and power mains. The gigantic frequency changer consists of a synchronous motor directly connected to a generator and a starting motor all on the same shaft and mounted on the same base. A few figures will show the enormous size of this machine. The total length of the frequency changer is 30 feet and the motor and the generator are each 15 feet high, the total weight is 250 tons. The set, composed of a 7500-hp. motor driving the 5300-kw. generator, approximates 14,600 hp. There were already in the station five smaller 800-kw. frequency changers, so that the capacity of this sub-station is now doubled. The company is now delivering in Montreal a total of 9000 hp., and is making arrangements to increase this output to satisfy demands for additional power already made.

With its cheap power and good railway facilities, which are this year being added to by new lines of railroad, Shawinigan Falls possess advantages which offer large attractions to companies now located there, and has a great future before it as a manufacturing town.

Mechanical Desulphurizing Furnace.

In the following we give a description of the O'Brien mechanical roasting furnace which was worked out by Mr. A. P. O'Brien and installed at his works, the Richmond Guano Company, Richmond, Va., about four years ago. Within the past year the American Coke & Gas Construction Company has secured the rights to this roaster and has thoroughly perfected it mechanically. Some 25 installations made during the past few months have given such fully satisfactory results that the following description should be of interest to engineers and managers of metallurgical, chemical and fertilizer works.

The furnace is built in two distinct types:

Type A.—Plain furnace for desulphurizing metallic bisulphides, iron and copper pyrites, fines and concentrates, etc.

Type B.—Muffled furnace for desulphurizing zinc blende, pyrrhotite refractory sulphides, etc.

These furnaces embody a number of distinctive and valuable features which, for the most part, will be noted by reference to the accompanying illustration of the type "A" furnace.

The furnace consists of an iron shell enclosing and supporting brick shelves and lining. This shell is supported by channel and I-beam posts resting on a circular foundation wall. Two of these posts are connected by horizontal I-beams, upon which in the center is a ball-thrust-bearing, which supports the tapering, tubular air-cooled shaft. The I-beams supporting the shaft rest on set screws which provide for the adjustment and alignment of the shaft. This shaft may be removed while the furnace is under fire, and a new shaft may be used or the old one repaired without letting the fire go out. The air-cooled rakes are inserted in removable sockets in the shaft and cannot rise or turn; they keep a true and even bed on each shelf, being locked in place by means of a slot and key; they may readily be removed or replaced. The teeth of the rakes are in proportion to move the ore at a uniform speed from the center to the circumference of the shelves, and vice versa. The furnace is provided with ore feed hopper, and a mechanically-driven screw feed; in the brick shelves are provided a sufficient number of openings for the dropping of the ore from the upper to the lower shelves; tubular outlets discharge the cinder from the bottom of the furnace; these latter are fitted with closing slides so that the air drawn into the furnace may be regulated to afford proper combustion; a further regulation of air is afforded by adjusting slides in the doors opening to each shelf.

The furnace is provided with two sets of doors opening into each compartment; one set is placed diametrically opposite the other so that repairs may easily be made. An iron ladder designed for water cooling, if desired, is supported by the shell and runs alongside each set of doors, and the doors are made to lock when shut or open, so that the examination and working of the furnace is extremely simple. The furnace shaft and rakes are driven by pinion and rack; the meshing of the rack and pinion teeth is so governed that any possible obstruction within the furnace will cause the teeth of the rack to jump out of mesh, and the loud clinking noise will attract the operator's attention, who may immediately stop the furance by the throw of a lever and so prevent damage.

The alternate brick shelves are provided with luting rings, and the shaft with lute covers which prevent the escape of gas through any but the desired openings; this is also true of the furnace top plate. An elbow provided with clean-out and shut-off damper serves as a gas off-take; both the gas off-take and ore-feed are set to the furnace with gas-tight connections.

The furnace is mechanically driven, and any number of them may be run in series as is shown in the illustration of the furnace room arrangement; any one or more so run may be independently cut out by the throwing of a lever to a position which raises the rack teeth out of mesh with the pinion; this is readily done owing to the flexibility of the rack plate. The power required for the mechanical operations of one furnace equals approximately ⅛ of a horse-power.

A few of the distinct advantages claimed for this furnace are:

The arms are put in and taken out horizontally and without raising.

Consequently the shelves are closer together, and the radiant heat is better utilized, securing a more complete roast and greater efficiency.

Further, for the same reason, there is room for an additional shelf without increasing the height of the furnace.

For the above reasons the capacity of this furnace is increased at least 20 per cent. over any other of the same size on the market.

Far less power is required than for any other furnace of equal capacity.

The feed is automatic, positive and easily regulated from the floor, and is equally efficient for all grades of ore. This feed will work so accurately that the operator will know exactly the amount of ore being fed.

Repairs are much less than on any similar type of furnace. The rabble arms, which are the wearing part, are shorter and lighter than in any other furnace; owing to this and their air-cooling system they last longer and cost less to replace.

The air-cooling system is positive; cool air enters at the bottom of the tapering tubular cast-iron shaft and must pass through every division of the shaft and rabble arms before it escapes from the shaft top.

No stack is required, nor other artificial means of creating a draft through the central shaft.

The central shaft can be withdrawn and replaced in a short time without cooling or tearing apart the furnace.

The driving mechanism is light, simple and near the periphery of the furnace where it can easily be reached for oiling and repairing, instead of being, as usual, at the center of the furnace where it is almost inaccessible.

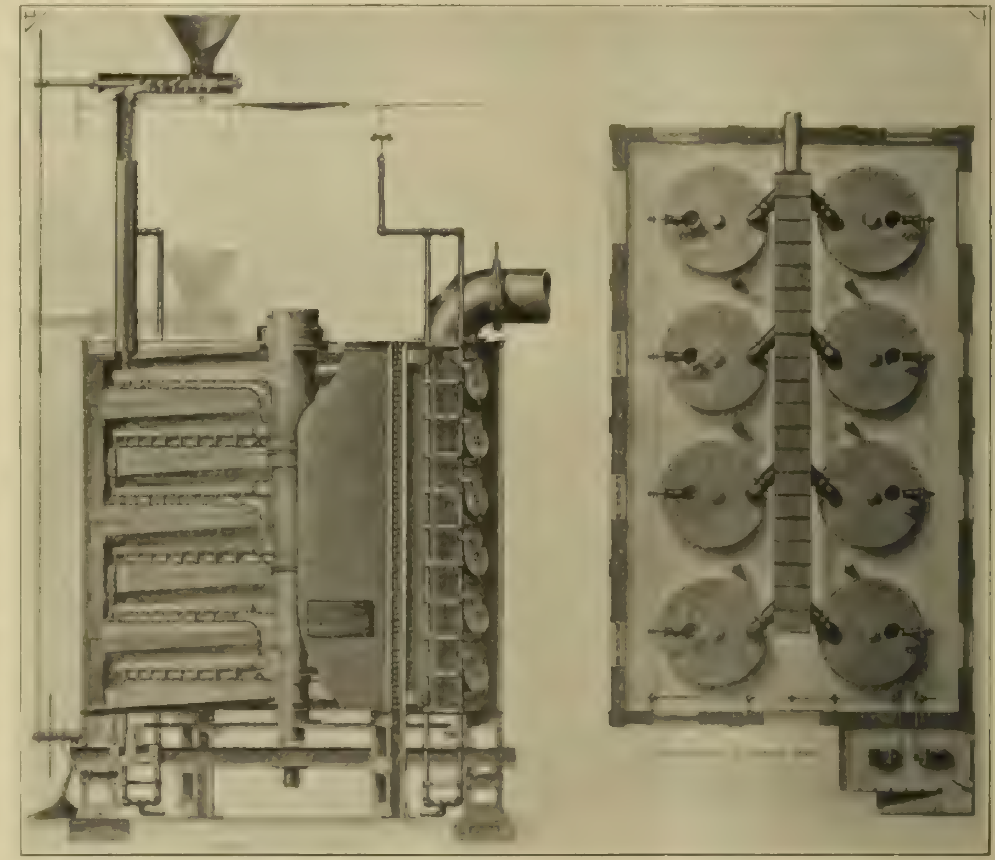
There are many other advantages which must be seen in operation to be appreciated.

The muffled furnace (type B) possesses all the advantages of the plain furnace, and closely follows the same in design, except that it is provided with combustion chambers for supplying extraneous heat for the satisfactory desulphurization of refractory ores. For this purpose chambers are built under each of the three lower roasting shelves to receive the fuel required to keep the heat of these shelves at the necessary temperature; the chambers are furnished with air inlet which may be so regulated that the complete combustion of the fuel supplied is assured therein; there are provided a sufficient number of flues, running up within the brick lining of the furnace, to carry off the products of combustion; otherwise the combustion chambers are air tight, and no escape of their products into the roasting chambers, or gases from roasting chambers to combustion chambers, is possible. In order to prevent undue radiation of heat an in-[illegible] lining of non-conducting material is set between [illegible] wall and steel shell.

An iron flue and stack connected to the flues from the combustion chambers lead the waste products off to discharge above the furnace. The additional heat required for desulphurizing the refractory ores can be applied either by natural or producer gas, fuel oil, pulverized fuel, or from an outside combustion furnace as may be most economical in any given locality.

The extreme mechanical simplicity and compactness of these furnaces, their great ease of operation and thorough regulation of all conditions affecting the roast, their construction, which minimizes the necessity of repairs, and the facility with which any necessary repairs may be made to any parts greatly commends them.

The type "A" furnace which is designed for roasting ores high in sulphur content requires no fuel on ores carrying from 30 to 50 per cent. of sulphur; such ores can be roasted down to a "dead" cinder by their own heat alone. It is adapted particularly for chemical and fertilizer works where the fumes from the ores are to be converted into sulphuric acid.



MECHANICAL DESULPHURIZING FURNACE

By passing these fumes or gases through an ordinary dust collector, a comparatively clean gas without material reduction of temperature is afforded, thus insuring the efficiency of the Glover tower and purity of the Glover tower acid; and by the use of especially designed cleaning system and absolutely pure SO_2 gas may be secured.

The type "B" furnace, designed for roasting refractory ores, is especially adaptable for the demands of metallurgical works, affording a large output of ore at a "dead" roast, or any required degree of desulphurization and securing a complete saving of flue dust and fume losses.

A number of these furnaces have been installed in works of varied character in this country and abroad and are exceedingly satisfactory in every instance. Further particulars regarding these furnaces may be obtained by addressing the American Coke and Gas Construction Company, 17 Battery Place, New York City, or their engineering offices, at Camden, New Jersey, which company has secured the sole right to the manufacture and sale of these furnaces in all countries, installing them under guarantee.

Bijur Storage Battery.

The first storage battery was devised by Plante, who formed two sheets of lead in sulphuric acid by a long and tedious purely electrochemical process. Faure and Brush avoided this long treatment by mechanically appplying the active mass to the electrodes. Then began an enormous activity of inventors who essentially worked in three different directions. One class worked along the lines of Plante, but endeavored to reduce the time of formation by the use of oxidizing electrolytes instead of, or in addition to, sulphuric acid, or by a purely chemical method of formation. A second class of inventors worked along the lines of Faure, but they were hindered in this country by the Brush patents, which only expired in 1903. A third class tried to develop an accumulator without lead, using a new electrochemical reaction; examples are the old Waddell-Entz and the new Edison batteries.

In view of this development it is certainly interesting to note that the latest type of commercial storage battery which, having been most carefully worked out, is now being placed on the market and manufactured on a large scale, is of the old Plante type, although a new (secret electrolytic) process of formation is used in its manufacture. In various respects, the Bijur battery differs radically in design, construction and manufacture from forms hitherto known in the storage battery art, yet its most conspicuous feature is a certain sound conservatism—the fact that its novel design is directly a consequence of combining old, well-known, sound engineering principles with results derived from the most reliable modern theoretical investigations on the reactions in the lead cell.

The battery is the result of several years' work on the part of Mr. Joseph Bijur, of New York City. With the collaboration of Dr. J. S. C. Wells, of Columbia University, in the earlier stages of the work, Mr. Bijur set out to produce a battery plate according to an ideal design which should be both mechanically and chemically perfect.

To get a mechanically perfect plate it is all important to provide a rigid structure, free from tendency to distortion and with the active material applied in such a way that it cannot be displaced. The ideal construction of a plate fulfilling this requirement is to make it up of small, finely sub-divided structures, each free to expand unrestrainedly, and firmly welded to a strong supporting and conducting grid. This construction has been adopted by Mr. Bijur. For this purpose a large amount of experimental work had to be done. To reduce the cost, automatic machinery had to be devised. But, above all, by no method of metal working then existing was it possible to commercially weld together two elements, of which one was a heavy, solid form and the other of very fine, thin parts. It was, therefore, necessary to devise a new method of metal welding which, after much patient work, in the face of seemingly insuperable difficulties, was successfully accomplished, and for the carrying out of which, on a commercial scale, an entire line of machinery was constructed on original lines.

But before describing the details of the construction we must refer to the chemistry of the subject which determines, in many respects, the satisfactory behavior of an accumulator.

The reaction during discharge is

$$PbO_2 + Pb + 2H_2SO_4 = 2PbSO_4 + 2H_2O.$$

The reaction during charge is the reverse. The e. m. f. of the cell depends on the concentration of the acid and decreases with it. Moreover, the concentration of the acid in the pores of the plates is the determining factor. Now, from the above equation it will be seen that acid is consumed during discharge, *i. e.*, the solution becomes more dilute, the voltage decreases. If the discharge is made at a high rate, the solution in the pores of the plates becomes more dilute than outside, because diffusion cannot bring fresh solution quickly enough into the pores. For this reason there is a further drop of voltage. Thus, the possibility of free diffusion does not allow the voltage to drop so quickly during a high-rate discharge. On the other hand, with free diffusion the voltage does not rise so quickly during a quick discharge. Hence, diffusion is a most important factor in determining the efficiency of a lead cell.

It is also of great importance with the respect to the phenomenon called sulphatation or sulphatization. While the formation of sulphate during discharge is the natural thing and the very reaction which gives the electrical energy, yet over-production of sulphate, and especially formation of sulphate crystals, is one of the worst diseases of a lead cell. According to one of the most reasonable explanations of this phenomenon, it is in intimate connection with the solubility of lead sulphate[1]; this seems to have a minimum in a 14 to 16 per cent H_2SO_4 solution. If the concentration becomes very much higher, or much lower, the solubility, and with it the liability, of sulphatation increases. For this reason it is so detrimental for an accumulator to stand for a long while in an acid which is too strong or too weak. Here, again, free diffusion is beneficial, since it prevents the concentration in the pores from becoming excessively high or low. These brief remarks do not approximately exhaust the subject, but they must suffice to indicate the importance of free diffusion. It is in this respect that the design of the Bijur battery is especially remarkable; his large surface of active mass is so arranged that the outside solution has always free access to the pores of the active mass.

As is clearly shown in the adjoining illustrations, the Bijur plates are composed of multiples of pure lead structures in the shape of gratings or "grills." These are welded to, and in one piece with, a stiff frame made of lead and antimony only. The weld goes clear through the plates and is produced without the use of tin, solder flux or any extraneous material whatsoever. The frame merges into the grill without joint. At each end of the grill a space is provided for its elongation by expansion, and provision is made for expansion sidewise. The result is a plate having the stiffness, strength and inoxidizable support of the alloy grid, rigidly held active parts, yet complete provision for their expansion without the setting up of any strains that could produce buckling.

The grills are composed of a multitude of thin partitions, open through from face to face. When in the untreated metallic state these grills are very open structures, having a large number of component ribbons running in the vertical direction, supported by heavier cross members running horizontally, which serve as conductors and give lateral stiffness. During the formation process part of the metallic lead is converted into oxide, which, by reason of the large increase in volume, nearly fills the openings. The ribbons on the positive elements have a large amount of metal left as a reserve for the oxidation which is attendant on the normal action of every storage battery. The vertical direction of the ribbons aids diffusion, since during discharge the dilute solution from the pores of the plates rises upwards, while fresh solution enters the pores.

When the oxide is formed in the rectangular cells it assumes a slightly elliptical shape, which, expanding into the rectangular containing space, produces a locking in that is so positive that nothing short of destruction of the plate can dislodge it. At the same time, even when the oxidization is carried to the point of apparently completely filling the cells, the tendency of the ellipse to grow larger along both its axes, insures the presence of a minute slot through the center of the oxide mass, which allows the flow of electrolyte to take place freely through it. The resultant oxidized plate, therefore, is rigid, the grills are firmly held, they expand without producing strains; each particle of oxide is firmly pressed against the lead from which it is grown, and the entire plate is open through and through to circulation and diffusion of the electrolyte.

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[1] See, for instance, Dolezalek, Theory of Lead Accumulator. American edition, page 140.

Owing to the character of the design and the structure of the oxide formed, the plates possess several features hitherto unattained in battery practice. "Buckling" cannot take place, and so far repeated attempts to intentionally produce distortion have failed. Also, since the perfect diffusion through the plate prevents too high or too low acid concentration in the pores, deleterious sulphation does not take place, even when an effort is made to produce it.

One of the important advantages gained by this form of design is that in a plate of given dimensions, 20 per cent greater capacity can be obtained with about 10 per cent more reserve lead than in any form of plate hitherto constructed. The fine sub-division of the oxide into small masses, combined with the fact that the surfaces of the plates are composed of ribbons on edge precludes the possibility of blistering or shrinkage that is liable to occur when plates contain oxides in large masses.

The makers claim that owing to the rapid acid diffusion full charge may be given the batteries at a voltage of 2.4 to 2.5 volts instead of 2.7 volts per cell, as is usually required. This low-charging voltage results in increase of efficiency, and reduces the size of boosters required to charge.

The structure of the plate forms a conducting network to all parts, insuring equality of action all over its surface, and this, together with the facility for escape of gas without dislodging active material, favors the ability to stand very high rates of charge and overcharge without marked deterioration.

In short, since storage battery plates can be calculated out scientifically and then constructed in exact accordance with the design, the series of developments and inventions of Mr. Bijur places the manufacture of storage batteries on the same basis as that of dynamo-electric machinery. These processes permit of any suitable forms of active parts and supporting frame-work being united into one integral structure at a low cost of production and without the introduction of any foreign [illegible]

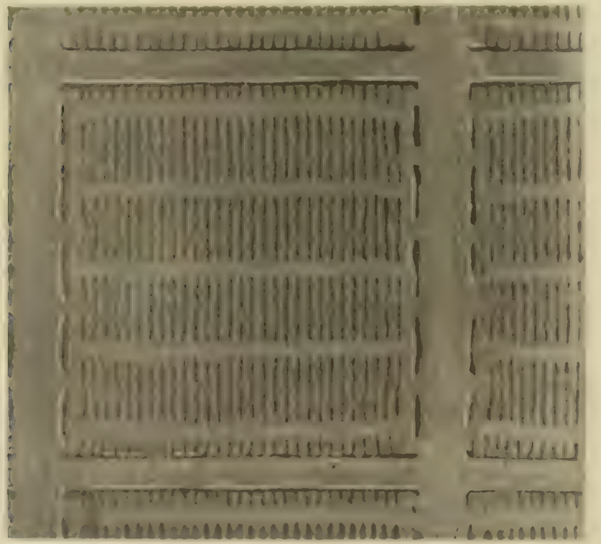
FIG. 1.—SECTION OF FORMED BIJUR PLATE.

The General Storage Battery Co., which is now putting elements of this type on the market under the name of the Bijur "High Duty" battery, has recently equipped a factory at Boonton, N. J., with special automatic machinery for the cheap [illegible] tion of this battery, where, by reason of an [illegible] cheap hydraulic power and the automatic machinery used, storage batteries of the highest grade can be produced, it is claimed, for a lower cost per kilowatt-hour capacity than is possible anywhere else in the country.

The company has also developed a new and complete line of automatic regulating boosters and cell switches, and other auxiliary apparatus for all classes of central station work and power plants, descriptions of which will appear later. It may be said that all of these devices differ as radically from the present known types as does the Bijur "High-Duty" battery from the conventional storage battery electrodes, and the improvement in methods and degree of regulation represents a marked advance in storage battery engineering.

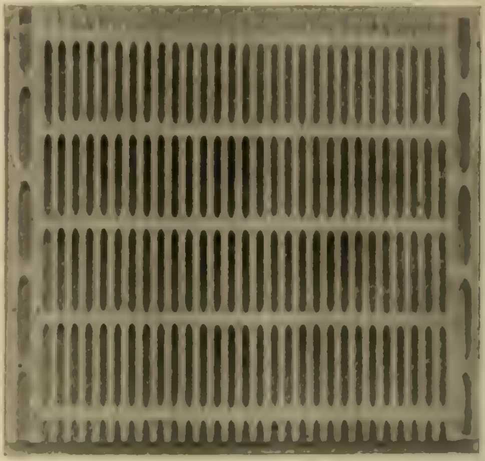
FIG. 2.—UNFORMED BIJUR GRILL.

A Preventive for Lead and Mercury Poisoning.

It has long been recognized as the result of extended researches that lead poisoning is due to the introduction of the poison from the hand into the mouth, while eating, and that this process goes on even if the workingmen, before taking their meals, have taken great care to clean their hands thoroughly with ordinary soap. The reason is that lead soaps are thereby formed which adhere to the skin firmly, and are not less poisonous than the original lead compounds. The lead is then introduced through the mouth into the system.

The only remedy is evident. It is to use such a soap that a harmless, non-poisonous compound is formed by the soap on the hands. This effect is claimed for the "Akremnine" soap, by the application of which non-poisonous lead sulphide is produced. The soap is stated to have the same effect in preventing mercury and copper poisoning. The Akremnine soap has been used for several years in Europe, and, according to published statements of Dr. Blum, Dr. Umbash, Dr. Merckens and others, has proven completely satisfactory, combining the advantages of ordinary soap with those of a disinfectant.

The Akremnine soap is now being introduced in this country by Mr. Felix Hamburger, 90 William Street, New York City

Welding of Broken Bosses on Rolls by Thermit.

When, in the beginning of the war in the Far East, the Russian battleships were disabled by the Japanese, considerable surprise was expressed at the remarkable quickness with which the Russians were able to make repairs. This was in most cases due to the application of thermit, which was carried in considerable quantities on every battleship. The Japanese quickly learned the lesson, and are now paying due attention to the possibilities of thermit.

In this country, the Goldschmidt Thermit Co., of New York City, commences to introduce thermit very extensively for more peaceful purposes, in making repairs of broken castings in railroad shops, steel plants, rolling mills, etc. The general possibilities of thermit for repairs were fully discussed by Dr. Hans Goldschmidt himself in our Vol. I, page 527 (see also Vol. II, page 404). In the following some more recent developments shall be dealt with.

In railroad shops the most important, although most difficult, problem is the welding of broken locomotive frames.* By means of thermit it is possible to perform the whole repair in the round-house within fifteen hours (against two weeks by the older method). There is no transference to the machine shop, no dismantling of the locomotive. However, this special problem requires considerable experience in the handling of thermit; nevertheless, one railroad company has already repaired more than forty locomotive frames with thermit with perfect success.

For general repair work in machine shops thermit is now very largely used, and recently some very interesting repairs of broken cast iron were performed. In one case it was the part of a machine used to press wheels on axles; this part bears directly over the hub of the wheel during the passage of the axle under a pressure of 60 tons per inch. On account of this high pressure the part had broken into two pieces, but was repaired by thermit with perfect success by the engineer of the company itself. This indicates considerable skill on his part, but it also indicates the possibilities which the application of thermit offers to every iron and steel plant.

Of special interest for rolling mills is the use of thermit for welding broken bosses on rolls. Hitherto it has been the practice to make the broken boss red-hot by means of a basket of coke placed upon it, and then pour liquid iron or steel on it. This is rendered unnecessary by the application of thermit, which does the work more simply, cheaply and effectually, as will be described.

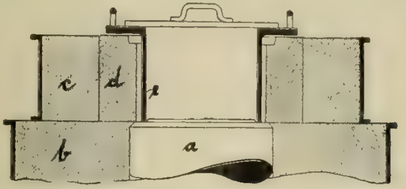

FIG. 1.—THERMIT WELDING OF BROKEN ROLL.

Thermit ignited on a cold surface deposits on it a layer of corundum (aluminium oxide) slag, which adheres firmly. To avoid this, a thin layer of cast iron or steel is poured onto the welding surface before thermit is placed on it, and to avoid the adhesion of corundum at the sides, a concentric iron ring is so fixed in the mold that it can be withdrawn after ignition of the thermit, as shown in Fig. 1. In this illustration *a* represents the roll, *b* and *c* dirt, *d* the mold. The method of procedure is as follows. The roll is firmly fixed in such a position that the welding surface lies horizontally. After placing the first mold, which has a diameter exceeding 2½ inches that of the boss to be welded, a cast-iron ring, with two lugs attached for lifting, is placed in the mold. This ring *e* is ½ inch thick, and stands off about ½ inch from the inner wall of the mold. It is constructed so as to overlap like a flange the upper part of the lower mold box without touching its edge. The ring must be well covered with graphite and the inner surface well smeared with clay and dried. It must tightly fit onto the roll, and, where necessary, spaces must be stopped with clay. The clay must not exude onto the inner face, as moisture on the welding surface is bad.

The flange-like rim of the ring is made to overlap the edge of the mold and protect it from any drops of liquid iron which may splash up during the reaction and pouring of the metal.

About 30 pounds of thermit are required to the superficial foot of cast-iron surface. This quantity will soften the metal to the depth of about 2 inches. For steel rolls, 10 per cent to 20 per cent more thermit is required than for cast iron. It is not necessary to make the roll red-hot; a basketful of coke placed inside the mold until the iron cannot be touched by hand is sufficient. A few sand pits surrounding the roll, and filled with a few hundred-weight of cast iron, will do as well, even with very large rolls.

After firmly imbedding the ring in the mold box, a thin layer of cast iron and steel, say ½ inch, is poured into it, which must completely cover the welding surface. A thin layer of powdered charcoal can be used instead, but has the objection of producing a spray of sparks.

The whole of the thermit, accurately calculated and weighed, is then quickly added and ignited in the usual way, viz.: a thimbleful of ignition powder is placed upon it in a small heap and lighted with a hot iron or a fuse. Immediately after ignition, the mold should be covered with an iron or wood lid, covered with clay on its under side; the wood will char, but not burn.

The reaction, even with large quantities of thermit up to 1½ cwt. at a time, is finished in half a minute, and the lid must at once be removed and the thermit stirred with iron rods so as to disturb the slag (corundum). The heat of the reaction, even with only 20 to 30 kg. thermit, is so intense that the workmen must protect their hands and faces in the usual way for work at crucible furnaces, by using colored glasses, and covering their hands with cloth.

After a few minutes cast iron or steel is poured out of the ladle, which must be in readiness, into the mold to about a third of its capacity. When cast iron is poured on to thermit during reaction, it is especially necessary to stir the metal thoroughly, in order to obtain a thorough mixture of the thermit iron with the cast iron and prevent any steel being formed on the welding surface, but if properly stirred steel never forms.

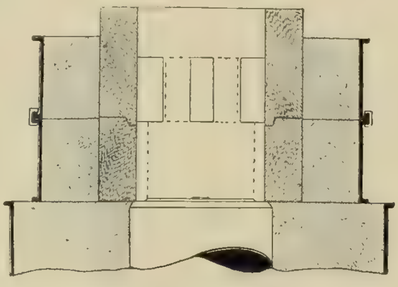

FIG. 2.—THERMIT WELDING OF BROKEN ROLL.

A number of stirring rods must be held in readiness, as the heat melts them quickly, until the cast iron is added. Two or three men ought to be employed simultaneously to stir the metal for several minutes, after which the iron ring is lifted out. The whole of the corundum will then be found adhering to the inner surface of the ring, and is lifted out with it; the ring is protected from melting by the adhering corundum and can be used several times; the ring prevents the slag from clinging to the mold itself.

After withdrawing the ring, more iron, constantly stirred up, is added, until the lower mold is filled. The remaining slag, which will have coagulated in small pieces, is carefully taken off before the second mold, Fig. 2, which may be of trefoil shape, is placed in position.

A great advantage is gained by this method of reducing the waste of metal resulting from the usual rift in the cast. The subsiding takes place very regularly in consequence of the addition of the thermit. The rifts have much less depth, particularly if kept open by pumping with feeding rods as long as possible.

The cast iron added in the process has a peculiarly beautiful and firm structure entirely due to the thermit. What takes place during the welding process is as follows: On the bottom of the mold a number of particles of iron are collected and brought to a very high temperature through the action of the thermit; these have a tendency to rise to the surface, while the upper parts sink to the bottom. This causes a very lively "play" of the iron, and the impurities (particularly of cast iron) are more readily separated off; the cast is therefore appreciably improved; the addition of the thermit also renders the cast iron more fluid, which can be observed even with an addition of only ½ to 1 per cent, but thorough stirring is absolutely necessary in order to obtain a good mixture of the specifically heavier thermit iron with the lighter cast iron.

If the break is cup-shaped, the hollow which is formed must first be filled with cast iron or steel, upon which the thermit is ignited. In order to heat up this greater quantity of metal, an excess of thermit over what is stated above must be used. Such hollow breaks could not be repaired at all by the old method of letting metal run over them. If the break is very slanting, part of the roll must be knocked off to obtain a somewhat horizontal surface, but there is no need to have it absolutely level.

Not only can cast iron or steel be repaired in this way, but a cast boss can be welded on to a steel roll and a steel boss on to a cast iron one. Rolling mills, being as a rule unable to obtain liquid cast iron easily, the latter application is of special interest to them. They need not send the broken roll away, but can repair it in their own shops with liquid steel, and avoid the stoppage of work often necessitated by a broken roll.

It has been calculated by several works who have taken this method into regular use, that the cost is barely half that of the old method, even when the iron which is allowed to overflow can be utilized for smaller and less important casts.

Some works have carefully tested the strength of such thermit welded rolls. Hammer tests have proved that any breaks occur outside the welds. It is interesting to note that test bars cut out of the rolls always tore a long way from the weld.

Of course all solid pieces of metal can be repaired in exactly the same manner as described for rolls.

Welds to be effected upon cast-iron pieces of less solid construction are more difficult, as, in consequence of the "tensions" that exist in every piece of cast iron, breaks occur near the welds. This is particularly the case in repairs of cast-iron boilers, pipes, cylinders, or pulleys. Even with such slight or divided pieces, however, a favorable result may be obtained by first annealing the whole piece. After welding wrought-iron pieces or steel castings, a break in the same place need never be feared if the work has been carefully executed, and the experienced iron founder will find his own way of overcoming all difficulty.

A New Portable Balance.

The accompanying illustrations show a new portable button balance just placed on the market by Wm. Ainsworth & Sons, the well-known balance manufacturers of Denver, Col.

This balance is of the columnless type which has come into such favor for field work during the past few years and is constructed to withstand a moderate amount of rough handling. It has a 5-inch beam, agate edges and bearings, fall-away pan rests, and is made with either single or double-rider apparatus. An [illegible] locking device locks the beam in position on the balance for carrying. The balance is [illegible]rated from the left-[illegible] and [illegible] [illegible] of improved design. Two level vials are placed in the [illegible] of a single circular level as sometimes used. The [illegible] polished mahogany, and a carrying case of [illegible] provided with a leather carrying strap. [illegible]

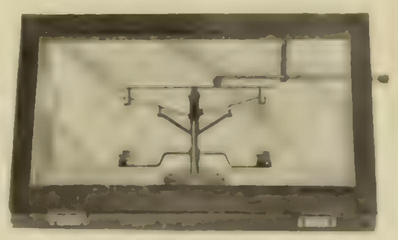

FIG. 1.—PORTABLE BALANCE.

A combination of the portable button balance with a port-[illegible] in a single carrying case is the most com-[illegible] possible. [illegible] the pulp balance pack in the drawer.

Full particulars are given in the Portable Balance Bulletin, sent on request by the manufacturers.

Storage Battery.

The unit accumulator made by the National Battery Co., of Buffalo, N. Y., has a positive plate of the Planté type, and a negative plate of the pasted type. It is built with the view of being used especially for high charge and discharge rates, combined with long life and high efficiency.

Positive plates of the Planté type have a natural tendency to increase in volume or "grow," on account of new active material being formed. In many cases this growth results in the distortion of the plates while in service to such an extent that they have to be removed at frequent intervals to be straightened, which is an expensive operation. In the construction of the positive plate of the unit accumulator, room is provided to care for this natural growth. The units or active portions are hung in a rigid lead frame, with ample space at the bottom, and at the two sides of each unit to allow for expansion, without in the least straightening or distorting the supporting frame.

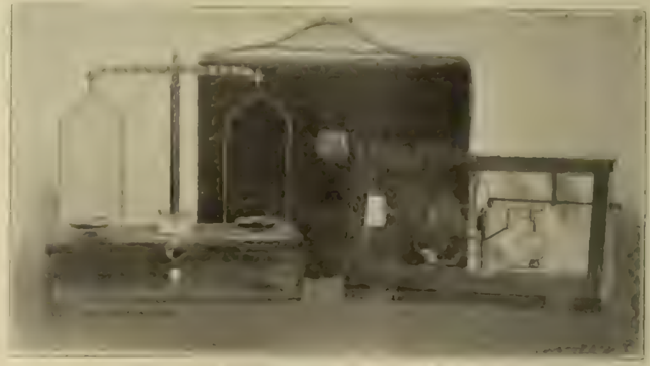

FIG. 2.—COMBINATION BALANCE OUTFIT.

For the negative plate a grid has been developed in which the natural molecular shrinkage of the active material has been provided for, and taken advantage of, so that the contact between active material and grid is maintained as the shrinkage progresses, owing to the fact that the active material enclosed in the contact portions is drawn more tightly around the grid projections or pins. The contact between the active material and the grid is therefore rather improved. The unit accumulator is made by the National Battery Co. in six different types, which are described and illustrated in a pamphlet recently issued by this company.

Industrial Notes.

Emil Greiner, manufacturer and importer of laboratory glassware and apparatus, announces his removal about May 1 to more commodious quarters, at 45 Cliff Street, New York City.

Nernst Lamp.—The Nernst Lamp Co., of Pittsburg, Pa., has issued an artistically illustrated pamphlet on Church Lighting by Electricity, for which purpose the Nernst lamp is specially suitable.

Among the metallurgical machinery shipped during the last three weeks by the Power & Mining Machinery Co., of Cudahy, Wis., were a complete copper converting plant for the Omaha works of the American Smelting & Refining Co.; a 100-ton copper smelting plant for the New England Milling Co., of Greenfield, Mass.; a carload of steel furnace jackets for the new works of the Canadian Copper Co., Copper Cliff, Ont.; a 10-stamp mill complete for Trapper's Flat Mining & Milling Co., Nevada; a Huntington Mill concentrating plant complete for the R. E. Briggs Co., Mexico, and cyanide tanks for the Imperial Gold Mining Co., of Deadwood, S. D. Fifteen

McCully crushers were sold during the first three months of this year.

Electric Power in Industrial Applications.—The number of motors in use ten years ago in manufacturing plants was very limited, but as the size of factories increased, spreading over greater areas, the older forms of power transmission and machine drive became more impractical, and electricity rapidly moved to the front. This increase in the applications of electric motors has grown to such an extent that it would be difficult to name a single industry into which this form of drive has not entered. As a single instance it may be mentioned that the Alliance Machine Co. and the Morgan Engineering Co., both of Alliance, Ohio, last week placed orders for a total of 120 crane motors, with an aggregate of 2000 horse-power, with the Westinghouse Electric & Mfg. Co. They vary in size from 1 to 100 horse-power. The Pennsylvania Railroad Co. on the same day contracted for twenty induction motors, to be added to their present equipment, and the Bethlehem Steel Co. entered an order for direct-current motors to be used in their mills. Numerous smaller orders would make up a list of applications which would cover the whole field of motor-driven machines.

The executive offices of the American Steel Foundries until lately were located at No. 74 Broadway; with the object of concentrating all of the departments it was found necessary to lease the entire eleventh floor of the recently completed building known as No. 42 Broadway, and henceforward communications should be sent to this new address. The executive officers of the American Steel Foundries are inaugurating, simultaneously with the removal, a new system of accounting and distribution of orders, which will improve the organization and simplify their work. The output of their eight plants comprises, mainly, all kinds of steel castings for railway and marine purposes, besides special castings for mining, metallurgical and general engineering purposes.

Westinghouse Electrical Apparatus.—That the steam turbine and turbo-generator are destined to be one of the greatest power developing and distributing factors is evidenced by the number of units of this type which have been installed, and which are now in the process of construction. Owing to the restrictions placed by reciprocating-engine speeds upon the designs of engine-type generating machinery, their dimensions and bulk, as also the cost, have increased enormously in the past few years with the increase in capacity. With the advent of the steam turbine the speeds have increased so as to secure in generator construction minimum bulk and cost consistent with strength and durability. A striking example of this may be seen in the power equipment of the Rapid Transit Company in New York, where turbo-generators, with a rated output of 5000 kw., weighing 234,000 pounds, run at 750 revolutions per minute. Generators of the same output, driven by reciprocating engines at a speed of 75 revolutions per minute, would weigh 980,000 pounds. Orders for eight turbo-generators have been placed with the Westinghouse Electric & Mfg. Co. in the past few days, mostly for 400 and 500 kw. units, with one 2000 kw. and one 2500-kw. machine. Turbo-generators are now being installed in many places where it becomes necessary to increase the output of the station, and where it is not possible to increase the floor space, and also in such cases where the low weight is of importance, in view of transportation difficulties (f. i. to mining camps). Among a great number of orders recently placed with the Westinghouse Co. for electric machines of various types, an order of the Pittsburg Reduction Co. is of special interest to our readers. They will install two new direct-current generators, each rated 2200 kw. at 500 volts. These machines will run at 140 revolutions per minute, and embody all the lastest features of Westinghouse design.

The new plant at Ithaca, of the Morse Chain Co., Trumansburg, N. Y., will consist of machine shop, forge shop, hardening and tempering shop, wood shop, pattern storage, foundry and offices. The power plant will consist of Westinghouse single-acting compound engines direct connected to 80-kw., 125-volt, direct-current generators from the World's Fair at St. Louis, supplied by Babcock & Wilcox boilers at 130-pound pressure. The machine shop equipment will be modern in every respect, with individual and group drives from 110-volt motors. Work of completing the details is now progressing rapidly, preparatory to commencing work of erection in the spring. The silent chain of the Morse Chain Co. was described in our Vol. II, page 515.

Price List of Chemicals.—We have received from Messrs. Chas. Cooper & Co., 194 Worth Street, New York City, their regular price list of chemicals for laboratory and industrial work. The price list is issued monthly, and is sent on request to all interested parties.

The Industrial Laboratories, of 164 Front Street, New York City (Dr. J. E. Teeple, director), which were established for chemical research and analyses, have recently purchased a large part of the splendid German Government exhibit of chemical laboratory apparatus at St. Louis. Their laboratory is now believed to be one of the most completely equipped with modern facilities in the world.

The Crocker-Wheeler Co, has been awarded the contract for four 4000-kw. three-phase 60-cycle alternators for a hydro-electric station at Lockport, Ill., near Chicago. The company has been building and marketing for the past eight months a line of alternating-current generators of the engine, belt, water-wheel and turbine types, according to the renowned designs of the Brown-Boveri Co., of Switzerland (who, by the way, have installed numerous machines in various prominent electrometallurgical works of Europe). This latest contract, with the recent order for three 4000-kw. alternators, to be driven by gas engines, places the Crocker-Wheeler Co. in the front rank of American manufacturers of alternating-current generators of large capacity.

Price List of Chemicals.—A neat "blue book," recently issued by the Charles E. Sholes Co., 164 Front Street, New York, contains one of the most complete and conveniently arranged lists of chemicals which has ever come into our hands. The list has evidently been prepared with a full appreciation of what is needed for laboratory work as well as for commercial purposes. The chemicals are arranged in alphabetical order. The president of the company is Mr. Charles E. Sholes, who is widely known through his former connection with the General Chemical Co. as assistant manager of the sales. The price list distinguishes between C. P. (construed to mean strictly chemically pure), U. S. P. (indicating a degree of purity which fully conforms to United States Pharmacopœia standards), and Pure (a grade of their own, manufactured to meet special commercial requirements).

National Electric Co.—The First National Bank of Milwaukee has had assigned to it about two-thirds of the capital stock of the National Electric Co., as security for loans. The bank has now taken over the control of the latter company, and John I. Beggs, Charles V. Pfister, Frederick Vogel, Jr., J. H. Van Dyke, Jr., all First National Bank directors, have been elected directors of the National Electric Co., replacing S. W. Watkins, president; F. H. Bigelow, chairman of board; F. C. Randall, director, vice-president and general manager, and Gordon Bigelow. Former Directors A. N. McGeoch and B. T. Becker will remain on the board. John I. Beggs has been elected and accepted the presidency of the company, and will direct its affairs. Mr. Beggs announces that the business of the company will be actively continued and all contracts promptly completed. The indebtedness is now being ascertained, and when it is known, a meeting of the creditors will be called to consider the best plan to protect all creditors and promote the future welfare and progress of the company, which Mr. Beggs believes can be made successful and profitable.

Personal.

Mr. C. C. **Speiden**, manager of the chemical department of A. Klipstein & Co., sailed with his family on April 29 for a few weeks' vacation in Europe.

Messrs. **Byrnes & Townsend**, patent lawyers in Washington, D. C., announce that they have associated with them Mr. **Eugene C. Brown**, late examiner of alternating current machinery and system in the United States Patent Office, and that they give special attention to electrical, chemical and metallurgical lines, including soliciting, expert testimony and interference and court practice.

In the investigation of the internal affairs of the Equitable Life Assurance Society, to be conducted by a committee of the Board of Directors, Mr. Henry C. Frick, as chairman of this committee, has selected as his principal assistant Mr. F. W. **Haskell**, widely known in electrochemical and metallurgical circles as the president of the Carborundum Company, of Niagara Falls. The New York *Herald* writes: "Mr. Haskell was formerly secretary and treasurer of the Southwest Connellsville Coke Company at Pittsburg, and is a man who has a considerable reputation as an auditor and mathematician."

Mr. F. E. **Drake**, recently president of the Lanyon Zinc Company, and formerly director of machinery and electricity for the United States at the Paris Exposition of 1900, sailed on the Kaiser Wilhelm II for Europe, where he will take over the management of the interests of the Chicago Pneumatic Tool Company, whose business on the Continent has assumed such proportions as to require the establishment of a principal office at Berlin, where Mr. Drake will reside. Following the Paris Exposition work, Mr. Drake devoted nearly two years to the reorganization of the works of the Union Electricitäts Gesellschaft, which is now controlled by the Allgemeine Electricitäts Gesellschaft, so that he goes to a field in which he has a large acquaintance and a wide experience. During the past two years Mr. Drake has paid close attention to the possibilities of electric smelting, and has directed some very important experiments in electric furnace methods.

Mr. **Wm. Howard Cole**, who has been connected with Goldschmidt Thermit Co., of New York City, in the capacity of chief engineer for the past nine months, has resigned his position and has left for Europe, in order to pursue professional work in connection with electric tramways. Mr. R. F. **Kelker**, Jr., has taken up the position as engineer for the Goldschmidt Thermit Co., and will superintendent the Street Railway Department. Mr. Kelker was lately connected as assistant engineer with the Brooklyn Heights Railroad.

Mr. **Chas. B. Jacobs** has opened a chemical and electrochemical laboratory at 52 Beaver Street, New York, for the purpose of carrying on general consulting and industrial research work, especially examination of and report on new processes, improvements in existing processes and plants, research and development work on the utilization of waste products, etc. Mr. Jacobs was graduated at the School of Chemistry, Lehigh University. He was for several years connected with the General Electric Co., in charge of the chemical laboratory at Schenectady, and for the past seven years has been actively engaged in experimental and development work, having been closely connected with some of the recent developments of electrochemical industries at Niagara Falls.

Digest of U. S. Patents.

Compiled by Byrnes & Townsend, Patent Lawyers, National Union Building, Washington, D. C.

Electric Smelting and Reduction Processes.

(Continued from page [illegible].)

575,788, January 26, 1897, T. L. Willson, New York, N. Y.

[illegible] is produced in accordance with the general [illegible] of [illegible], powdered alumina and carbon, intimately mixed, being heated in an electric arc in the presence of a base metal capable of alloying with aluminium. It is stated that the output of product for the energy consumed is nearly doubled by the use of an alternating arc of appropriate volume and frequency, it being alleged that the alternate expansion and contraction of the gases produced by the arc keeps the mass in the furnace in constant and rapid vibration, whereby fresh material is automatically brought into the zone of reduction. In a specific example a current of 1500 amperes at 55 volts and having a frequency of 60 cycles per second is named. The specific application of this method to the production of calcium carbide is described.

577,802, February 23, 1897, Gustaf M. Westman, New York.

A furnace for the treatment of mispickel and other ores of arsenic comprises a closed chamber in the bottom of which is a pool of metallic lead, maintained melted during the operation and serving as the cathode of an electric circuit. The anode is an iron block adjustably supported in the furnace roof. The sides of the anode are enameled with a slag composition in order to prevent the adherence of metallic arsenic. The arsenical ores are introduced between the electrodes and heated by the current, the arsenic distilling into a suitable condenser while any precious metals are retained by the lead cathode. The residual ferrous-sulphide is tapped from the furnace at the close of the operation.

588,266, August 17, 1897, G. De Chalmot, Leaksville, N. C.

Smelts phosphate rock to render it more soluble. The rock is broken into lumps and fed into an arc furnace, wherein it is continuously melted and overflows at one side. The furnace has side walls of phosphate rock and a horizontal bottom of carbon, supported on an iron plate and serving as one electrode. The other electrode is a group of carbon bars depending into the furnace. The molten phosphate is run onto a horizontal, hollow, revolving iron drum, which may be internally cooled by water. Sand is simultaneously fed onto the drum, to cover it with a protecting layer and to react with the molten phosphate with the formation of a silico-phosphate of lime. The product falls from the drum into a water tank and is thereby cracked in all directions. When most of the water in the tank is evaporated it is inverted, and the phosphate falls onto a grating. It is then crushed to powder, for use as a fertilizer. The percentage of phosphoric acid is thus raised from 23 to 29.50, and the portion which is soluble in Wagner's citrate solution from 1.39 to 9.50. The liquid phosphate in the bottom of the furnace is not run off, as it contains calcium phosphide, due to reduction at the carbon bottom.

589,161, August 31, 1897, Frederic Chaplet, Laval, France.

Produces hard bodies for cutting chrome steel, rubies, sapphires and diamonds. The bodies contain a metalloid such as carbon or silicon, and a metal, such as titanium, chromium, molybdenum or tungsten, or three of these constituents. Specifically, produces silicon titanid; silicon carbo-titanid; chromium silicide, molybdenum carbo-silicide, and tungsten carbo-silicide. The charge usually contains oxides of the metals and carbon in the form of pulverized coke, the carbon being either in equivalent proportions or excess, according to the desired product. The mixture is smelted and allowed to cool and crystallize.

589,415, September 7, 1897, G. De Chalmot, Leaksville, N. C.

Produces crystalline silicon by first producing an alloy of silicon and a silicide of calcium, iron or manganese. The alloy is boiled with hydrochloric or sulfuric acid, then treated with hydrofluoric acid and washed, whereby the silicides are dissolved out. Calcium silicide can be separated by running water, the resulting silicide acid being washed out or dissolved in an alkaline solution. Mentions a silicon alloy of tungsten and silver containing free silicon. The charge, a mixture of silica or a silicate, a metallic oxide or salt and carbon, is smelted with an arc, the current being high and the voltage low except at starting. A direct current is employed and the alloy is deposited upon the cathode.

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New York, June, 1905.

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Zinc in the Electric Furnace.

It is with peculiar pleasure that we approach this subject, for zinc is "par excellence" the electrical metal. In the first half of the nineteenth century, zinc as the negative of the primary cell was the only source of electrical energy available to the experimenter. Without it the researches of Faraday, which are the foundations of the electric dynamo, would never have been possible. Up to the present day it has a wide use in practical electrical engineering, though to-day secondary cells are rapidly displacing the primary battery. Electricity thus owes a debt to zinc. Electrolytic copper-refining has, by its cheap separation of the precious metals and attendant production of a pure high-conductivity copper, been helpful to electrical engineering. The great basic future use for aluminium—a metal commercially possible only by electrical aid—will be for electrical conductors. In both these instances electrometallurgy reacts broadly on electrical engineering, and there is a "community-of-interest" idea in their relations.

Dr. F. Meyer gave, in our January issue, a review which showed that there has been real progress made in the electrostatic separation of low-grade zinc-lead-silver ores. This is a courteous and gentlemanly coming of aid on the part of the metal's old rival—the electrostatic generator—to the metal. Electromagnetic separation of zinc-iron and zinc-iron-manganese ores is further evidence of the curious interdependence of metallurgy and electricity. Many attempts have been made to treat these low-grade ores by leaching and subsequent electrolytic reduction to metal, but either with this or with the Swinburne-Ashcroft process, there has been but small commercial success attained. Work has also been done since the days of the Cowles Brothers, in 1884, in the domain of reduction in the electric furnace. This had but little success up to the present day. However, Mr. G. De Laval, a versatile inventor, as is seen by his cream separator and his steam turbine, has developed a commercial electric zinc furnace, as we noted several months ago. His plant has been producing for some time electric spelter of extreme purity. This metal is pure enough to be used for the manufacture of the brass which is turned into cartridges at several governmental factories in Europe. It evidently has the requisite "physical properties" to stand the most rigid test, and is sold at a profit, because its high quality commands a high price. De Laval has used his commercial furnace almost exclusively on redistilling impure spelter or spelter products. He has, however, made some trials on the lower grade zinc ores found in abundance in the Scandinavian Peninsula. Whether he has preferred to keep tonnage of plant at maximum by working metal, and so does not care to use ore, or whether he is not now able to treat ore at a profit, is at this date unknown to us. The production of high-grade spelter in this country is limited to the New Jersey Zinc Co., an old, well-established, strong conservative concern, and the Edgar Zinc Co., controlled by the American Steel & Wire Co., a subsidiary company of the United States Steel Corporation. The brands, "Horse Head," "Bertha" and "Glendale," sell for from 2 to 4 cents above the market for "Prime Western." Here it is to be noticed that electrometallurgy has won its first victory in refining metal, just as it has done in the case of crucible steel or electrolytic copper refining, and electrolytic nickel refining. In these days of small profits, $40 or $80 per ton looms up as a pretty large differential, and it is believed that negotiations for the exploitation in this country of the De Laval furnace are under way at this time. This achievement gives encouragement to the solution of the problem of treating ores in the electric furnace. Heretofore, electric furnaces working on zinc ores have resembled the horse of Haroun-al-Raschid. This horse, it will be remembered, possessed a most glossy coat, beautiful ears and head—in short, it was the perfection of equine beauty. It had, however, one great fault. It was quite dead.

These experimental zinc electric furnaces were also perfect in all save one respect—they did not make any metal. Imperfect condensation caused the formation of "blue-powder," or zinc dust. If De Laval has solved the problem of the condensation of zinc, other men can effect it, too. If a large scale zinc furnace could be produced it would have exactly the same advantages that we have on several occasions enumerated for electric crucible steel and electric brass furnaces: (1) Large units in place of small units, with consequent smaller labor charge; (2) putting heat inside of furnace instead of forcing it through fire-clay; (3) ease of control over heat and thus over smelting operation; (4) if the furnace was durable, lower cost for renewals. There is no reason why this cannot be effected on a large scale, or why the same change from an externally heated retort to an electric furnace cannot be made in distilling and condensing zinc, as has been made in case of phosphorus and carbon disulphide. Whether this can be successfully done commercially is less certain. It is to be noted zinc is a much more important article of commerce than either phosphorus or carbon disulphide, and so more work has been done on the old zinc process than was the case with them. Further improvement of the old process is possible with zinc. Its metallurgy, though apparently simple, is really extremely complex, and so perhaps the right conditions cannot be held on large-scale electric furnaces. We hope that some one will take up this problem and try it out to the uttermost—to a flat failure or a pronounced success. It would be a satisfactory triumph of electrometallurgy if zinc ore were concentrated electrostatically and electromagnetically, and the concentrate reduced and refined electrothermically. But as in all things we shall see what we shall see.

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The Effect of Ferro-Alloys.

The ferro-alloy industry is dear to a great many of our readers. The introduction of alloy-steels for high-speed tools has brought about a revolution in machine-shop practice. True, the production of special alloy-steels, when measured by tons and compared with the production of pig-iron, is ridiculously small. But this is not the way to look at the matter. Mr. R. A. Hadfield, one of the foremost among those who did recognize the importance of this subject at an early moment, was perfectly justified in saying in his recent presidential address to the Iron and Steel Institute that if alloy-steels were now taken away there would be an end of progress, so much does advancing civilization depend upon the work of the modern steel metallurgist. Many of our conveniences, our comforts and—in war material—many of our defenses, are directly due to metallurgical research and the practice that springs from it. While the use of alloy-steels has been so rapid as to assume a revolutionary character, the manufacturers of ferro-alloys have not been sleeping, and it is with particular pleasure that we emphasize the part which the electric furnace has played in this field. But all this matter has been so fully discussed in articles and serials in our columns that we may refrain from saying here anything more. What we wish to call attention to in these notes is the fact that in spite of all practical progress we are far from having the subject well in hand; we are still very far from an exact classification of the practical achievements so as to be able to say that we perfectly understand the effect of the introduction of certain elements into steel.

How complicated the problem of solid solutions really is may best be judged by comparison with dilute aqueous solutions. In the latter case, whether or not we pretend to know the exact molecular or ionic constitution of the solution, yet to a certain extent we are enabled to predict what change in the physical properties (electric conductivity, etc.) will be produced by the addition of a given quantity of a certain salt. Here we have to deal with additive properties. But as yet we can predict very little of the kind in the case of solid solutions. Above all, however, the physical properties of an aqueous solution are determined by its chemical composition. This is not the case with solid solutions, as may be quickly appreciated in the case of iron-carbon alloys from Roozeboom's diagram (for instance, our Vol. II, page 49). In this case we have to do with a variety of compounds and solid solutions, and the physical condition of a sample of carbon-steel depends on the way in which it was cooled from the state of fusion; it depends on its previous history. Thus chemical composition is not everything, and chemical analysis cannot reveal everything we want to know concerning the alloy. If we consider how extremely complicated the study of alloys of iron with carbon alone is in itself, the increased complication will be at once manifest, which results from the introduction of another element into steel, thus giving rise to the possibility of further phases. The difficulty of a systematic investigation of such steels containing various elements was forcibly brought out in the introduction of Dr. Goldschmidt's article published in our last issue. Since the previous history of an alloy determines its physical properties, apparent differences in the results obtained by different investigators with alloys of the same chemical composition may be explained. There has already been done an enormous amount of research work in this field, but yet not too much. We are only in the beginning.

The limitation of chemical analysis as indicated above is well emphasized by some of the results recorded in this issue by Dr. Hans Goldschmidt, with respect to the effect of titanium on the strength of steel. In spite of practically the same chemical composition there is a distinct difference in the physical properties of the two samples, when treated with titanium thermit or not. It is thus clear that to understand the subject, chemical analysis must be supplemented by microscopic investigation of the structure, and the results must be considered in the light of the theory of solid solutions, on the basis of Gibbs' and Roozeboom's views. However, the effect of the titanium-thermit treatment just noticed emphasizes another point. Here we have not simply the production of titanium-steel. (As a matter of fact, we have yet not found any exact and clear-cut statement of the effect of the introduction of titanium into steel.) Dr. Goldschmidt carefully emphasizes that at least part of the effect of the titanium-thermit treatment is due to the mechanical stirring of the bath, and the combination of nitrogen with titanium. For exactly the analagous reason, copper-silicon has long been used for copper castings, in order to combine with silicon the oxygen of the cuprous oxide dissolved in the copper. There is still another point. We know that the "occlusion" of gases may have an enormous effect on the physical properties of a metal; thus the extreme hardness of electrolytic iron was found by Prof. C. F. Burgess to be due to the occlusion of hydrogen in it. "Occluded" gases may play a similar important part in other metals. However, enough has already been said to indicate the great complication of the problem while the necessity of solving it is manifest.

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Metallurgical Geography.

The question of where a metallurgical plant should be placed depends on conditions favorable to the cheap production and easy sale of the finished product. There are three main factors which enter into the determining of the best location. These three factors are essential to production, the mine, the market and the fuel supply. These are fixed while capital and labor are flexible. In general, the smelter or concentrating mill is placed near the mine, while the refinery is placed near the market—a perfectly natural and to-be-expected distribution of metallurgical work. This is always the case where the ore is susceptible of being concentrated into a crude product in the ratio of several tons of ore to one of matte or metal. The location of both smelter and refinery is further influenced by the adjacency of the fuel supply. As the attraction of the two forces on a particle results in a dynamic equilibrium, so also will the interaction of economic forces result in some sort of a stable equilibrium. Thus iron ore from Lake Superior meets coke from Connellsville most favorably on the shores of Lake Erie. Silver-lead ores are either smelted at Leadville with coke brought from the Pueblo district, or are brought to Pueblo and smelted there. The base bullion from these Colorado plants, and also the Utah and Mexican plants, is brought to some center of consumption, Omaha, Denver, Chicago or New York, to be desilverized and refined. Within a trust, which is an industrial unit, there is free play of economic forces, and the result of these forces is advantageous to cheap production. Nearness to a financial center is another reason for placing the refinery near the market, for refined metal is first-class collateral for loans. Then there is a floating supply of labor in all large cities

A case similar to that found with lead is found with copper and nickel. The largest nickel smelter is at Sudbury, while the refinery is at New York Harbor. Copper is usually brought into a crude product, either "converter bars" or "anodes," at the mines or at some railroad center near the mines, while the refining is effected to a large extent in five large plants on New York Harbor. Zinc is usually concentrated mechanically or electromagnetically near the mine, but is usually brought to metal near a coal field or some other supply of potential energy, as natural gas. The larger proportion of coal, two to four tons needed to treat one ton of ore, makes the fuel supply the dominant factor in the metallurgical geography of the zinc plant. Aluminium, which resembles zinc somewhat chemically, and in a broad way metallurgically, is brought in form of concentrated pure alumina to Niagara Falls, Massena, or Shawinigan Falls, where it meets its source of potential energy—the electric current.

In the location of plants, many interesting phases of development will arise in the future. Of great bearing is the fact that many manufacturing centers are usually not remote from the coal fields, and often situated directly in the coal fields. Fortunately, for the wide industry of the country, in more than half of the States are found coal supplies. The vast sedimentary deposits of rock in North America are broken through only in spots by igneous intrusions. Coal measures are found associated with these other sedimentary deposits. In the past this wide distribution of potential energy has had a peculiar, fortunate influence on American industry. As a direct result of coal, Pittsburg, Birmingham, Chicago, Cleveland, Pueblo have arisen as great metallurgical centers. New York, though near the anthracite fields, with its great harbor and its connection with domestic and foreign commerce, is in a different class, almost by itself. In the future there will be a further concentration of metallurgical plants and the resulting associated manufacturing plants to these centers. Along with this there will be considerable development of electrochemical works near these centers. The water-power installations, as at Niagara, Shawinigan, the "Soo" and elsewhere, have two special advantages for electrochemical industries; first, the low cost of power and, second, the fact that no capital is tied up in power plants. This for a new enterprise allows all possible capital to be invested in the plant. For these reasons there is every likelihood that these centers will grow. On the other hand, with cheap coal or blast furnace gas, power is nearly as cheap as at the water falls. Coal is used for many other purposes, evaporating liquids, drying solids, roasting and calcinating materials, melting metals, etc., etc. This will strike a balance in some cases in favor of the manufacturing center near coal. Strong competition in the lines of prime-movers and electric generators, especially that competition due to the entrance into this double field of the largest engine building concern and the two large electrical companies, guarantees that the capital outlay will be kept low. Thus, the rapid and free growth of America will increase by the interdependence of metallurgy and electricity, during the twentieth century, the importance of these wonderful coal fields so extended and so rich. It will be to the lot of some future Mahan to sketch the influence of coal power on industrial history.

Buffalo Meeting of the American Chemical Society.

The following preliminary programme has been issued for the thirty-second general meeting of the American Chemical Society to be held in Buffalo, N. Y., from June 22 to 24. The sessions will be held in the rooms of the Buffalo Society of Natural Sciences.

Thursday, June 22, 10:00 a. m.—After a brief address of welcome by Herbert P. Bissell, on behalf of the Buffalo Society of Natural Sciences, and a response by President Francis P. Venable, for the society, the following addresses will be given:

"The Classification of Carbon Compounds," by Marston T. Bogart.

"Some Recent Advances in Physiological Chemistry," by John H. Long.

These addresses will be followed by selected papers of general interest.

2 p. m.—Meetings of the Section of Agriculture, Sanitary and Biological Chemistry, with John H. Long, as chairman; of Physical Chemistry, with W. R. Whitney, as chairman; and of Organic Chemistry, with Marston T. Bogart, as chairman.

4 p. m.—The members of the society will visit the Gratwick Laboratory.

8 p. m.—An address by Francis A. J. Fitzgerald, on "The Electrochemical Industries in Niagara Falls."

Friday, June 23, 9 a. m.—An address by Victor Lenher, on "Telleurium," to be followed by brief reports from colleges and universities of the researches carried on during the past year. The Section of Inorganic Chemistry, with Victor Lenher, as chairman, will then meet, and the Section of Industrial Chemistry, with F. A. J. Fitzgerald, as chairman.

2 p. m.—Excursions to visit industrial establishments in Buffalo.

8 p. m.—A subscription dinner at Hotel Iroquois.

Saturday, June 24, 9 a. m.—Brief adjourned sessions of the sections of a general session, according to the exigencies of the programme, to be followed by an excursion to Niagara Falls.

Society of Chemical Industry.

The general meeting of the Society of Chemical Industry, which will be held in London, beginning July 10, promises to become a most splendid and enjoyable affair. From reports received from England it is evident that the enthusiasm is running high on the other side of the Atlantic in respect of the welcome and reception of the Americans, and that no efforts are spared to make the coming meeting as memorable as that of last year in this country.

The Hotel Russell will be the headquarters of the American visitors during the week to be spent in London. The provisional programme for this week is as follows:

Monday, July 10.—The annual meeting, at which Dr. Wm. H. Nichols will deliver the presidential address, will be held at 12.00 or 12.30 p. m., and will, as usual, be preceded by the Council meeting. It will be held at the University College, London, by permission of the Council, and a light lunch will be provided by the University authorities. At 3.30 p. m. conveyances will be provided to drive members through Hyde Park and Richmond Park to Richmond, where Sir Max Waechter (of Baesler, Waechter & Co.) and Lady Waechter will give a garden party. The picture galleries of Sir Frederick Cook will be open for inspection. Drive back to hotel. At 9.00 or 9.30 p. m. there will be a reception with music, supper, etc., at the Botanical Gardens, Regent's Park, which will be illuminated for the occasion.

Tuesday, July 11.—Excursion down the river on one of the London County Council's new steamers, to the Royal Arsenal, admission to which has been obtained by Dr. Hodgkinson. (This is the first time that the Government authorities permit a party of foreigners to visit the Arsenal.) The party will be conducted around the works by the director of artillery. Thence by steamer to the Ship Hotel, Greenwich, where a fish lunch will be provided, after which Greenwich Hospital, including the Museum and Nelson relics, will be visited, and the party will be shown over the Royal Observatory by the astronomer royal. In the evening there will be a special reception by the Right Honorable the Lord Mayor of London at the Mansion House.

Wednesday, July 12.—Windsor and the neighborhood will be visited, with special permits to view the Castle. The party will afterwards drive to Virginia Water and the Mausoleum at Frogmore, and after lunch Eton School and the playing fields will be visited. At 7.00 p. m. the annual banquet will be held in the Goldsmith's Hall by permission of the chief warden and the Council of the Goldsmiths' Co. There will be dining accommodation for 170 only. Places will, of course, be found for the visitors, and if the English members swell the total beyond that number, they (the English members) will ballot for places. For those who do not obtain admission there will be an overflow dinner at the historic hall of the Merchant Tailors' Co., together with the ladies of the party. All will subsequently unite in a conversazione and musical evening in the reception rooms of the Goldsmiths' Hall, where all the historic and almost sacred plate will be on view.

Thursday, July 13.—Train to Bisley to see the National Shooting Competition. The party will then proceed to Haslemere, and a garden party will be given by Mr. and Mrs. Garton at Lythe Hill. The home of Tennyson, which adjoins the estate of Mr. and Mrs. Garton, will be visited, as well as the Punch Bowl and Hindhead, thus comprising some of the most beautiful of Surrey scenery.

Friday, July 14.—Parties to be arranged under suitable leaders to places of archæological and historic interest, such as Westminster Abbey, St. Paul's Cathedral, the Tower of London, the Royal Mint, etc., also Whitbread's Brewery, Price's Patent Candle Co., Clarke, Nickols & Coombs' confectionery works, City of London Electric Lighting Co.'s power station, and in the evening, opera or theater, as the case may be.

Saturday, July 15.—Messrs. Burroughs, Wellcome & Co.'s works at Dartford will be visited, after which a garden party will be given by the Wellcome Club. In the evening there will be a special dinner and reception at Earls Court Exhibition.

Sunday, July 16.—Train to Canterbury, service in the Cathedral; then train to Dover to visit the Castle, and lunch, and return to London.

Washington Meeting of the American Institute of Mining Engineers.

The thirty-fifth annual meeting of the American Institute of Mining Engineers was held in Washington, D. C., from May 2 to 4, under the presidency of Mr. James Gayley. Both with respect to the social functions and to the professional proceedings the meeting was a decided success. While the evening session of May 2 was attended by some seventy-five members and guests, the number of those participating on Thursday afternoon in the excursion to the Indian Head proving grounds was at least twice as large.

Following the usual custom of the Institute, only a small number of the papers on the programme (some seven out of more than forty) were read at the meeting. Compared with the custom, for instance, of the American Electrochemical Society, the policy of the Institute has one distinct advantage in that ample time is left for thorough discussion. As a matter of fact, the professional proceedings of the Washington meeting turned out particularly successful, by the lively discussions elicited by some of the papers.

Tuesday Session.

At the evening meeting on May 2, after the speeches of welcome and the reply by President Gayley, Dr. Chas. Kirchoff reported on the progress of the United Engineering Building, while the secretary, Dr. R. W. Raymond, spoke of the incorporation of the Society, which became necessary on account of its property rights in the Engineering Building.

Dr. Raymond then read biographical sketches of two highly distinguished members of the Institute, Sir Lowthian Bell and Dr. Thomas M. Drown, the biography of Sir Lowthian being written by Prof. H. M. Howe, that of Dr. Drown by Dr. Raymond.

Wednesday Session.

The features of the Wednesday session were two animated discussions; one on the subject of Mr. Gayley's famous dry-blast experiment, the other on wrought iron versus steel. Mr. Gayley gave some additional data on his dry-blast process, and replied to some of his critics. A large number of members participated in the discussion, among them Mr. J. E. Johnson, Jr., with quite an elaborate paper. (An abstract of a very interesting analysis of the dry-blast process by Dr. J. W. Richards may be found under Synopsis of Current Literature on another page of this issue.)

The discussion on wrought iron versus steel was started by a paper by Mr. Jas. P. Roe, on the production and characteristics of wrought iron. Mr. Roe, well known by his mechanical puddling process, discussed the reactions of the puddling process, the structure of puddled iron and its relation to the physical properties, the resistance to oxidation, and finally summed up the conditions essential to successful puddling as follows: Large scale manufacture, adequate mechanical means, cinder of proper composition, a flame of the right composition and temperature, a relatively permanent furnace lining, a relatively small loss of iron, and simplicity of means and method.

Mr. C. E. Stafford, Dr. C. B. Dudley, Dr. Cushman, Mr. J. Hartshorne and Mr. N. B. Wittman participated in the discussion. It was emphasized that while the advent of mild steel caused a considerable reduction in the number of puddling furnaces, now the opposite tendency is observed. Wrought iron is now preferred for many uses, especially when the resistance to fiber stress is of greater importance than high tensile strength. Wrought iron has now a better chance with large unit manufacture. Mr. Stafford spoke very favorably of the Roe puddling process, which, in his opinion, constitutes a great advance in economic metallurgy.

Thursday Session.

In the first paper of the Thursday session, Mr. Geo. T. Wickes described a machine for drawing coke from beehive ovens.

The other papers of this session were devoted to geological subjects. Mr. M. R. Campbell presented a paper on "A Classification of Coals and Lignites," based upon ultimate analysis. Prof. James F. Kemp discussed "The Copper Deposits of San Jose, Tamanlipas, Mexico," and Mr. A. C. Spencer the "Origin of the Magmatic Vein Waters in Alaska."

A good deal of the success of the meeting was due to the excellent preparations made by the local committee, with Mr. S. F. Emmons, of the United States Geological Survey, as chairman, and Mr. John M. Boutwell, also of the Geological Survey, as secretary.

Notes.

American Electrochemical Society.—At the May meeting of the board of directors the following gentlemen were elected members of the Society: W. H. Kimball, Portland, Me.; A. S. Ramage, Detroit, Mich.; J. M. Woodward, Washington, D. C.; C. A. Hansen, Madison, Wis.; J. W. Beckman, Tarentum, Pa. At the next meeting of the board of directors the names of the following gentlemen will come up for election: Henry F. Lewis, Boston, Mass.; Edgar F. Smith, University of Pennsylvania; Theodore W. Richards, Harvard University; Hubert Buckley, Niagara Falls, N. Y.

New York Section American Electrochemical Society.—The last meeting of the session was held on the evening of May 23 in the hall of the Chemists' Club. There were three papers, all of which were listened to with great interest. Prof. Turner, of Columbia University, described the equipment of the electrochemical engineering laboratory of Columbia University, and gave a review of some of the work done there during the last year. Some of the specially interesting lines of work were the production of boron carbide and the manufacture of ferro-alloys. With respect to the latter Messrs. G. W. Maynard and J. T. Morehead made some remarks in the discussion. Among other things Mr. Morehead stated that the monthly output of ferro-chromium by the Wilson Aluminium Co. is now 200 tons. The second paper dealt with the method of manufacturing alundum (artificial emery) in the electric furnace, the author being Mr. C. B. Jacobs, the inventor of the process which is in use at the Niagara works of the Norton Emery Wheel Co. (our Vol. I, page 15). Replying to some questions by Mr. H. N. Potter in the discussion, Mr. Jacobs gave a comparison of the uses of alundum and of carborundum, respectively. The third paper was presented by Mr. Maximilian Toch, on insulating paints and varnishes. He explained the problem and related very interesting observations he had made on the iron structure of the Sixth Avenue Elevated Railway in New York.

Lehigh University.—Among the subjects of theses to be presented by candidates for degrees at Lehigh University in June, are the following: C. H. Young on history, methods of manufacture and trade conditions of acetic acid (bachelor of arts); R. G. Kirk and F. C. Ryan on the briquetting of iron ores with cement (metallurgical engineer); L. F. Blume and S. H. Fleming on the polarization factor of electrolytic cells, and E. L. Rich and R. H. Smith on over-voltages in electrolysis (electrical engineer). The number of candidates for the degree of master of science is 1; bachelor of arts, 7; civil engineer, 31; mechanical engineer, 31; metallurgical engineer, 2; engineer of mines, 6; electrical engineer, 16; analytical chemist, 3; total, 97.

Electroplated Laces.—According to a recent consular report, the production of electroplated laces is another new industry in Continental Europe. Experiments were tried long ago to make non-conductive articles susceptible to the plating process, either dry or wet, by sprinkling them with a very fine powder of metal or graphite, or by immersing them in a solution of metallic salts. In this manner many articles, such as flowers, leaves, fruits and branches, and even delicate articles, received a metallic coating. An Italian engineer used the process to metallize parts of corpses to serve for medical demonstration, while a New York firm makes a specialty of electroplating first-baby shoes. A Brussels chemist devoted his attention especially to metallizing embroideries, and created wonderful effects. Until quite recently, however, they could not be made of practical value, and "only now a beginning has been made by an invention, the details of which are kept a secret. A stock company has been formed for its exploitation. According to the Paris *Cosmos* the company has created a sensation with their product." Laces are mainly treated. The colors of the metal coating and other properties can be regulated ad libitum. These metallic goods are used for table ornaments, decorating furniture coverings, wainscoting in parlors, and for incrustation of fine woods, and the metallized laces may be polished. Fashion will try what effect such gold or silver laces will have when used as trimmings for dresses or shawls or in the hair. Two processes of this kind were de-compiled from United States patent specifications on page 67 of of our Vol. I, and on page 34 of our Vol. II.

Progress in Iron and Steel Metallurgy.—The recent presidential address to the Iron and Steel Institute by the famous British steel expert, Mr. R. A. Hadfield, was devoted to a review of the progress in iron and steel metallurgy. In this address Mr. Hadfield made some remarks on the use of electric furnace, but it seems that, unfortunately, he discussed only one and the least promising problem, *i. e.*, the reduction of pig-iron from ores. He concludes that "blast furnace owners can breathe freely" until the day comes when electrical energy can be produced more cheaply. Mr. Hadfield paid a special tribute to "the early workers in this important branch of scientific manufacture, the Brothers Cowles, Waldo and Hunt, who fought their way to success through difficulties innumerable." Mr. Hadfield also discussed advances made in the manufacture of ordinary steel; the increasing uses of alloy-steels; the enormous importance of employing just exactly the right temperature in the heat treatment of steel; progress in metallography and low-temperature experiments; the industry of steel castings, and the manufacture of armor plates and projectiles.

CORRESPONDENCE.

Detinning.

To the Editor of Electrochemical and Metallurgical Industry:

Sir:—Permit me to call your attention to an error in "Analysis of Current Electrochemical Patents" in your May number. In the notice of "Process of Detinning," patented by F. von Kuegelen and G. O. Seward, Dr. Scholl states that this patent was assigned to the Union Carbide Co. The patent was, in fact, assigned to the Willson Aluminium Co.

Yours very truly,
Willson Aluminium Co.,
By George O. Seward

Holcomb Rock, Va.

Refractory Materials for Metallurgical Furnaces.

To the Editor of Electrochemical and Metallurgical Industry:

Sir:—In the design of any furnace, the character of the lining and of the bricks that back the lining and form the arch and walls is of paramount importance. Materials that withstand the corrosive action of the molten slag or metal, and at the same time are not affected by high heat, usually have small tensile strengths. It is therefore always necessary to make use of their compressive strength, and let the proper iron work outside carry the tensile strains of the structure.

This chain of facts permits of the use of large furnaces for cold iron is exceedingly strong. Where reliance is put on the tensile strength of fire-clay we have the small "graphite" or fire-clay crucible retort, or muffle used in the crucible furnace, the zinc furnace and annealing furnace respectively. Cast iron can only be used where the temperature does not much exceed 700° C.

Unfortunately, the knowledge of this important subject of refractories is scattered, illusive and indefinite. The makers of fire-brick have some valuable practical data due to this contact of the whole field, and the metallurgists in the special fields also know much that is useful and applicable. This, of course, must be true, for furnaces do make metal, and last for long periods, sometimes for years.

But this data which is placed before the metallurgist is unsatisfactory and unreliable. How unreliable and unsatisfactory is shown by the fact that the best blast furnace engineers,—men who, for mental ability, stand second to no other class in this country,—cannot accurately say, when blowing in a new iron blast furnace, whether it will operate without relining for three months or twice that many years. Indeed, we were talking with two metallurgical engineers, when A said to B, "What do you know about the heat conductivity of fire-brick?" B replied, "I know this much, that it is a very important subject, and I would like to know a good deal about it." This is a lamented state of affairs for a profession that should base its knowledge on scientific facts. In the future it will not be enough to say put "two courses there" and "three here." We must know how many calories of heat for a given brick will pass through a cubic inch at definite temperature difference.

In electric furnaces, especially those which are permanent, refractories are an essential. It is obvious that an electric furnace has an advantage in putting the heat right in or next the charge. That the efficiency with which this heat is applied overbalances in some instances its cost as compared with heat derived from carbonaceous fuel. Consequently the materials which conceive this heat must be refractory, durable, and possess high heat insulating properties. In a copper matting furnace, the entire lower part is usually water jacketed and the lining is congealed slag. This allows of long campaigns. But an electric furnace, because of the high price of electric heat, cannot well be water-jacketed. Indeed, it is usually out of the question to water-jacket an electric furnace for electrical reasons alone. If water jackets are used in an electric furnace, they should only protect special points, as electrodes or tap holes.

Another most important branch is the class of refractories that transfer the heat from the fuel to the charge. In usual practice, strange as it seems, these are of fire-clay, the same that is used in keeping heat from going from the furnace to the outside. For two diametrically opposite purposes the same material is used. This is incredible, and a sufficient commentary on the backward state of our knowledge of refractories. In all crucible melting, in muffle roasting, in a by-product coke oven, in much chemical work, in many annealing furnaces, the heat must be forced through a poor conductor. Of course, chrome brick conduct heat much better than ordinary fire-clay brick, but their cost is $170 or more per thousand. If a refactory brick would be made of magnetite or other conducting material and sold at a price equal to the best grades of fire-brick, or even as high as $60 per thousand, it would find a warm welcome from the metallurgical profession. If such a brick "were true to size" and could be easily cut by the mason, it would be even more acceptable. But if such a brick could also be made in special shapes cheaply, if it could stand temperature changes without cracking, if it were non-porous, if it could stand high temperatures and fluing action of various materials, it would be the *ne plus ultra* for all purposes of indirect heating.

We have but touched on a few points in the résumé of this important subject. We hope that this will be investigated in the future most thoroughly, and that the subject will be attacked from the scientific and practical standpoint. Indeed, we confess ourselves optimists for the future of metallurgy and development in the line of refactories will give, let us hope, in the future, fair play for our optimism. For time always satisfies long-felt wants.

In conclusion, I might refer in this connection to the important serial on metallurgical calculations which Prof. Richards is now publishing in your journal—a series which will do much to simplify metallurgical knowledge of the country. If data for the conductivity of different makes of fire-clay brick, etc., were known, we could calculate loss by radiation, now called "unaccounted for," and thus check up the "heat balance sheet." This would make a surer and better method of determining the loss by radiation which should be known definitely.

Woolsey McA. Johnson.

New York City

The Borchers Furnace.

By F. A. J. FitzGerald.

In the year 1849, Despretz presented to the Académie des Sciences, in Paris, certain papers that have considerable interest in connection with electric furnace work, and are by no means as well known as they ought to be. Despretz described how he had fused and volatilized various refractory substances, such as magnesia, silicon, boron, platinum, etc. Finally he read papers specially devoted to certain experiments, which, in his opinion, clearly demonstrated the fusion and volatilization of carbon. Moissan has since shown[1] that carbon is infusible at ordinary pressures, even when raised to the highest attainable temperature; but that does not lessen the interest in Despretz's paper, which contains much that is important.

The body of the furnace used by Despretz was made of cast iron, and had a movable cover. The latter carried a leather stuffing-box, through which a rod passed vertically, the rod being insulated by two glass plates and two small leather washers. At the end of the rod was a carbon terminal, the other carbon terminal being connected to another rod passing through a suitable tube in the side of the apparatus. The interior of the furnace body could be observed through two large tubes closed with thick plates of glass. A fifth tube could be connected either to a vacuum or compression pump, while another tube permitted connection with a manometer.

With this apparatus Despretz was able to make experiments, using either carbon resistors or arcs. When he wished to develop heat by means of a resistor he connected the carbon terminals carried by the vertical and horizontal rods with a small rod of carbon, through which the current was passed. When an arc was required the furnace was fitted with a different cover, so arranged that by means of a third rod the substance under examination could be introduced into the arc formed between two carbon rods.

The apparatus described above had a capacity of 10 litres, and was only used in experiments where it was desired to work under increased pressure. In experiments under atmospheric pressure the electric furnace was covered with a large bell-jar.

Despretz's source of current was 600 Bunsen cells, which could be connected in various ways, according to the nature of the experiment.

Several experiments were made with carbon rods through which the current was passed, and Despretz observed that when these rods became very hot they tended to bend, the carbon apparently becoming soft. Usually the experiments did not last long, for the carbon rods would presently break. Various kinds of carbon were used: retort carbon, sugar carbon, anthracite, etc. When the carbons broke, Despretz observed that the broken ends usually presented a fused appearance, and were converted into graphite. Although Despretz made analyses of the carbons, which seemed to show that they were very pure, it is probable that the impurities present formed carbides and gave the appearance of fusion.

Despretz notes an inconvenience in his experiments which has been experienced by many experimenters since his time. He says:

"Thus, if it is desired to demonstrate the volatilization of platinum in air or that of iron in a vacuum or in nitrogen, carbon is found mixed with the metal in the porcelain dish even when this is placed 1 decimeter above the carbon crucible in which is the substance submitted to the action of the electric arc."

He notes in various experiments the formation of graphite and a curious increase in the volume of the carbon under treatment.

"A rod of retort carbon 15 mm in diameter and 25 centimeters long, connected with the two terminals, closed the circuit of the battery of 600 cells connected in six parallel series.

"The pressure of nitrogen (used in the furnace) was two and a half atmospheres. The carbon was raised to a dazzling white heat, the substance seemed to collect at the lower part, and it broke at this point. The two separated parts had double their original diameter.

"This increase in volume by the action of heat alone could scarcely occur unless the substance had begun to fuse."

It is now known that certain forms of carbon increase greatly in apparent volume when converted into graphite.

An interesting experiment was made with a carbon rod which passed through a clay crucible filled with sand. The current was passed through the rod which "fused and volatilized. I obtained a sort of very hard fulgurite (lightning-tube) the interior of which was lined with smoky-quartz; the inner diameter of this tube was at least ten times that of the carbon."

Mr. Elihu Thomson has recently obtained a patent for making quartz tubes in this manner.[2]

Despretz also made several experiments on the effect of heat on diamond. For this purpose he used a tube furnace which he made by connecting the carbon terminals with a carbon tube in which the diamonds were placed. Despretz' paper may be found in *Comptes Rendus* No. 25, T XXIX, 1849.

This furnace, designed by Despretz, may be looked upon as the progenitor of the furnace known as the Borchers type. An illustration of the Borchers furnace is given in Fig. 1. T1 and T2 are heavy carbon terminals which are connected electrically by the resistor R. The connections between the resistor and the terminals may be made by drilling holes in the terminals and packing with graphite, G1 and G2, as shown in the figure. The connections to the source of current are made at H1 and H2. The material to be heated surrounds the resistor R, and the furnace is covered by the bricks B. According to Borchers, all oxides may be reduced in this furnace.

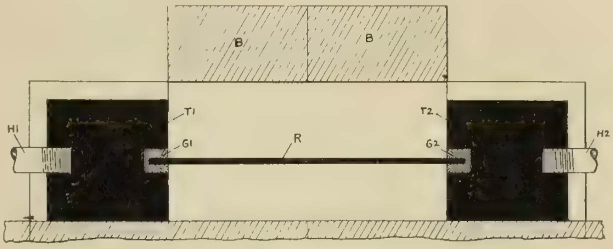

FIG. 1.—DIAGRAM OF BORCHERS FURNACE.

The data given by Borchers for this furnace are as follows:

"An electromotive force of 0.3 to 0.4-volt is necessary to drive a current of 1 ampere through 1 mm. of a carbon pencil at the temperatures of these experiments, with a current density of 6 to 10 amperes per sq. mm. of sectional area."[3]

This would give a resistivity of 0.005 to 0.004 ohm, which is somewhat higher than required in the case of graphite rods, as will be shown later. It should also be observed that a current density of 6 to 10 amperes per square mm. of sectional area in carbons, having the resistivity of those mentioned, would give an enormously high temperature unless the diameter of the carbon was very small.

In designing a Borchers furnace the general principles that have been considered in dealing theoretically with the resistance furnace must be kept in mind. It is not necessary to make very close calculations of dimensions because of the unknown factors which enter into the actual working of the furnace; but time is saved and results more quickly obtained by making approximations to the theoretical conditions.

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[1] "Le Four Electrique," Paris, 1877, page 160 et seq.

[2] U. S. patent 778,286, December 27, 1904. Electrochemical and Metallurgical Industry, vol. III., page 71.

[3] "Electric Smelting and Refining," by W. Borchers, London, 1897, p. 111.

A furnace actually built at the FitzGerald and Bennie Laboratories for certain experiments is shown in elevation, partly sectional, in Fig. 2, and a plan of the same furnace in

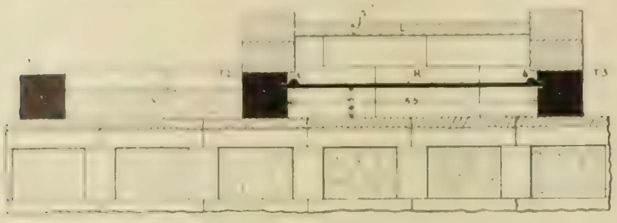


FIG. 2.—SECTION OF FURNACE.

Fig. 3. In the plan the covering tiles and bricks are removed. All dimensions are given in centimeters. T1 is the terminal of the granular carbon regulating resistor V, T2 the terminal common to the regulating resistor and the furnace, T3 the other terminal of the furnace. R is the furnace resistor, *a* and *b* two carbons which pass through the side wall of the furnace, and can be connected to a voltmeter. The furnace is built on a fire-brick base of cellular structure, and the cells filled with a suitable heat insulator. The top of the furnace is covered with the tile L, and this in turn should be covered with a heat insulator. As shown in Fig. 2, R is connected to the terminals T2 and T3 by introducing its ends into the holes of larger diameter in the terminals and packing the space between the rod and the inner surface of the holes with graphite. This method of connection is found to give very satisfactory results, as it takes care of the contraction and expansion of the rod accompanying variations in temperature.

The voltmeter carbons *a* and *b* have a circular cross section, and the ends resting on R are filed to a concave surface corresponding to the curvature of R. The total length of R is about 60 cms., the effective part, that is the part between *a* and *b*, is about 52 cms. long.

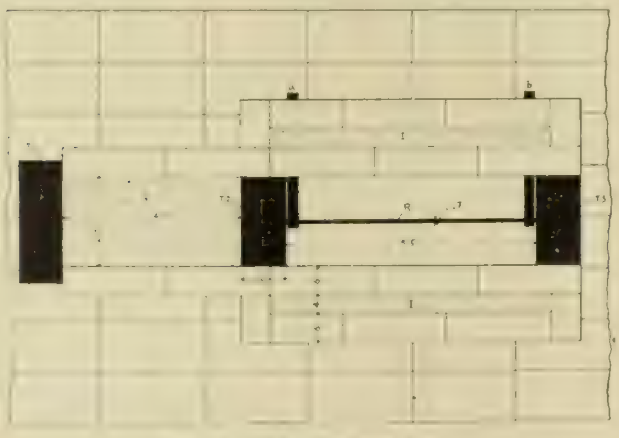


FIG. 3.—PLAN OF FURNACE.

The figures show the furnace as it was actually built; but for permanent work, or where it was desired to build as efficient an apparatus as possible, certain changes would be made. Thus the voltmeter carbons *a* and *b* ought to be of smaller diameter than the resistor, for with the large sized carbons actually used there is much loss of heat by conduction. The terminals also should be much smaller than those shown, and should be made of amorphous carbon, in order to avoid unnecessary loss of heat. It will be seen later that the loss of heat at the ends of the resistor was very great.

In the furnace, the resistor R was made of graphite, so that its resistivity may be assumed to be 0.00076 ohm. Then the [illegible] a and b would be

$$0.00076 \frac{55}{1.27} = 0.033 \text{ ohm}$$

Actually the minimum resistance observed was 0.0325 ohm, but as the distance between *a* and *b* was only roughly measured, this is very close to the calculated quantity.

The furnace was constructed originally for the purpose of studying the behavior of this type, and was used for making a silica tube. After it was built, as shown in the figures, it was filled with pure quartz sand, which completely surrounded R. Two runs were made on consecutive days, the furnace thus having plenty of time to cool between the first and second runs. The power curves of the two runs are shown in Fig. 4.

Taking the two curves together, the mean ordinate is found to be 4.2 kilowatts, therefore the total power used was 5.0 kilowatt hours. Taking the mean value of the energy developed in the resistor, it is found that the average watts per square decimeter of resistor surface were 1.9 kilowatts. The maximum load was 5.9 kilowatts, that is, approximately 2.7 kilowatts per square decimeter.

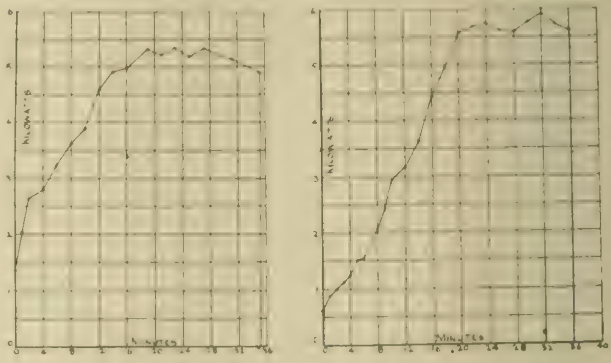


FIG. 4.—POWER CURVES.

After the furnace had cooled, following the second run, the resistor was removed from the furnace and was found to be surrounded with a fused quartz tube as shown in Fig. 5. In Fig. 6 is a sectional view of the quartz tube and the resistor. The maximum exterior diameter of the quartz tube was 6 cms., and its length 52 cms. It weighed 1.54 kilograms. It will be observed in Fig. 5, that the ends of the tube taper off, showing the cooling effect of the terminals and the voltmeter rods. Fig. 6 shows how the fused tube has shrunk away from the resistor. By measurement it is found that the average diameter of the interior of the tube is 2 cms.

The temperature to which the tube was heated must have been very high, for taking the inside diameter of the tube it is found that with the maximum kilowatts of 5.9, there were 1.8 kilowatts per square decimeter of the inside surface of the tube. This must have produced a very high temperature in the interior of the tube, and judging by its appearance the silica must have been raised to its boiling point. Through an accidental occurrence an approximation to the temperature of the tube was obtained. Some small pieces of coke dropped into the sand used in the experiment, and one piece was found in a small cavity, near the outer surface of the quartz cylinder, about 2.5 cms. from its axis. Here, when the maximum kilowatts were developed, there were only 690 watts per square decimeter, yet the grain of coke was covered with crystals of silicon carbide.

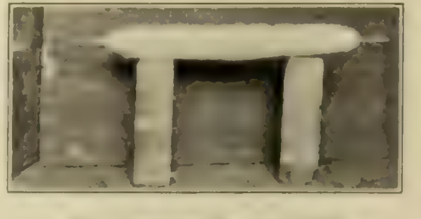


FIG. 5.—FUSED QUARTZ TUBE.

Whatever was the actual temperature in the tube, it certainly was very high, and therefore there was a very rapid vaporization of the silica, and the vapor produced reacted with the carbon of the resistor, forming silicon carbide. Careful measurement showed that this did not appreciably alter the diameter of the resistor, the silicon carbide apparently taking the place of the carbon on the outside of the resistor. A fair sized piece of the resistor was heated in oxygen to remove all free carbon, and the residue was found to consist of silicon carbide. The carbide at the outer part of the resistor was left in the form of a solid ring (Fig. 7) or cylinder with an average thickness of 1mm., while the carbide from the interior of the resistor was in the form of powder. The ash in the carbon amounted to 7.3 per cent, while in the original resistor it was only 0.099 per cent.

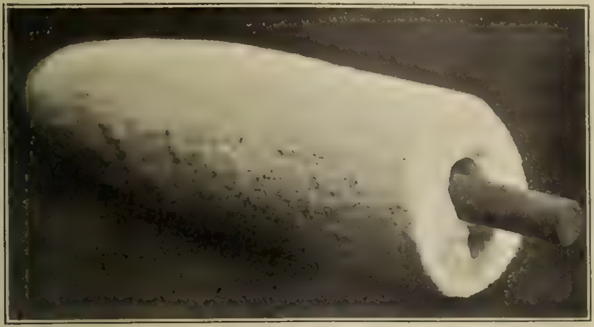


FIG. 6.—QUARTZ TUBE AND RESISTOR.

The approximate amount of silica vaporized from the interior of the tube and combined in the carbon as silicon carbide may be calculated from the figures given above. The total weight of the resistor affected amounted to 108 grams, and from the analysis it is found that the silicon carbide present amounts to 7.88 grams, corresponding to 11.8 grams of silica. This phenomenon of the vaporization of silica and formation of silicon carbide is important, since it has a marked effect on the resistance of the resistor. Thus, at the end of the first run, the resistance of R was 0.037 ohm, and at the end of the second run was 0.041 ohm.

As was pointed out before, the furnace described above must not be taken as a model of design, for it was not built with that end in view. The shape of the silica tube showed the serious heat losses caused by faulty design. The illustration must therefore be merely considered as giving some notion of the working of a Borchers furnace.

It will be readily seen that a serious objection to a furnace built with a solid resistor as described above is the low resistance of the solid carbon resistor, which involves the use of heavy currents of low voltage if large amounts of power are required. This objection may be illustrated by some simple calculations.

Let it be assumed that we have at our disposal a 5-kilowatt transformer giving a current of 200 amperes at 25 volts. Then if it is desired to use 5 kilowatts in a furnace of the kind described above, the resistance of the resistor must be

$$\frac{25}{200} = 0.125 \text{ ohm}$$

If the resistor has a working resistivity of 0.00076 ohm, then we have

$$0.00076\,\frac{l}{A} = 0.125 \qquad (1)$$

where l is the length and A the sectional area of the resistor, and from (1) we find that the ratio between the length and sectional area is

$$\frac{l}{A} = 164 \qquad (2)$$

This plainly would necessitate very poor furnace design, as the length of the furnace would be enormously greater than its width or depth, and it has been shown elsewhere, that the nearer a furnace of rectangular shape approaches a cube in its dimensions the greater the efficiency, because in this way the radiation of heat from the outside of the furnace is reduced to a minimum.

But in addition to this question of design, there are other considerations which are fatal to the construction of the furnace if we attempt to use a graphite rod as resistor. Let it be assumed that to carry out the furnace work, we require 2.5 kilowatts per square decimeter of resistor surface. Then if the length of the resistor is l, as before, and the radius of the resistor is r, we have the following equations to determine its dimensions. To fulfill the requirements as to resistance equation (2) becomes

$$\frac{l}{\pi r^2} = 164 \qquad (3)$$

and the equation for the surface is $a =$

$$\frac{5000 \times 100}{2\pi r l} = 2500 \qquad (4)$$

From (3) and (4) we find

$$l = 80.5 \text{ cms.}$$
$$2r = 0.79 \text{ cms.}$$

While carbon rods of these dimensions might be obtained, they would be found very unsatisfactory, for they would be too delicate to handle without breaking.

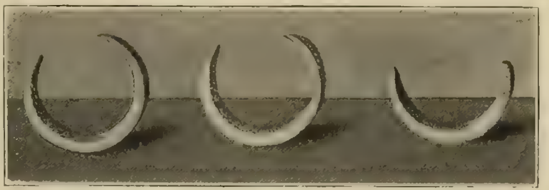


FIG. 7.—OUTER PART OF RESISTOR.

A closer approach to a practicable resistor might be made by using one having a rectangular cross-section, for this shape would give a greater heating surface for the electrical resistance of a given length. Assume, for example, that we decide to use a plate 0.3 cm. thick. Then if the width of the plate is b, equations (3) and (4) may be written as follows:

$$\frac{l}{0.3b} = 164 \qquad (4)$$

$$\frac{5000 \times 100}{2(0.3 + b)l} = 2500 \qquad (5)$$

From (4) and (5) we get

$$l = 62.5 \text{ cms.}$$
$$b = 1.27 \text{ cms.}$$

This is somewhat better, but the plate would be a very delicate article to handle, and even if successfully put in place in the furnace, would probably be broken while in use.

Some examples of improved forms of resistors are shown in Fig. 8. These may easily be made from graphitized articles with the help of a hack-saw. The forms shown in the figure have been made and used in the laboratory for various purposes and were found to give satisfactory results. It should be noted here that in connecting the resistors having a rectangular cross-section to the terminals a good method is to cut slots in the latter, the dimensions of the slots being greater than those of the section of the resistor. The ends of the resistor are then put in the slots, and the empty spaces tightly packed with graphite.

The advantage of using resistors made on these principles is obvious, overcoming as they do the difficulties considered above. Take for example the middle resistor shown in Fig. 8. Calculating the ratio between length of the resistor as regards its electrical resistance and the length in a straight line, we find it to be 2.4; that is, if the distance between terminals is 10 cms., the effective length of the resistor so far as electrical resistance is concerned, is 24 cms. In this way the difficulty of impracticable length needed to obtain a given rate of energy-expenditure with a given heating surface is avoided.

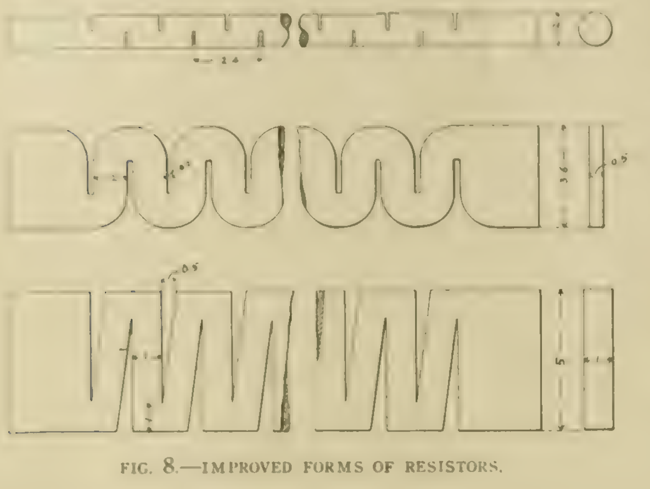


FIG. 8.—IMPROVED FORMS OF RESISTORS.

For example, assuming the same conditions as before: 5 kilowatts, with 25 volts and 200 amperes, we have

$$0.00076 \frac{l}{0.5 \times 1.2} = \frac{25}{200}$$

$$l = 99 \text{ cms.}$$

Denoting the distance between terminals by L, we have

$$L = \frac{99}{2.4}$$

$$= 41 \text{ cms.}$$

Finally the watts per square decimeter of heating surface would be approximately 2,500 as required.

In cases where it is desired to work at a lower resistor temperature, using less watts per square decimeter, another form of resistor shown in Fig. 9 may be advantageously used. This is a cylinder which will take approximately 400 amperes with 10 volts, and treating the carbon as though it were not cut, as shown, will give about 1,000 watts per square decimeter. Another obvious method of treating hollow cylindrical resistors so as to increase the resistance per unit of length is to cut a helical slot in the cylinder, thus forming a helical resistor.

Sometimes it may be desirable to introduce gases into a furnace while it is working, and for this purpose a resistor of the form shown in Fig. 9 is suitable. In making connection to one of the terminals by the method already described, the hole drilled in the terminal is carried all the way through and suitable connection made to the gas tank or generator.

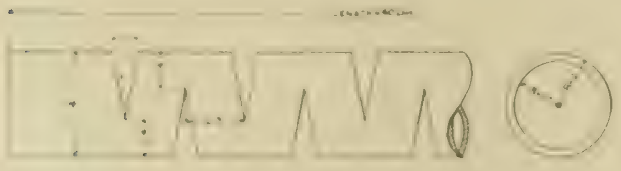


FIG. 9.—RESISTOR.

In using resistors of the kinds described above, it may be objectionable to have the furnace charge get into the spaces between the sections of the resistor. In that case the spaces may be filled with agglomerated charcoal powder, which is a non-conductor, compared with the resistor material. Before using the charcoal for this purpose it should be ignited at a very high temperature to remove all shrinkage. A satisfactory agglomerating material is a strong solution of sugar. It is a good plan to use this charcoal in any case where, for example, the resistor is like the lowest one shown in Fig. 8. Unless this resistor is handled carefully it is likely to break, but if the V-shaped spaces are filled with the agglomerated charcoal, it makes a pretty strong article.

Sometimes in making experiments with a Borchers furnace it may be important to avoid contact between the carbon resistor and the charge. Thus in reducing the oxides of some metals the metal when set free may attack the carbon resistor if in contact with it. In this case the charge may be heated by radiation from the resistor. A furnace used for this purpose is shown in Fig. 10. The resistor used here is of the third form shown in Fig. 8. With a distance of 50 cms. between terminals, this resistor would take a current of about 260 amperes with 40 volts. In the Figs. T2 and T3 are the terminals of the furnace; R, the resistor; M, the charge which is insulated, form the terminals by the granular charcoal B. The resistor is covered with a slab of charcoal C, and above this should be placed a good heat insulator, not shown in the drawing. The lower drawing in the figure shows a transverse section of the furnace. The V-shaped spaces in the resistor should be filled with agglomerated charcoal to strengthen the carbon, which otherwise would certainly get broken by its own weight.

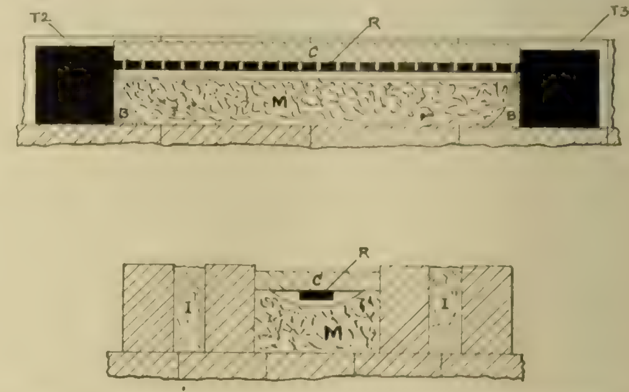


FIG. 10.—FURNACE WITH RADIATION OF HEAT FROM RESISTOR.

Electrometallurgy of Iron and Steel before the Faraday Society.

In our December issue, 1904, we covered very fully the important report of the Canadian Commission which studied European industrial developments in the electrometallurgy of iron and steel. Some new interesting information is given by Mr. F. W. Harbord, who was the metallurgical expert of that Commission, in a recent paper presented before the Faraday Society in London. In the following we give those portions of Mr. Harbord's paper which contain new information:

The author first described the Kjellin induction furnace (our Vol. I, pages 70, 141, 283, 376, 462, 526, 576; Vol. II, page 479) and mentioned that Mr. Kjellin has sent him particulars of recent developments in a letter dated January 7, 1905, which is as follows: "Having at last got a new turbine, we are able to increase the power at the furnace to 175 kilowatts (239 electric hp) instead of the maximum being 165, and this increase of power, together with a diminished area of contact between steel, masonry and slag, has increased the output of the furnace from 4 tons to from 5.2 to 5.5 tons in twenty-four hours, which naturally makes the melting costs lower. Another reason for these good results is that, owing to some slight alterations, I have got the power applied more uniformly over the whole time of the charge. In future I shall be able to build the furnaces so that they shall consume the maximum power of the turbine or engine all the time. I have already done so in the case of small trial-furnace, put up last summer in the south of France. The capacity of the Gysinge furnace with 175 kilowatts is now 1450 kilograms, of which 900 kilograms are tapped each time. I have recently used pig and ore instead of pig and scrap, and this seems to answer very well. The ore used is briquettes from Herrang concentrates, containing about 70 per cent iron, and consequently practically no gangue, so that the wear and tear is not increased to such an extent as would be the case if the ore contained much gangue. The briquettes being very low in phosphorus, I can in this way get a high quality steel cheaper than by using scrap and pig, though the production of the furnace is only 60 to 65 per cent of the production from cold scrap and pig. If using molten pig and ore I should get the same output from a given power as from cold pig and scrap, but the waste would be much lower, as the iron of the ore would be almost perfectly reduced."

It will be seen from the above that pig and ore can be used in the furnace, but to obtain a high-class product, none but extremely pure materials can be employed, and at present Swedish materials are probably the only suitable ones which are available. The furnace in its present form can only be used for the manufacture of tool and other high-class steels, and in the majority of cases, at all events outside Sweden, it will probably pay better to use pig and scrap only, as the saving effected by ore and pig, when the reduced output is considered, cannot be very great.

The author then discussed the Héroult process (our Vol. I, page 63, 287, 449, 461, 467; Vol. II, page 408, 481), and makes a comparison between the processes of Kjellin process and the Héroult process. He thinks that as they are now being worked, there is very little difference between them in the cost of operation. The chief point, however, when one process has an advantage over the other, is in the cost of raw material. The Kjellin process, to produce the highest class tool steel, is limited to the best Swedish scrap and the best Swedish pig, or pig and ore, whereas common miscellaneous scrap is the raw material used in the Héroult process. In Sweden this high-class scrap can be obtained at a comparatively low figure, but in other countries, Walloon scrap, if obtainable at all, which is not likely, would cost probably $60 per ton, and possibly more, and it would most likely be necessary to buy the Walloon bars at the same price as bars for cementation. On the other hand, ordinary steel scrap in England can be bought from $12.50 to $15 per ton, so that in countries outside of Sweden the Héroult process has a great advantage. It is quite possible that Mr. Kjellin may be able to modify his furnace, so that he can renew his slag, and thus effect purification in the same way as in the Héroult process, but in its present form it is limited to the purest material either as scrap or scrap pig ore. His latest results with pure ore have no doubt reduced the costs, but the Héroult process still has a great advantage as regards the raw materials.

In the present stage of development, neither type of furnace can be regarded as a competitor to either the Siemens or the Bessemer processes for the production of rail and structural steel, and they can only compete successfully in the production of high-class crucible steel or steels for ordinance and other special purposes made in the Siemens furnace. In cases where very large steel castings are required of crucible steel quality, several electric furnaces, working so that they could be tapped into a common receptacle, before pouring the steel into the mold, should give excellent results and be much more economical than the crucible process. Under favorable conditions electric energy might compete with gas as regards cost, but until it is possible to use furnaces of from 30 to 40 tons capacity, the extra labor charges inseparable from small furnaces will prevent them from holding their own against the Siemens or Bessemer process.

"So far as the steel works engineer is concerned, there is no difficulty in designing a furnace of any size required, as furnaces, very similar in general design to the Héroult furnace, holding 200 tons of molten steel, are at present working both in England and America, and it only remains for the electrical engineer to arrange for the electrical heating without unduly weakening the roof or other parts of the furnace. Provided this can be done, under favorable conditions, where electric energy is so cheap that it can compete with producer gas as a heat producer, there is not the least reason why the present 5-ton electric furnace should not develop into a 40 or 50 ton furnace, in the same way that the 5-ton ordinary Siemens furnace has developed during the last twenty-five years."

Mr. Harbord then gave a review of the experiments made at Livet on the reduction of pig iron from the ore (our Vol. II, page 483) in the electric furnace. He mentioned briefly the manufacture of ferro-alloys at Livet, as follows: The furnaces used were identical with those employed in the pig-iron experiments, and the only difference was in the alteration of the furnace mixture, and in the regulation of the electric current to obtain the necessary heat required for the particular operation. Ferro-chromes from about 45 per cent to 60 per cent, and ferro-silicons from 25 per cent to 80 per cent, were being made regularly, although 50 per cent ferro-silicon was the alloy that they were making most largely at this particular time. In the manufacture of this alloy, a very intense temperature is necessary to obtain a good tapping heat, and it is desirable to work with as little flux as possible, so that a very small quantity of slag is produced. The alloy is tapped into horizontal molds holding about 2 cwt., and the little slag formed is raked off the surface of the alloy before it solidifies. In the manufacture of these alloys, oxide of iron is not used as the source of iron, but selected steel scrap. In the case of chrome-alloys it is desirable to use rich chrome iron ore. The demand of these alloys undoubtedly increasing, many steel foundries using nothing below 25 per cent to 30 per cent ferro-silicon, and some nothing below 50 per cent. The increasing demand for chrome and tungsten alloys for the manufacture of the various special steels required for ordinance and other purposes is too well known to need comment.

Up to about 12 per cent to 15 per cent ferro-silicons, and 40 per cent ferro-chrome, the blast furnace is probably a more economical metallurgical appliance than the electric furnace, but for higher alloys the latter holds the field.

Mr. Harbord finally passed over to the Stassano arc furnace (our Vol I, page 247) and quoted from a letter of Stassano dated January 15, 1905, the following particulars of recent developments in the Stassano furnace now employed for scrap melting and purifying at the government works at Turin. The furnace is almost entirely charged with scrap or a mixture of scrap and pig-iron, so that the original intention of Stassano to make steel from iron ore in one process has been abandoned. The letter of Stassano reads as follows:

"The furnace at Turin has been working regularly since March last, producing normally the grade of steel required by the government for the manufacture of artillery projectiles. The furnace is standing at present, in consequence of the transformer having been damaged, but a new one is being installed which will reduce the current from 3000 to 80 volts; the furnace is expected to be ready to commence work again in March.

"The usual charge consists of 350 to 400 kgs. of pig iron, 200 to 250 kgs. of scrap, the necessary amount of ore being added to oxidize the impurities, and sufficient lime to form a basic slag to protect the lining of the furnace and remove the phosphorus and sulphur in the material employed. During the last year's working the following data have been obtained: Loss on weight of materials charged 1.5 to 2.01 as a maximum; consumption of electrodes, 5 kg. per ton of steel produced; mean consumption of current 1.2 kw. per kg. of steel produced, equal to 186 ehp year per ton of steel. Six men—an electrical mechanic, melter and four laborers—are more than sufficient for one furnace."

* * * * * * *

Concerning the presentation and discussion of this paper which had attracted the largest attendance of members and visitors which have been present at any meeting of the Faraday Society, our special London correspondent writes as follows: The paper was not read. Instead, Mr. Harbord lectured on the subject with the illustration of some scarcely distinguishable lantern views. Some lantern slides showing views of the works at La Praz, sent by M. Adolphe Minet, could not be shown to the meeting by reason of the fact that lantern slide dimensions are not internationally standardized. Some samples of Héroult steel, and some pieces of ferro-silicon, ferro-chrome and ferro-tungsten, all sent by M. Minet, evoked much interest.

The discussion was opened by Mr. Morrison, who testified to the accuracy of Mr. Harbord's conclusion as to the energy required for the direct smelting of the ore, his own experiments agreeing almost identically with Mr. Harbord's figures. He also endorsed Mr. Harbord's view that the electric furnace, even under English conditions, is a cheaper producer of high-grade steels than the crucible furnace. As to the cost per electrical horse-power year, he thought £4 would represent the cost with water power, and £7 to £8 for steam-driven plants. So far as the commercial possibilities of the case went, the Héroult, Keller, and perhaps the Kjellin furnaces were commendable—the rest might be eliminated as merely theoretical. Mr. Morrison disagreed in one respect with the writer of the paper, and that lay in the view that the production of structural steel electrothermically was possible. This view was based on some new information sent by M. Héroult, who was now putting down a 50-ton electric furnace, together with a mixer to hold 300 tons of steel. Also starting with ore only, the latest figures showed a consumption of 0.15 electrical horse-power year per long ton of steel, or starting with molten metal 0.05 electrical horse-power year. Mr. Morrison therefore looked forward to the use of the Héroult furnace in England.

Mr. Hutton was inclined to regard the whole question as in too early a stage for the passing of a definite verdict. In any case the Kjellin furnace did not seem technically good, or likely to be economical. The temperature of the smelted metal was likely to be low, an important matter with low carbon steels. Personally, he favored the electric treatment of an already-heated metal, and urged the trial of the electric furnace as an adjunct to the ordinary blast furnace. A combination of large gas engines using furnace gas, directly coupled to large electric generators, would probably be a feature of the first English experiments. With regard to Mr. Harbord's calculations of the cost of the amount of power required, had the effect of long stoppages on the cost been considered? Mr. Harbord had referred to the value of water-cooled electrodes, his (Mr. Hutton's) experiences with these had been very favorable, indeed he had been able to melt platinum wire within 2 inches of the water-cooled part of the [illegible].

Mr R. L. Gamlen (of the Lancashire Electric Power Company) dealt with the subject from the point of view of a supplier of energy in bulk. A single huge furnace would involve large fluctuations in the demand for power—perhaps several smaller furnaces might be so worked in alteration as to give a load factor approaching 100 per cent.

At the suggestion of the Chairman, Dr. Perkin, Mr. Harbord intervened to meet the remarks of the previous speakers. He held to his views regarding the commercial production of structural steel chiefly by reason of the cheapress and efficiency of the modern Siemens furnaces, wherein the cost of producing [illegible] steel from molten pig was only [illegible] per ton. He had estimated the cost of power in England at £10 per electrical horse-power year, this being the quotation of a large municipality for a 2,000 kilowatt undertaking, and would be glad if any engineers present could assure him of cheaper power. Of course the use of several furnaces would equalize the load. While he had advocated the separation of the blast furnace from the electric steel furnace, he had done so with a mental reservation in favor of the interposition of the ladle and traveling crane to connect the steel furnace with the blast furnace.

Mr. Gamlen responded to the invitation by offering power at £7 per electric horse-power year for 24 hours a day. If, however, its use could be confined to the 12 hours of the night, the charge would be about £4 per electric horse-power per year of 365 days. This being obviously impossible, the discussion was continued by Dr. Steinhart, who spoke of the inherent conservatism of the Sheffield manufacturer as the chief obstacle to the adoption of the electric furnace. With regard to arc furnaces, was not the carbon from the electrodes likely to contaminate the steel? Could the electric furnace give a guaranteed percentage of carbon? Mr. Hughes then congratulated the authors on having set out the essentials of the question without inflation, which had been so disastrous to electricty in England in the early years of the secondary battery. Letters were read from Mr. B. H. Thwaite, and from Mr. Bertram Blount, which were generally very favorable in the views entertained as to the commercial prospects for England.

In proposing a vote of thanks to the author, Dr. Perkin spoke of the industrial importance of such matters, and the need for experiment. Above all, the British manufacturer needed to be technically educated. In Mr. Harbord's reply, which concluded the meeting, the chief point of interest lay in his pointing out that the only unstopped source of fuel—the cheapness of which was so manifestly important—lay in the extensive use of blast furnace gas.

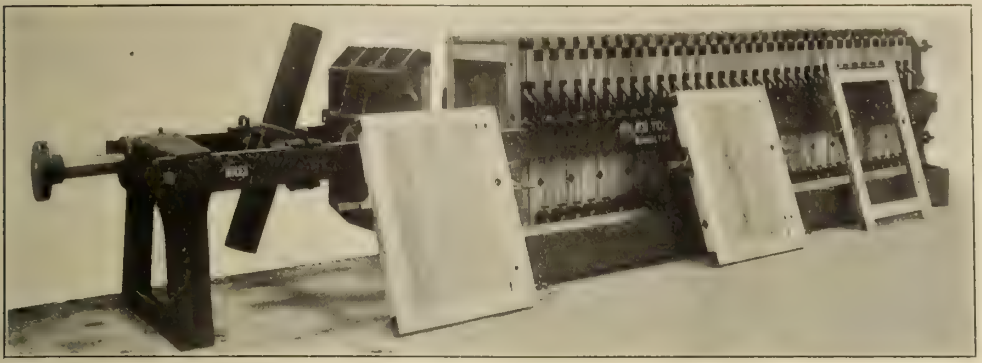
Materials Used in the Construction of Filter Presses.

By Emil Hatschek.

When designing plant or appliances for the chemical manufacturer, the engineer is often confronted with a difficulty which rarely presents itself in his other work: the difficulty that the materials on which he is accustomed to rely—practically three, cast iron, steel and gun metal—are unsuitable. The substance to be handled may either attack these materials so rapidly as to impair their strength in a short time, or, if their action is not quite so energetic, the presence of even small amounts of iron, etc., in the finished product, may be objectionable. This difficulty may be aggravated by mechanical conditions, as in the case of filter presses, when the apparatus is exposed to considerable pressure, and by the obvious impossibility of using the materials on which the chemist relies in the laboratory, namely, glass and platinum.

Notwithstanding these limitations, it has been found possible to construct filter presses suitable for practically all technical purposes, and as the various expedients adopted are not perhaps generally known to manufacturers, a brief description of the materials available and employed in filter press construction may be of interest, and may lead to the adoption of this very useful appliance in cases where some apprehension as to the possibility of using a suitable material has hitherto prevented its introduction.

Where no objection exists to its use, iron is, of course, the material generally used for filter press plates and frames. The large majority of makers employ cast-iron plates, either directly supporting the filter cloths, or covered with perforated steel plates, which give a better support to the cloths and prolong their life. The Niles-Bement-Pond Company have lately introduced a rather radical departure, by building up their plate from a corrugated steel panel and a cast-iron frame. This type was described in the issue of December, 1904, of this journal. The only possible objection to the use of steel is that it is more readily attacked by some liquors than is the skin of cast-iron plates, and that steel in contact with the cast iron sets up electrolytic action.



LARGE WOODEN FILTER PRESS.

The fittings on iron presses, viz.: the outlet cocks for the filtrate and the cloth unions, are generally made of bronze or brass, to prevent rusting or the possibility of sticking. When these materials cannot be used, *e. g.*, with alkaline or cyanide solutions, these parts can easily be made of cast iron, steel or malleable iron.

In a large number of cases, however, iron cannot be used, either because the liquor to be filtered is acid, or because it is absolutely essential to preclude any possibility of contaminating the solid matter by iron oxide, *e. g.*, with barytes, blanc fixe, china clay, etc. These requirements may be met by making the plates and frames of the press of an alloy or metal not attacked by the acid, such as gun metal, phosphor bronze or aluminium, or else of wood.

Gun metal presses, when they can be used, have the advantage of great strength and of being suitable for hot liquors, but their cost is very considerable. The frame work of the press is always made of iron, with gun metal liners in all passages for the liquor. In spite of their high price, gun metal presses have found a good deal of use in special cases: in England, for instance, gun metal presses 18 inches or 24 inches square, with frames 2 inches deep, are used by most of the white lead manufacturers.

When the cost of the machine or the nature of the liquid precludes the use of metal, recourse is had to the wooden filter press. This machine has plates and frames built up from a number of suitable pieces. The design is slightly modified from that of metal plates, to suit the exigencies of the material. The wood employed is pine, which is particularly suitable on account of its resinous nature, oak, or maple, and the writer has even found it possible to use white poplar, if impregnated with a heavy hydrocarbon. The bolts holding the plates and frames together are generally steel bolts, painted with asphaltum paint, but they, as well as the screws fastening the handles to the wood, can be made of bronze at little extra cost. If individual outlet cocks on all plates are used, these are often wooden faucets, but they are not very satisfactory for washing presses, on which they are exposed to the full pressure of the wash water, and are liable to leak. Hard rubber cocks answer very well in these cases.

The frame work of the press is, of course, iron, and similar to that of iron machines. As all passages for the liquor, washings, etc., are formed in the iron head, it is necessary to provide them with suitable liners. Gun metal, lead, tin or hard rubber pipe may be used, and one or the other of these materials will answer in most cases.

A large wooden press, in which most of the available materials have been drawn upon, is illustrated in Fig. 1. The machine is used for white lead and holds about 3000 pounds at a charge. The plates and frames are made of Georgia pine, and are bolted together with steel bolts, painted with asphaltum paint. The feed inlet is fitted with a gun metal liner, while the inlet for the wash water, as well as the outlet for the washings, are provided with hard rubber liners and cocks—these passages being subject to a slight pressure only. The outlet cocks on the plates are also made of hard rubber. The filtrate is discharged into a wooden trough. It may be worth mentioning that the press is provided with a special distance piece, which allows the press to be opened to the unusual extent of 4 feet, so that the frames may be tipped into a horizontal position, and the cake, which is very sticky, be discharged readily.

The presses just described answer for all organic and for dilute inorganic acids, but even concentrated sulphuric acid does not present an insuperable difficulty, as it is possible to construct filter presses of lead. The plates are either cast of reguline lead round a core of steel plate, which gives the necessary strength and rigidity, or cast-iron plates coated with lead are used. It will thus be seen that there is practically no substance requiring to be filtered on a commercial scale for which a suitable filter press cannot be made.

Metallurgical Calculations.—IV.

By J. W. Richards, Ph.D.

Professor of Metallurgy in Lehigh University.

The Eldred Process of Combustion.

A means of regulating the temperature of the flame has been proposed by Eldred, and described by Mr. Carlton Ellis in the December, 1904, issue of this journal. The proposition is simply to mix with the air used for combustion a certain proportion of the products of combustion themselves. The principle is easily understood when the requisite calculations of the theoretical flame temperatures are made. When solid fuel is burnt the temperature is often too high locally, and results in burning out grate bars or overheating the brick work of the fire-place, or overheating locally the material which is mixed with the fuel. If the air is diluted with products of combustion the initial theoretical temperature is lowered, and the above evils may be obviated. Using solid fuel, the heat in the fuel before the air actually burns it must be added to the heat generated by combustion to get the actual temperature in the hottest part of the fire.

Examples: (1) What will be the highest temeprature in a charcoal fire fed by air, assuming complete combustion without excess of air?

Assuming the charcoal to be pure carbon, and to be heated to the maximum temperature t before it burns (by the combustion of the preceding part), the heat available is:

Heat of combustion of 1 kilo of carbon...... 8100 Cal's.
Sensible heat in the carbon at t°............ $0.5t - 120$ "

Total heat available to raise the temperature. $7980 + 0.5t$ "
Products of combustion.........CO^2 1.85 m^3
N^2 8.89 "
Heat of product at t°..........CO^2 $1.85\,(0.37t + 0.00027t^2)$
N^2 $8.89\,(0.303t + 0.000027t^2)$

Sum $3.378t + 0.00074t^2$

therefore $3.378t + 0.00074t^2 = 7980 + 0.5t$
whence $t = 1872$

(2) What will be the temperature in the same case, if the air used is diluted with an equal volume of the products of combustion?

The heat available is the same as before, 7980 + 0.5t; but since the mixed air for combustion contains only half as much oxygen per cubic meter, the products will be exactly doubled in amount for a unit weight of carbon burnt, and we therefore have directly

$$2(3.378t + 0.00074t^2) = 7980 + 0.5t$$

whence $t = 1026°$

It is therefore evident that the maximum temperature of the hot gases at their moment of formation is nearly halved by the procedure stated.

(3) Taking the cases cited by Mr. Ellis, where the products contained originally 15 per cent oxygen and 6 per cent carbon dioxide, and after mixing the air used with half its volume of the chimney gases, 9 per cent oxygen and 12 per cent carbon dioxide (the gas-air mixture entering containing 15 per cent oxygen and 6 per cent carbon dioxide), what are the maximum temperatures obtained in the two cases?

Case 1: The heat available is as before; the products of combustion are CO^2 1.85 m³, O^2 4.62 m³, N^2 24.36 m³, and their sensible heat at temperature t—

$$\text{Heat in } CO^2 = 1.85\,(0.37t + 0.00027t^2)$$
$$\text{"}\quad O^2 + N^2 = 28.98\,(0.303t + 0.000027t^2)$$
$$\text{Sum} = 9.46t + 0.00128t^2$$

therefore $9.46t + 0.001280t^2 = 7980 + 0.5t$
whence $t = 800°$

Case 2: The products, per kilogram of carbon burnt, will be CO^2 3.70 m³, O^2 2.77 m³, N^2 24.36 m³, and we have

$$\text{Heat in } CO^2 = 3.70\,(0.37t + 0.00027t^2)$$
$$\text{"}\quad O^2 + N^2 = 27.13\,(0.303t + 0.000027t^2)$$
$$\text{Sum} = 9.09t + 0.00173t^2$$

therefore $9.09t + 0.00173t^2 = 7980 + 0.5t$
whence $t = 764°$

Conclusions: The calculations regarding the Eldred process bring out what was not stated in the printed description of the method, viz.: that if a small excess, or no excess of oxygen escapes, to the chimney, the temperature of the flame will be greatly reduced by the dilution of the air used, because more gas-air mixture will be required per unit of fuel burnt; but if there is any large amount of unused oxygen escaping in the first instance, the dilution practiced will scarcely affect the temperature of the flame at all, because it makes very little difference whether the fuel is heating up unused oxygen or carbon dioxide dilutant substituted for some of it, as long as there is more than enough oxygen to burn the fuel. The fact of the specific heat of carbon dioxide being greater than that of oxygen, is the only reason for the small calculated difference of 36°, in cases 1 and 2. It is true that the dilution practiced will result in less heat being lost in the waste gases, in the *Example* (3), but the same economy could be obtained by simply using less excess of air in the first instance.

TEMPERATURES IN THE "THERMIT" PROCESS.

The Goldschmidt process of reducing metallic oxides by powdered aluminium, igniting the cold mixture, is only a special case of our general rule, as far as concerns the calculation of the theoretical temperatures obtained. In any case, the total heat available is the surplus evolved in the chemical reaction, and the temperature sought is that to which this quantity of heat will raise the products of the reduction. The products are alumina and the reduced metal. The heat in the latter, in the melted state, is well known in many cases; it is most clearly expressed by the sum of the heat in such melted metal just at its melting point (which is easily determined calorimetrically, and is well known for many metals), plus the heat in the melted metal from the melting point to its final temperature, which is equal to t, minus the melting temperature, multiplied by the specific heat in the melted condition. These data are known for many metals; for some they may have to be assumed from some general laws correlating these values. The heat in melted alumina has not been determined calorimetrically, to my knowledge. Its sensible heat solid is $0.2081t + 0.0000876t^2$ (determination made in author's laboratory), which, evaluated for the probable melting point, 2200° C., would give the heat in it at that temperature 881.8 Calories; the latent heat of fusion of molecular weight is very probably 2.1T, where T is the absolute temperature of the melting point, making the latent heat of fusion per kilogram $2.1(2200 + 273) \div 102 = 5193 \div 102 = 50.9$ Calories. The specific heat in the melted state is probably equal to the specific heat of the solid at the melting point, viz.: $0.2081 + 0.0001752\,(2200) = 0.5935$. We, therefore, have for the heat in melted alumina at temperature t—

Heat in solid alumina to the melting point....	881.8 Calories
Latent heat of fusion......................	50.9 "
Heat in liquid alumina to its setting point....	0.5935 (t-2200)
Total.	932.7 + 0.5935(t-2200)

or, for a molecular weight, 102 kilograms: 95,135 + 60.54 (t-2200).

Examples:

(1) If black cupric oxide is reduced by powdered aluminium, what is the temperature attained?

The reaction is

$$\begin{array}{c|c} \multicolumn{2}{c}{3CuO + 2Al = Al^2O^3 + 3Cu} \\ -3(37{,}700) & +392{,}600 \\ \hline \multicolumn{2}{c}{+279{,}500} \end{array}$$

The products must therefore be raised to such a temperature that they contain 279,500 Calories. The heat in molecular weight of alumina at temperature t has already been found; that in copper at t° is, for one kilogram (using the author's determinations):

Heat in melted copper at setting point = 162 Calories.

Heat in melted copper above 1065° = 0.1318 (t — 1065) Calories.

Total = 162 + 0.1318 (t — 1065).

Per atomic weight (63.6 kilos.) = 10303 + 8.3825 (1 — 1065).

From these data there follows the equation:

$$95{,}135 + 60.54\,(t - 2200) + 3[10{,}303 + 8.3825\,(t - 1065)] = 279{,}500$$

whence $t = 3670°$

A little further calculating will show that approximately one-third of all the heat generated exists in the copper, and two-thirds in the melted alumina.

(2) If pure ferric oxide is reduced by the "Thermit" process, what is the temperature of the resulting iron and melted alumina?

The reaction and the heat evolved have been already given in the preceding instalment of these calculations. (q. v.). They show that per molecular weight of alumina formed there are two atomic weights (112 kilos.) of iron formed, and there is disposable altogether 197,000 Calories. The heat in a kilogram of pure iron at its melting point (1600°) is 300 Calories, the latent heat of fusion approximately 69 Calories, and the specific heat in the melted conditions 0.25. The total heat in a kilogram of melted iron is therefore 369 + 0.25 (t — 1600) Calories, or per atomic weight = 20,664 + 14 (t — 1600) Calories.

We, therefore, can write the equation:

$$95{,}135 + 60.54\,(t - 2200) + 2[20{,}664 + 14\,(t - 1600)] = 197{,}000$$

whence $t = 2694°$

A similar calculation made for the reduction of MnO by the theoretical amount of aluminium, shows a reduction temperature less than the melting point of alumina. This would mean that the melting down of the mass to a fused slag of pure alumina could not take place. What happens in the reduction of manganese is that an excess of manganous oxide is used, whereby all the aluminium is consumed, and none at all gets into the reduced manganese, and, furthermore, the excess of manganous oxide unites with the alumina to form a slag of manganous aluminate, which is fusible at the temperature attained. Without the latter arrangement no fused slag could result.

Similar calculations, made with silicon as the reducing agent, show similar difficulties regarding the theoretical temperatures attainable, when iron or manganese are reduced. By using an excess of the oxides of these metals, however, calculation shows that temperatures sufficient to fuse the metals and the manganous silicate, or ferrous silicate produced, ought to be obtained.

THE THERMOCHEMISTRY OF HIGH TEMPERATURES.

The problem is: Knowing the heat evolved (or absorbed) in the formation of a compound, or in a double reaction, starting with the reacting materials cold, and ending with the products cold, what is the heat evolved (or absorbed) in either of the following cases?

(*a*) Starting with the reagents cold and ending with the products hot.

(*b*) Starting with the reagents hot and ending with the products hot.

(*c*) Starting with the reagents hot and ending with the products cold.

Of these three cases (*b*) is the most general form of the problem, and occurs frequently in metallurgical practice, particularly in electrometallurgy; (*a*) is a more limited form, and requires less data for its calculation, while it is very frequently the desideratum in discussing the thermochemistry or heat requirements of a metallurgical process; (*c*) is derivable at once if the data exist for calculating (*b*), and is of such rare occurrence in practice that we can dispense with its lengthy discussion.

The thermochemical data already given and described in preceding sections enable us to calculate the heat of any chemical reaction starting with cold reagents and ending with the products cold. For instance: (Zn, O) = 84,800 Calories means that if we take 65 kilograms of solid zinc, at room temperature, and 16 kilograms of oxygen, as gas at room temperature, ignite them, and after the reaction cool the 81 kilograms of zinc oxide formed down to the same starting temperature, there will be developed a total of 84,800 Calories. Similarly, using the datum (C, O) = 29,160 Calories, we can construct and interpret the reduction reaction, starting with cold materials and finally ending with cold materials, as follows:

$$\begin{array}{c|c} \mathrm{Zn\,O} + \mathrm{C} = & \mathrm{Zn} + \mathrm{CO} \\ -\,84{,}800 & +\,29{,}160 \\ \hline \multicolumn{2}{c}{-\,55{,}640} \end{array}$$

This reaction, as interpreted, stands for none of the above cases (*a*), (*b*) or (*c*); in fact, it represents only a calorimetric determination in the laboratory, and does not correspond to either of the three cases actually taking place in practice, that is, it is not directly applicable to practical conditions, without being modified by the conditions actually arising in practice.

Case (*a*): If we, in practice, start with the reagents cold, and the products pass away from the furnace hot, at some determined temperature t, the total heat energy necessary to cause this transformation is calculable in two ways. The first way is to follow the course of the reaction, and to say that the total heat absorbed is that necessary to heat the reacting bodies to the temperature t, plus the heat of the chemical reaction assumed as starting and finishing at that temperature. The first of these items can be obtained if we know the sensible heat in the reacting bodies up to the temperature t; it is a question of specific heats of the reacting bodies up to t (including any physical changes of state occurring in them between o and t); the second item requires a knowledge of the heat of formation of all the compounds involved, starting with their ingredients at t, and ending with the products at t. But the latter item is the general question of the heat of a reaction starting with the ingredients hot and ending with the products hot; it is the most general case, which we have designated as *Case* (*b*), and which will be discussed later. This way of solving *Case* (*a*), therefore, really includes the solution of *Case* (*b*), and we will defer its consideration for the present. The second method of solving *Case* (*a*), and one which does not involve the more general solution, is to take the heat of the reaction, starting with the ingredients cold and ending with the products cold—the ordinary heat of the reaction from ordinary thermochemical data—and to add to this the amount of heat which would be required to raise the products from zero to the temperature t. This, of course, does not actually represent the sequence in which the reactions take place in practice, but it accurately represents the heat involved or evolved in passing from the cold reagents to the hot products, and is, therefore, exactly the practical quantity which we are endeavoring to find. Moreover, it involves a knowledge of only the specific heats of the products, and not that of the ingredients or substances reacting.

Illustration: Starting with a mixture of zinc oxide and carbon in the proportions Zn O and C, at ordinary temperature, and shoveling them cold into a retort, how much heat is absorbed in converting them into Zn vapor and CO gas, issuing from the retort at 1300° C?

If we start to calculate this quantity, by first finding the heat necessary to heat Zn O and C up to 1300°, that is obtained by multiplying the weights of each by their mean specific heats from 0° to 1300°, and then by 1300, as follows:

Calories.

81 kilos. ZnO $\times^{*} [0.1212\,(1300) + 0.0000315\,(1300)^2] = 17{,}075$

12 " C $\times^{*} [0.5\,(1300) - 120] = 7{,}800$

Sum $= 24{,}875$

To this must then be added the heat of the reaction $ZnO + C = Zn + CO$, starting with the reacting bodies at 1300°, and ending with the products at the same temperature. This can only be found by solving the general *Case* (*b*) for this particular reaction, which we will find involves a knowledge of the heat required to raise Zn, O, C, ZnO and CO from 0° to t. Anticipating such a solution, we may say that the heat of the reaction starting and ending at 1300° is — 75,216 Calories, making the sum total of energy required 100,091 Calories.

The solution is usually much simpler if we take the second method, and add to the heat of the reaction, starting and ending at zero (— 55,640 Calories), the heat required to raise 65 kilos. of zinc and 28 kilos. (22.22 cubic meters) of carbonous oxide, from their ordinary condition at zero to their normal condition at 1300°. The calculation for the CO gas is simply:

$22.22 \times [0.303\,(1300) + 0.000027\,(1300)^2] = 9{,}766$ Calories

For the zinc, the calculation is more complicated:

Heat in solid zinc to the melting point (420°):

$65 \times [0.0958\,(420) + 0.000044\,(420)^2] = 3{,}077$ "

Latent heat of fusion $65 \times 22.61 = 1{,}470$ "

Heat in melted zinc, 420° to boiling point (930°):

$65 \times [0.0958 + 0.000088\,(420)] \times (930 - 420) = 4{,}228$ "

Latent heat of vaporization (Trouton's rule)

$20 \times (930 + 273) = 24{,}060$ "

---

* Heat in 1 kilo of carbon, for temperatures above 1,000°, 0.5 t — 120 (deduction from Weber's results); zinc oxide 0.1212 t + 0.0000315 t^2 (determination by the author).

Heat in zinc vapor (monatomic) 5 ×
(1300 — 930) = 1,850 Calories

Sum = 34,685 "

The total sensible heat required is, therefore, 9,766 + 34,685 = 44,451 Calories, which, added to the 55,640 absorbed in the chemical reaction, if it started and ended at zero, makes a total heat requirement of 100,091 Calories for the practical carrying out of this reaction, starting with the reagents cold and ending with the hot products at 1300°.

[To be absolutely accurate, regard should be paid in the above case to the fact that the above calculations are based on the substances being all at atmospheric pressure, while in the mixture of Zn vapor and CO gas each is under only 0.5 atmosphere tension. Since each of these represents a molecular weight, the outer work which has been included in the calculations is $2 \times 2T = 2 \times 2\ (1300 + 273) = 6292$ Calories, whereas it should really be only half that much, or 3146 Calories. The corrected heat required is, therefore, 100,091 — 3,146 = 96,945 Calories, or 1464 Calories per kilogram of zinc. This datum is exactly the net heat requirement on which calculations of the net electrical energy required to produce zinc from its oxide, or calculations of the net efficiency of an ordinary zinc furnace, would be based.]

Case (b): To calculate the heat of a chemical reaction starting and finishing at any temperature t, two methods are available: The most general solution, and that easiest to understand, is to calculate for each compound involved the heat of its formation at the temperature t, that is, the heat evolved if the elements start at t and the product is cooled to t. Having these heats of formation at t, they are used in the equation in just the same manner as the heats of formation at zero are ordinarily used in obtaining the heat of the reaction starting and ending at zero. The calculations are based on this general principle: The heat evolved when the cold elements unite to form the hot product at temperature t equals the heat of union at zero, minus the heat necessary to raise the product from zero to t; if to this difference we add the heat which would be necessary to heat the uncombined elements from zero to t, the sum is the desired heat of formation at t°.

Illustration: The heat of formation of ZnO at zero is 84,800, starting with cold Zn and O and ending with cold ZnO. If we started with cold Zn and O and ended with hot ZnO, say at 1300°, the heat evolved altogether would be less than 84,800 by the sensible heat in the 81 kilograms of ZnO at 1300°, which has already been calculated (see previous illustration) to be 17,075 Calories. The difference is 67,725 Calories, and represents the transformation from cold Zn and O to hot ZnO. If the Zn and O were heated to 1300° before combining, they would contain as sensible heat the following quantities:

Heat in 65 kilograms of Zn (0 to 1300°)
already calculated 34,685 Calories
Heat in 16 kilograms of O = 11.11 m^3 ×
$[0.303\ (1300) + 0.000027\ (1300)^2] = 4{,}884$ "

Sum 39,569 "

And the heat evolved in passing from the hot reagents to the hot product at 1300° must be 67,725 plus this sensible heat, or 107,294 Calories. We can express this datum as follows: $(Zn, O)^{1300} = 107{,}294$, which means that when the zinc and oxygen taken in their normal state at 1300° combine to form ZnO at 1300°, 107,294 Calories are evolved.

A similar calculation for CO is as follows:

(C, O) = 29,160 Calories
Sensible heat in CO (0 to 1300°) already
calculated = 9,766 "

Heat evolved when cold C and O form CO
at 1300° = 19,394
Sensible heat in C (0 to 1300°) = 7800
" " O (") = 4884 12,684 Calories

Heat evolved hot C and O to hot CO = Sum = 32,078 "
Or $(C, O)^{1300} = 32{,}078$ "

Uniting the two data found, for the heats of formation of ZnO and CO at 1300°, in the equation of reduction, we have

$$
\begin{array}{lr|l}
 & ZnO + C = & Zn + CO \\
\text{at } 1300^\circ = & -\ 107{,}294 & +\ 32{,}078 \\
\hline
 & -\ 75{,}216 &
\end{array}
$$

The actual reduction is thus seen to absorb 19,576 Calories more at 1300° than at zero, an increase of over 35 per cent.

If to this heat of reaction at 1300° we add the heat necessary to raise the ZnO and C to 1300° (already calculated under *Case (a)* as 24,875 Calories), we will have a total heat requirement of 100,091 Calories, which would be required practically if we started with cold ZnO and C and ended with the hot Zn and CO. This agrees, as it indeed must, with the sum of the heat absorbed in the reaction at zero, 55,640, increased by the sensible heat in Zn and CO at 1300°, already found to be 44,457, or a total of 100,091.

Another method of calculating the heat of the reaction ZnO + C = Zn + CO at 1300°, without using the heat of formation of ZnO and CO at 1300°, is based on the following general principle: If from the heat of any reaction, starting and ending cold, there be subtracted the heat necessary to raise the products from 0 to t, the difference is the heat of the transformation from cold reagents to hot products; if to this be added the heat which would be contained in the reagents if they were heated to t, the sum is the heat of transformation from heated reagents to heated products, all at the temperature t.

Illustration: The heat of the reaction we have been studying, starting and ending cold, is — 55,640 Calories
Sensible heat in Zn at 1300° = 34,685
" " CO " = 9,766 44,451 "

Difference — 100,091 "
Sensible heat in ZnO at 1300° = 17,075
" " C " = 7,800 24,875 "

Sum — 75,216 "

The above is the simplest way of calculating the heat of any chemical reaction at any desired temperature, since it involves the knowledge of the heat capacities of only those substances which occur individually in the reaction, and not that of the elements of which the compounds present are composed. The above calculation, for instance, involves the heat capacities of ZnO, C, Zn and CO, but not that of oxygen, which does not occur free in the reaction.

Case (c): This hardly needs discussion, because if we have the data for calculating *Case (b)*, this case can be easily worked. If to the heat of the reaction at t° we add the heat given out by the products in cooling from t to 0, the sum is the total heat evolved in passing from the hot reagents to the cold products. This solution of the problem presupposes, however, the solution of *Case (b)*, and requires the maximum amount of data, but it has the advantage of following and representing the logical course of the reaction. A simpler solution is to add to the heat of the cold reaction at zero, the heat necessary to heat the ingredients to t, and the sum will be the quantity required.

Illustration: Taking the same case as before:

Heat of reaction at 1300° = — 75,216
Sensible heat in products at 1300° = 44,451

Sum = — 30,765

or, Sensible heat in reagents at 1300° = 24,875

Heat of reaction at zero $= -55{,}640$

Sum $= -30{,}765$

The reasoning involved in the above is simply that, starting with the reagents hot and ending with the products cold, the heat evolution must be the same whether we suppose the system to pass along the one path or the other.

GENERAL REMARKS.

It will be evident from this enunciation of principles and the illustrations adduced, that for many practical purposes the calculation according to *Case* (*a*) will suffice for finding the net energy involved in many metallurgical operations; it gives the sum total of energy necessary to be supplied to pass from the cold reagents to the hot products as they come from the furnace, but the two items of which this sum is composed do not actually represent the two items of the division of this total in the furnace, *i. e.*, into heat necessary to heat the reagents to the reacting temperature and heat actually absorbed as the reaction takes place. The latter item can only be calculated by one of the two methods explained under *Case* (*b*), and then the former item can be obtained by difference, or by direct use of the heat capacities of the reacting substances.

It is, of course, obvious that this whole subject of calculating the thermochemistry of high temperatures (as distinguished from the ordinary *zero thermochemistry*) necessitates the use of all available data regarding specific heats in the solid, liquid and gaseous states of both elements and compounds, and also in many cases of their latent heats of fusion and vaporization. When, however, the necessary physical data are known, or can be assumed with approximate accuracy, the way is open to make many calculations of the greatest value in practical metallurgy and chemistry. The exact heats of formation of chemical compounds at a certain temperature, which may be, and often are, very different from these values at zero, and having frequently different relations to each other, will explain in many cases many hitherto little understood and apparently contradictory reactions. The calculations also enable us to understand many reactions taking place only at high temperatures, to calculate whether they are really exothermic or endothermic at the temperatures at which they occur, and to compare these data with the data as to the heat of such reactions obtained by studying the chemical equilibrium of such reactions and deducing the heat of the reaction from the rate of displacement of the chemical equilibrium. One such example may suffice to suggest the large field here open to the scientific metallurgist.

Example: G. Preuner (*Zeitschrift für Physikalische Chemie*, March 15, 1904, p. 385) reports a long and careful investigation of the equilibrium of the reaction

$$Fe^3O^4 + 4H^2 \rightleftarrows 3Fe + 4H^2O$$

in which he finds the equilibrium constant for different temperatures, and calculates from its value at 960°, by the use of Van't Hoff's formula, that the heat value of the reduction at that temperature is — 11,900 Calories, and noticing the great difference between this value and the value derived from the ordinary heats of formation $(-270{,}800 + 4(58{,}060) = -38{,}560)$, he concludes that the Van't Hoff formula gives wrong results when applied to this class of reactions. Now, the facts are that Preuner's observations were good, Van't Hoff's formula is correct, and applies, but the thermochemical value of the reaction, — 38,560 Calories is the correct value only *for the reaction beginning and ending at ordinary temperature*. Our thermochemical principles enable us to calculate the heat of the reaction at 960°, as follows:

Heat of the reaction beginning and ending at zero $= -38{,}560$ Calories

Heat in products at 960°:

$3Fe = 3(56) \times [0.218(960) - 39]$ Pionchon $= 28{,}560$

$4H^2O = 4(22.22) \times [0.34(960) + 0.00015(960)^2] = 41{,}300$

69,860 Calories

Heat in reagents at 960°:

$Fe^3O^4 = 232 \times [0.1447(960) + 0.0001878(960)^2]^{*} = 72{,}384$

$4H^2 = 4(22.22) \times [0.303(960) + 0.000027(960)^2] = 28{,}075$

100,459 "

The heat of the reaction at 960° is, therefore, according to our method of *Case* (*b*), $-38{,}560 - 69{,}860 + 100{,}459 = 7{,}961$ Calories. It thus appears that Preuner's dilemma was mostly caused by his thinking that the thermochemical value of the reaction at zero should be a constant for any temperature; a proper thermochemical calculation removes the dilemma.

Since the whole treatment of the subject of the thermochemistry of high temperatures requires a knowledge of data concerning the heat capacity of elements and compounds in the solid, liquid and gaseous states, as well as of their latent heats of fusion and vaporization, the next instalment of these calculations will supply these data as far as they are known, and discuss a number of applications of these principles to various metallugical processes.

Gold Extraction by Cyanide.

Before a meeting of the Scottish Section of the Society of Chemical Industry, held at Glasgow, on March 7, Mr. JOHN S. MACARTHUR, one of the inventors of the MacArthur-Forrest cyanide process, delivered a most interesting lecture on "Gold Extraction by Cyanide—a Retrospect." The paper is full of reminiscences concerning the early days of this process, which, during the last twenty years, has conquered the world. The full paper may be found in the issue of April 15, of the *Journal of the Society of Chemical Industry.*

The process was the outcome of extended systematic researches made by the MacArthur-Forrest Research Syndicate, to help the Cassel Gold Extracting Company in London out of their troubles. The problem was to find a proper solvent. Here, as always in commercial work with wet processes, the chief difficulty was to get pure solutions. In this special case, it meant to find a solvent which would dissolve gold, but not the base metals. This latter stipulation made a great many solvents of gold unsuitable for industrial work. "Naturally this excluded all reagents of the aqua regia and acid type. We could not hope to restrain an acid solution from a vigorous attack on the base metals, which, owing to their stronger affinity and greater mass, competed unfairly with the gold. In fact, the industry of molecular attack on gold had to be protected to prevent the indiscriminate dumping of base metals into the solution."

Potassium cyanide was first tried in November, 1886, but the result was apparently negative, since they always used sulphuretted hydrogen for precipitation of the gold, and, getting none, they assumed erroneously that no gold had been dissolved. Eleven months later this mistake was recognized and the development of the process followed.

Mr. MacArthur dealt in the lecture at some length on the question which was disputed for such a long time, whether air is required for the solution of the gold. Elsner claimed this. MacArthur's own experiments appeared to point the other way, since it was not known at that time that oxygen remains uncombined in aqueous solution along with the cyanide. MacLaurin established this fact, and to him is due the equation for the solution of gold.

$$2Au + 4KCy + O + H_2O = 2AuCyKCy + 2KOH.$$

Mr. MacArthur then discussed the question (still little understood although really fundamental), whether the reaction between cyanide and metallic gold is reproduced exactly in

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[*] Approximate determination of heat in Fe^3O^4, made recently in author's laboratory, $Q = 0.1447t + 0.0001878t^2$.

the case of the gold contained in ores. "No allowance has been made for the number and variety of mineral substances, possibly unacted on themselves, but still capable of modifying the reaction between gold and cyanide. Highly suggestive work directed towards the elucidation of this complicated problem has been carried out by Gore, who found that metallic gold was dissolved more quickly when in contact with some minerals than with others, and that such inert substances as ground glass and sand had a distinctly accelerating influence. He found that a gold disc lying on clean white sand dissolved nearly five times as quickly as when merely immersed in cyanide solution. What is the nature of the action in this case, I do not pretend to say. No one has yet suggested that sand has any chemical action under the circumstances. Possibly they occlude oxygen—I cannot say. Another investigator found that the merest trace of ferric oxide induced the action. Again I do not pretend to explain; but, if this is so, the comparatively easy and rapid dissolving of gold from ores is accounted for, as practically all auriferous ores contain more or less ferric oxide."

The problem of getting the gold (and only the gold) into solution, was by far the most difficult part of the question. The precipitation of the gold from solution was comparatively easy. Mr. MacArthur favors, of course, precipitation by zinc shavings, although he says that "zinc precipitation is one of the weak points in the process; with care, the precipitation is complete to a few grains per ton, but the gold is collected in such fine mud, drying into dust, that loss is inevitable."

With electrolytic precipitation the trouble is that it gets more and more difficult to precipitate the gold out of the solution, the further the precipitation proceeds. "In practice there was always a comparatively high amount of gold left in the solution, which, though used over and over again on fresh lots of ore, led to loss by leakage and dissipation. In fact, it is a wonder to me, and a credit to the technical staff concerned, that good results were really obtained. The electric precipitation method is less used now than ever and seems to be dying out." This development is, according to the author, aided by the more extended introduction of the zinc-lead couple (by adding a small quantity of a lead salt to the gold-cyanide solution as it enters the zinc box).

Mr. MacArthur then gave some figures showing what the cyanide process has done for gold metallurgy. "One of the most beneficial effects of the cyanide process has been the introduction of exact methods, and working where rule of thumb had reigned supreme. Twenty years ago only a very few of the largest gold mines had their ores and tailings systematically assayed. Now, every gold mine of the slightest pretensions has its assay staff and equipment. In the old days the manager, who was miner, metallurgist and commercial head in one, knew how much gold he saved, but he did not know hom much he lost; if he were a 'practical man' he rather feared to know this latter, and stoutly maintained that he recovered all the gold. All this is changed; cyanidation compelled the exact weighing, measuring and computing in its own department, and the cyanide department is generally the last. The exactitude had to begin at the beginning, hence a gold mining and reduction establishment is now conducted on lines that would do credit to a highly organized factory or even to a royal mint."

Mr. MacArthur finally made some suggestions as to possible improvements of the cyanide process. "The first point on which improvement is suggested is reduction in price of potassium cyanide. In the early days of the process (1892) we could buy potassium cyanide 98 per cent strong at 1s. 4d. per pound, now it is about 8d.; but as the consumption does not average one pound per ton of material treated, the saving of 8d. per ton is not of such importance as to materially affect the scope or economy of the process. In fact, if cyanide were as cheap as common salt, the economy on material presently worked would be measured only by pence per ton. I look for improvements in other directions.

"In ordinary cyanidation the extraction of gold probably does not average more than 80 per cent—any modification or improvements that would raise this average to 95 per cent would be much more valuable than a fall in the price of cyanide to the price of common salt. Assuming the average grade of tailings now treated to be 5 dwts. per ton, an additional recovery of 15 per cent would equal 3s. per ton, which is about the whole working cost of cyanidation. Thus, in my opinion, the betterment of the extraction is a most tempting field for research improvement.

"There is also improvement wanted in widening the field of operation in which cheap cyanide or a cheap process of cyanide recovery may play an important part. Ores containing a small amount of copper absorb so much cyanide that they cannot be commercially worked. This barrier still exists, though efforts with good promise of success to overcome or remove it have been made by several skilful workers. I anticipate that some day we shall be able to cyanide cupriferous gold ores—if so, the scope of the process will have been substantially enlarged.

"The cyanide process has seldom, if ever, been adopted for working purely silver ores, though it successfully treats many ores containing more silver than gold. Silver ores are even more complex and difficult to treat than gold ores, and, being more plentiful, a silver extraction process is urgently wanted, and doubtless some person, destined to succeed, is working out the problem now."

Mr. MacArthur refrained from speaking on his patents and restricted himself to a sympathetic quotation from Sir Isaac Newton: "If I get free of this present business I will resolutely bid adieu to it eternally, except what I do for my private satisfaction or leave to come out after me; for I see a man must either resolve to put out nothing new, or become a slave to defend it."

Vanadium and Titanium.

By Hans Goldschmidt, Ph.D.

(Concluded from page 170.)

TITANIUM.

With respect to titanium it may first be pointed out that it is useful as an addition to steel as well as especially to cast iron. The addition of titanium to cast iron by means of the aluminothermic reaction has already been discussed in this journal (Vol. I, p. 533, and Vol. II, p. 405). To describe this application briefly, a box filled with an aluminothermic mixture, which gives off ferrotitanium, is introduced into the cast iron. This box is fixed on an iron rod and brought into the bath so that the reaction of the mixture occurs on the bottom of the crucible.

This produces an effect in two respects: firstly, titanium is alloyed with the cast iron; secondly, and this is of special importance, the whole content of the crucible is most thoroughly stirred and mixed while the reaction goes on (its duration being about a minute); the result is that all impurities, particles of slag, etc., are brought to the surface. At the same time the chemical composition of the iron is changed, not only by the introduction of small quantities of titanium, but by the thorough refining of the bath, due to the mechanical stirring.

It is remarkable that even very low percentages of titanium (below 0.1 per cent) may produce remarkable changes, if introduced into cast iron by the so-called box reaction. A large works in the Rhenish-Westphalian industrial district has recently made experiments in great detail on the action of titanium on cast iron. Table I gives the results with respect to bending strength and tensile strength for a certain kind of cast iron treated with thermit, the analysis being also given in the table. The figures are averages of several determinations. The tests were made with rods of four different dimensions;

in the case of bending strength the load being applied in the center. I is a rod, 1000 mm. long, between the two points near the terminals on which it rests; the cross-section is square, 30 x 30 mm. Rods II, III, IV have a circular cross-section: II, diameter 40 mm., length 800 mm.; III, diameter 30 mm., length 600 mm.; IV, length 200 mm. *a* always refers to cast iron without titanium thermit, *b* to cast iron treated with titanium thermit. Although the table shows that in this case the strength of the iron has been increased more or less by means of the "box reaction," yet these figures alone do not give an exact idea of the useful application of titanium, since, as well known, a greater strength of the iron may also be produced easily by other means.

Table I.

Rod I.		Rod II.		Rod III.		Rod IV.	
Bending Strength in Kg.							
a	*b*	*a*	*b*	*a*	*b*	*a*	*b*
22.12	22.48	22.83	27.62	26.54	26.80	27.69	30.39
Tensile Strength in Kg.							
12.64	13.39	12.59	14.57	14.37	15.19	15.29	15.9

Analysis.

a 0.766 Mn, 0.541 P, 3.030 Si, 0.153 S, 3.513 C, 2.805 graphite.
b 0.698 Mn, 0.534 P, 2.962 Si, 0.142 S, 3.515 C, 2.738 graphite.

The comparative analysis in Table I does not show very much, although it is evident that the content of manganese and sulphur has decreased in the iron treated with titanium. In general, however, the content of sulphur is even more changed by the "box reaction" than is shown by Table I.

It is somewhat surprising that in spite of the difference in the strength of the material the analyses show very small differences in the composition. This is due to the fact that the chemical analysis, as carried out at present with our ordinary means, do not show sufficiently the quality of iron and steel. For, above all, dissolved oxides or gases can be determined only with great difficulty by chemical analysis. As far as such analyses are possible at all, their great difficulty prevents them from being used in the ordinary course of a steel analysis. I will call attention specially to the impossibility of determining dissolved oxygen compounds of chromium and manganese—oxides which cannot be reduced by hydrogen—when metallic chromium and manganese are simultaneously present. It is just such dissolved oxides which play an important part in steels. By the "box reaction" and by the simultaneous introduction of small quantities of titanium, the iron is freed from impurities, like gases and oxides, the latter being reduced. This explains why in this case chemical analysis does not show the improvement of the material; the improvement manifests itself in mechanical respect, *i. e.*, the cast iron gets a denser structure, the formation of pores is avoided and the tensile strength and bending strength increase.

By means of the aluminothermic reaction a certain quantity of heat is produced in the cast iron, but it is comparatively small, when calculated in calories. Nevertheless, a much greater fluidity is imparted to the cast iron. The mechanical stirring effect is undoubtedly a special factor. Now, since the fluidity of the iron is increased, it is possible to cast it at a lower temperature. Every practical engineer knows what this means in the case of complicated castings. Moreover, in this case piping is much less than in iron cast at a higher temperature, and it is thus much easier to avoid the undesirable pipe holes in the casting. Titanium has an important effect on the density of the casting, since the formation of pores is entirely avoided by proper care.

It is unnecessary to say that the treatment of cast iron with titanium is not a universal remedy to prevent all and any undesirable surprise with the completed casting during testing and machining. It is impossible to give a prescription which fits all cases. In the introduction of titanium thermit into foundries it has been found that every foundry must acquire its own experience, especially on the question for which castings the use of titanium thermit is advisable, and for which the expense, although low, is unnecessary.

So much about the application of titanium in cast iron. More recently titanium is introduced into steel in form of a 20 per cent ferro-titanium alloy, free from carbon. After it had been placed on the market it took some time until this product became popular; but since about two years it is regularly used by a number of steel works. It is applied in a quantity which I had already proposed at an earlier date on the basis of some experiments, namely, about ½ to 1 per cent. Experiments, with up to 6 per cent titanium, have so far failed to give good results.

An addition of titanium to steel, which contains about 0.8 to 1.2 per cent of carbon, scarcely increases the hardness of the steel, but it raises its elasticity, and is, therefore, advantageous for steels which are subjected to much percussion, and which it is important to render as little brittle as possible. It is also used for cutting tool steels.

It is remarkable that an addition of titanium to steel increases its toughness, and there are a number of steel manufacturers who would add small quantities of titanium to any steel, in order to produce a denser structure and to combine nitrogen. Such a varied application of titanium, however, has not yet been introduced into practice. This is another indication of the difficulty of introducing a new element with sure success into commercial work; and with this remark I am led back to what I said in the introduction.

ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS.

By George P. Scholl, Ph. D.

Electric Furnaces and Furnace Products.

Material for Furnace Linings.—H. G. Turner, London, England. Patent 785,841, March 28, 1905. Application filed April 24, 1903.

The material is prepared by heating magnesite in an electric furnace at the temperature of the electric arc to a point at which the material crystallizes. The product is either allowed to cool or may be rapidly cooled by being poured or thrown into water or another liquid. The product is stated to be of a crystalline character, and black, grayish or greenish color, and has a specific gravity of 3.58. It is ground and mixed with a suitable binder, such as a solution of magnesium chloride, water glass, boric acid, borax, etc. It is then made into a paste with water and applied as a coating to the furnace lining, or pressed into bricks or plates, which are afterwards baked.

Electrode for Electric Furnaces.—E. F. Price, G. E. Cox and J. G. Marshall, Niagara Falls, Assignors to Union Carbide Co. Patent 785,832, March 28, 1905. Application filed Oct. 19, 1903.

The invention relates more particularly to electrodes used in calcium carbide furnaces. On account of their high conductivity, uniform composition and durability, artificial graphite rods have been found desirable for this purpose, but they suffer from the disadvantage of unduly transmitting heat from the zone of reaction upwards to the metallic holder. The electrode constructed by the inventors is intended to overcome this disadvantage. For this purpose the electrode holder is provided with a water chamber, and each of the graphite rods is screwed into a metallic socket and projects into the water chamber, so as to allow an intermediate space for the circulation of water. These water-cooled holders, moreover, are preferably introduced into the water-cooled cast-iron top plate of a hood, such as is used in the Horry carbide furnace. The high specific conductivity of graphite as compared with carbon allows the use of much thinner rods of the former material, but they are mechanically weak and will break if merely substituted for the carbon electrodes of a carbide furnace. Furthermore, these thin graphite rods are readily oxidized by the atmosphere, owing to the high temperature to which they are raised. It is, therefore, necessary to reinforce and strengthen them, and this is preferably done by a filling, of acement of high resistance, placed between and around the rods. The cement may consist of ordinary asbestos or furnace-cement, mixed with ground bituminous coal and siloxicon. To strengthen the body of cement and protect it from scaling off when heated, a support of thin "expanded iron" is wrapped around the rods and plastered over with the cement. It is also sometimes desirable to protect one or more of the front rods of each electrode, which are more exposed to air, by coating them with tar and pushing them into a closely fitting iron sleeve, the upper end of which is screwed into the electrode holder. It is stated that this invention makes it possible to use graphite electrodes in a carbide furnace, the greater cost of the graphite being offset by the reduction in the thickness and the length of the electrode rods.

ELECTROLYTIC PRODUCTION OF METALS AND COMPOUNDS.

Process of Electrodepositing Metals.—E. D. Kendall, Brooklyn. Patent 786,221, March 28, 1905. Application filed Oct. 28, 1903.

The process relates particularly to the deposition of zinc on iron and steel, the so-called cold galvanizing. It depends essentially upon the use of zinc sulphoglycerate as electrolyte. The electrolyte is prepared by first acting on anhydrous or nearly anhydrous glycerol with strong sulphuric acid, with or without the application of heat, in order to form sulphoglyceric acid. This product is then diluted with a limited quantity of water and then saturated with zinc, preferably by agitating it with an excess of zinc oxide or zinc hydroxide, and finally more water is added so as to form a comparatively dilute solution of zinc sulphoglycerate, which is used as electrolyte. If a somewhat concentrated solution of zinc sulphoglycerate is used as electrolyte, the liquid is stated to have a high conductivity, and a current having an e. m. f. of even less than 1 volt, when a zinc anode is used, will suffice for a rapid deposition of zinc. With a more dilute solution an e. m. f. of 2 or 3 volts is preferably used, unless the conductivity of the electrolyte be increased by the addition of a suitable salt, such as sodium sulphate. With a properly adjusted current, the density of which is not disclosed, the pure zinc deposited on polished cathode surfaces is stated to have remarkable smoothness and uniformity. Electrolysis is facilitated by heating the electrolyte, but when it is used at ordinary temperature the rate of deposition and the character of the deposited zinc is stated to be satisfactory, while the operation requires less supervision.

Deposition Tank.—T. E. Lewis and J. A. Corey, London, England. Patent 786,978, April 11, 1905. Application filed Feb 2[illegible] 1904.

The invention relates to the electrodeposition of metals and particularly to the production of printers' electrotypes. The apparatus comprises a circular vat, upon which is mounted an annular cathode, which can be rotated by suitable mechanical means. The moulds to be electrotyped are suspended from this cathode ring. The anodes are supported upon a metallic ring, [illegible] a spider, which is fastened upon a spindle centrally [illegible] within the vat. The cathode was made rotating for the purpose of using high-current densities, made possible by this means, so as to effect a uniform and reguline metallic deposit and to carry on the electrodeposition rapidly.

Electrolytic Method.—A. S. Ramage, Detroit. Patent 782,221, April 11, 1905. Application filed Jan. 31, 1905.

The method provides for the electrolysis of a metallic salt in aqueous solution in a suitable cell, which is usually divided into a positive and a negative compartment. According to the nature of the electrolyte, a metallic hydroxide or a metal is produced at the cathode, while at the anode a secondary product, the so-called aniline black, is produced by the oxidation of an aniline salt, preferably the sulphate or hydrochloride of aniline. As electrolytes either the sulphate or the chloride of sodium may be used, preferably in a cell of the mercury cathode or the gravity type. In either case sodium is deposited at the cathode and converted into hydroxide in the usual manner. In electrolyzing sodium sulphate anodes of lead are preferably used, no oxygen being evolved at the anode, but converting the aniline sulphate $(C_6H_7N)_2H_2SO_4$, into aniline black. If sodium chloride is electrolyzed with graphite anodes in the presence of aniline hydrochloride $(C_6H_7N.\ HCl)$ aniline black is formed. The oxidation in this case is stated to be presumably due to the decomposition of water by chlorine in presence of the aniline salt, according to the equation, $Cl_2 + H_2O = 2HCl + O$. If ferrous sulphate be electrolyzed in presence of aniline sulphate, aniline black is formed, while metallic iron is deposited on the cathode. The anode liquor containing the sulphuric or hydrochloric acid, as the case may be, formed during the electrolysis, and the aniline black in suspension is withdrawn from the cell, preferably continuously, the analine black is separated by filtration or otherwise, and the acid liquid utilized for the production of additional quantities of the aniline salt.

Manufacture of Carbon Tetrafluoride Gas.—J. A. Lyons and E. C. Broadwell, Chicago. Patent 785,961, March 28, 1905. Application filed Sept. 16, 1903.

The process is carried out by fusing in a metallic crucible a mixture of sodium and potassium fluoride, although the fluorides of any of the alkaline earth or earth metals may be used, among which CaF_4 is suitable and practicable. The crucible serves as cathode and is covered by a non-conducting lid, through an opening in which is slipped a non-conducting tubular vessel with an open-work bottom, a feed opening in its upper part and a gas escape pipe. Into this vessel is introduced a graphite rod, reaching to its bottom and connected with the anode. Around its lower end is placed a quantity of charcoal, lampblack or other suitable carbonaceous material, which floats upon the surface of the bath. A cylindrical partition, open at the bottom, prevents the products formed in the anodic tube from passing towards the cathode. When the melt is electrolyzed, the metal is set free upon the cathode, while fluorine gas is liberated at the anode, where it combines with the carbonaceous material surrounding the anode rod, with the production of carbon tetrafluoride gas. The reaction is stated to take place at a temperature of approximately 1000° C. The carbonaceous material surrounding the anode is stated to protect the comparatively costly anode from unnecessary waste.

Production of Boron by Electrolysis.—J. A. Lyons and E. C. Broadwell, Chicago. Patent 785,962, March 28, 1905. Application filed Sept. 16, 1903.

The invention aims at the production of boron in conjunction with highly electropositive metals, such as potassium, sodium, manganese, chromium, molybdenum, tungsten, uranium and vanadium. For this purpose the borates of these metals, either basic or neutral, are electrolyzed at a temperature of 1100° C., in a tapered iron vessel, which acts as the negative electrode. A carbon rod is used as the positive electrode, which is placed into a cylinder of graphite, open at the bottom, and suitably insulated, which acts as a partition, and thus serves to keep the products obtained at the anode and cathode separate. During electrolysis the borates are tested to be decomposed in such a manner that the metal is set free upon the inner surface of the containing vessel, while the hypothetical ion, or radical B_4O_7 or B_2O_4, is deposited at the positive electrode. The carbon of the latter, being kept at an incandescent heat by the current on account of its relatively small area, is said to reduce the B_2O_4 or B_4O_7 to boron, carbon monoxide gas being set free. By surrounding the carbon anode with different metals their borides are claimed to be obtained.

Method of Extracting Metals from Ore by Electricity.—E. L. Priest, Oakland, Cal. Patent 784,885, March 14, 1905. Application filed April 5, 1904.

The inventor first proposes again the well-known expedient of bringing the ores to be treated into suitable form, by finely grinding them and mixing them with carbon and a binding material, so that the resulting material may be used as electrodes. He also illustrates a tank, filled with a liquid of any suitable character, such as acidulated or salted water. At one side of the tank, quite close to its side, a zinc plate is introduced in a vertical position, which is connected with one terminal of the current. Above the middle of the tank, just touching the water, is an electrode prepared as above described. The process is explained by him as follows: "Where the prepared ore masses touch the water the current is completed through them, and as the arc is established the conductor is consumed, while the ore is reduced and drops into the water. The carbon conductors serve also to heat the ore. These ore masses, with their consumable conductors, may be prepared in any suitable manner, sizes and numbers, and a plurality of them may be connected in multiple arc. By such initial preparation of the ore, the electric current is completed through them, and large quantities may be handled with practicability. The metal cannot volatilize as it is precipitated into the water, which also purifies the metal, divesting it of sulphur, the latter forming sulphuretted hydrogen, which passes off." Comment is evidently unnecessary.

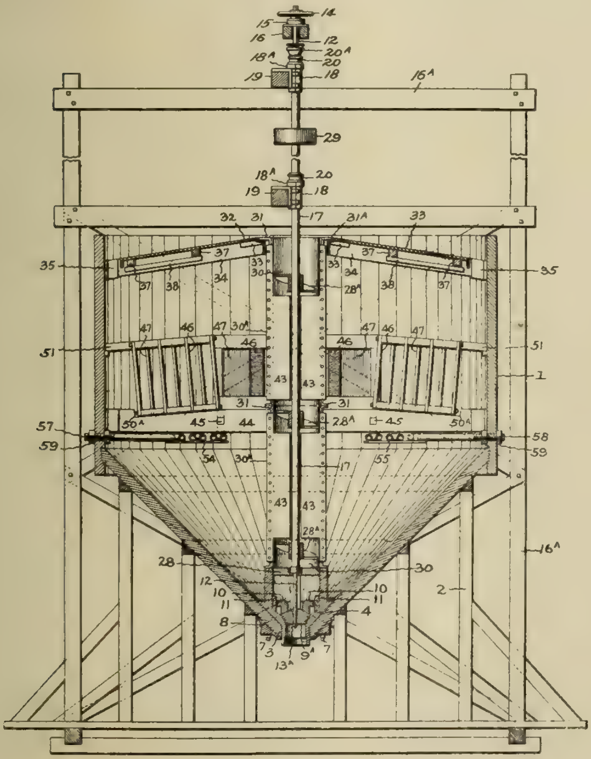


FIG. 1.—AGITATOR.

Apparatus for Extracting Metals from Their Ores.—W. A. Hendryx, Los Angeles, Cal. Patent 785,214, March 21, 1905. Application filed Aug. 29, 1904.

The apparatus is intended for the treatment of pulverized ore, in the form of a pulp consisting of ore and chemical solutions. It is illustrated in vertical cross-section in Fig. 1, and consists of a stationary tank 1, of a straight cylindrical form in its upper part, while provided with a sloping bottom. A discharge opening 3 is located in the axial center of the conical bottom, which is closed by a valve 9^a, which can be lifted by means of a hand-wheel 14 on top of the apparatus, fastened to the valve stem 12. A number of screw propellers 28^A are also placed on the shaft, the latter being revolved by means of a belt on the pulley 29. Upon the top of the valve casing is secured the lower end of a cylindrical casing 30, the interior of which communicates with the tank by openings in the valve cage. A device 32, called the deflector, serves to distribute the ore pulp. The anodes 46 consist of carbon plates suitably secured in frames, while the cathodes are lead plates Each cathode can be easily lifted out. As some ores are treated with advantage by heating the pulp, provision is made for heating coils 54 and 55. The operation of the apparatus is as follows: Pulp, to which solution has been added so as to bring it to the desired consistency, is run into the tank until the latter is about full. Power is then applied to the shaft and the propellers by means of the pulley 29, and the propellers are rotated rapidly, which has the effect of lifting the ore pulp bodily in the central casing. The pulp overflows over the top of the deflector in contact with the atmosphere and is thus aerated. As it drops from the edge of the deflector it falls into the tank and flows into direct contact with the anode and cathode plates, along which it descends in a gentle stream. It passes then into the central casing again, and is in turn elevated again and the whole solution is thus continuously circulated and agitated. Each charge of pulp is treated for a certain length of time, according to circumstances, and usually varying between 2 and 12 hours, until tests show that a satisfactory extraction of the values have been obtained. The charge is run into receiving tanks, and the clarified solution returned and strengthened and used again with other charges in the tank. (See also our volume II, 286.)

Process of Producing Metals and Alloys.—H. C. Blackmore, Mount Vernon, N. Y. Patent 786,185, March 28, 1905. Application filed Nov. 15, 1904.

The invention relates particularly to the production of aluminium alloys from metal aluminates. The process is carried out by first fusing, in a suitable vessel, oxides of lithium and calcium, preferably by the passage of an alternating current. After the charge is fused, an aluminate of such metal is added to the bath, an alloy of which with aluminium is desired. This readily dissolves and is simultaneously electrolyzed with the production of an aluminium alloy. The solvent bath of metallic oxides must consist of oxides of metals which have a greater affinity for oxygen than is possessed by the metals or alloys to be produced, the principal metals having a greater affinity for oxygen than aluminium being lithium, calcium and magnesium. Preferably, however, a mixture of calcium and lithium oxide is used, the latter reducing the spec. gr. of the bath and its melting point. The apparatus in which the process is carried

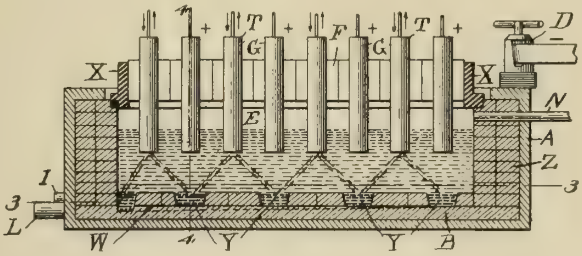


FIG. 2.—ELECTROLYTIC APPARATUS.

[illegible] as shown in vertical longitudinal cross section in Fig. 1, consists of an iron box, *A*, lined with magnesia brick, *Z*. The latter also covers the carbon bottom, *B*, except in channels beneath the direct-current anodes, *G*, towards which the uncovered parts of the bottom act as cathodes. They also act as electrodes opposed to the alternating-current electrodes, *T*. In starting the process an alternating current is established between the electrodes, *G*, and the bottom, so as to produce arcs, in order to fuse the material of the charge as it is fed in, the electrodes, *G*, being withdrawn more and more from the bottom as the depth of the molten charge increases. When enough has been accumulated, the alternating-current contact with the electrode, *G*, is broken and transferred to the electrodes, *T*. The electrodes, *G*, are then connected with the direct current, and a metal aluminate, such as copper aluminate, is fed in. The alloy or metal accumulates in the channels, *Y*, and is withdrawn at *L*. Instead of aluminates there may be used chromates, vanadates, stannates, tungstates, molybdates, etc., the alloys of which are desired.

Process of Extracting Aluminium or Other Metals.—H. S. Blackmore, Mount Vernon, N. Y. Patent 786,244, March 28, 1905. Original application filed Sept. 12, 1903; divided and refiled Jan. 7, 1905.

The electrolyte proposed by the inventor consists of a mixture of aluminium oxide with aluminium fluoride, about one of the former to two of the latter by weight. The charge is fused in about the same manner as outlined above, namely, by means of an alternating current between electrodes different from those used for electrolytic purposes. When enough of the fused charge has accumulated, electrolysis is started, by means of a current passing between carbon anodes and the carbon lining of the electrolytic vessel. The tension of the current is given as 3 volts, and the current density as 85 amperes per square inch of anode surface exposed. The fluorine contents of the bath are said not to be attacked during electrolysis. The temperature is maintained at approximately 1800° F. The bath is replenished with aluminium oxide as required.

Electroplating Apparatus.—H. Fleischer and C. H. Fleischer, New Britain, Conn. Patent 784,617, March 14, 1905. Application filed May 23, 1904.

The apparatus comprises a rectangular tank which contains the plating solution. The anodes as well as the cathodes are moved mechanically through the solution by means of driving chains, one of them being placed at each longitudinal side of the tank. The cathode chains are placed a little below the anode chains. A series of rods connects the two anode and the two cathode chains respectively, the anodes being suspended from the former and the cathodes from the latter. The guide sprockets over which the cathode chains run are so arranged that the cathodes will be lowered alternately with the anodes into the solution. Each anode is lifted up over the edges of a small tank and immersed for a short time in a cleansing solution, with which this tank is filled. By causing the articles to be moved through the solution alternately with the anodes, each line of articles suspended from a cathode carrier is moving into a sphere of solution which has been enriched by the dissolution of the anode immediately in front of it, which anode practically recharges the solution, which has been partially depleted of its metallic contents by the cathode immediately preceding it. Various preparatory baths for the articles to be plated as well as washing tanks can, of course, be added, to be traversed by the movable chains and the anodes and articles suspended from them.

Process of Metallizing Fabrics.—C. Danilevsky and S. Tourchaninoff, St. Petersburg. Patent 785,541, March 21, 1905. Application filed July 20, 1901.

The process applies to the deposition of metal by electrolysis through the whole thickness of fabrics which are porous and pervious to liquids, such as leather and its substitutes, asbestos, linen, wood, cotton, etc. It is carried out in an ordinary rectangular wooden vessel, provided with a series of rods placed across its top, from which are suspended alternately copper plates and frames containing the fabric to be metallized. The latter has been subjected to a preliminary boiling process in a suitable alkaline or acid liquid, and one of its faces has received a backing of a copper wire fabric or metal plate or other conducting layer. The bath for electroplating is either composed of citrate of copper, 10 to 20 grams, and boroglyceric acid, 2 to 6 grams, to 1 liter of water; nitrate of copper, 10 to 20 grams, and lactate of ammonium, 5 to 10 grams, to 1 liter of water, or borate of copper, 5 to 15 grams, and boroglyceric acid, 2 to 6 grams, to 1 liter of water. The backing consists preferably of thin wire gauze rubbed with graphite, so that it can easily be separated from the metallized fabric. The latter, after metallization, is washed and immersed in an aqueous solution of chloride of tin and cyanide of potassium, so as to cover the metallic particles with tin.

Plating Apparatus.—L. Schulte, New York. Patent 787,701, April 18, 1905. Application filed Sept. 17, 1904.

The apparatus is intended for plating all kinds of articles, especially sheet metal, band iron, wire and the like. It consists of a tank filled with the electrolyte, which contains two anode plates, between which passes a movable cathode carrier in electrical contact with contact members situated at the ends of the tank. The movable cathode carrier is preferably formed of a number of insulated endless cables, connected with each other by insulated transverse bars, provided with contact points. The insulated cables pass around rollers journalled in the sides of the tank at or near the ends. If it is desired to plate small articles the latter are placed in a metallic perforated tray or basket.

Method of Electrolytic Separation.—W. Hoopes, Pittsburg, Pa. Patent 788,315, April 25, 1905. Application filed Nov. 30, 1904.

The inventor claims to have discovered that such metals which cannot be readily separated from aqueous solutions of their salts can be readily obtained if these salts be dissolved in liquid anhydrous ammonia. The process is stated to be especially applicable to the production of magnesium by the electrolysis of a solution of magnesium chloride in anhydrous ammonia. The process requires a vessel in which the ammonia can be held in liquid condition. The apparatus shown in the specification consists of a glass vessel, set in an outer vessel filled with alcohol or other liquid of very low freezing point. It contains a coil of pipe, through which a refrigerating fluid, such as carbonic acid, ammonia, etc., circulates. This chills the alcohol, which in turn chills the liquid ammonia in the inner vessel, and prevents substantial loss by evaporation. The electrolytic vessel is divided by a porous partition into two compartments, one of which contains an anode of carbon, and the other a cathode of iron. Both compartments are filled with liquid anhydrous ammonia. Magnesium chloride is dissolved in the anode compartment, and anhydrous magnesium chloride, with or without a little sodium chloride, in the cathode compartment. Both solutions are preferably saturated, an excess of magnesium chloride being preferably added. Upon passing the current, magnesium is deposited at the cathode and chlorine liberated at the anode, a current of 5 volts with a current density of 0.1 amp. per square inch of anode surface having been found suitable. The inventor states that he has separated sodium, potassium, chromium, iron and silver by following methods similar to that described for magnesium. Thus, iron has been purified by immersing an iron cathode and an iron anode, containing silicon and carbon, in a solution of iron, cyanide and potassium cyanide in ammonia, pure iron being obtained on the cathode, carbon and silicon not being dissolved.

Process of Reducing Metals from their Solutions.—C. B. Jacobs, East Orange, N. J., assignor to the Ampere Electrochemical Co., Port Chester, N. Y. Patent 788,584, May 2, 1905. Application filed June 8, 1900.

The invention relates to the reduction of gold from its cyanide and chloride solutions by means of phosphide of hydrogen, and is an improvement upon a former patent granted to the inventor. He has now discovered that if the solutions containing the noble metals together with base metals be made acid before the treatment with phosphide of hydrogen, the base metals are entirely unaffected, and only the noble metal is acted upon. In case the silver is present with the gold the former is at once precipitated as silver chloride and filtered off. With cyanide of gold solutions the reaction is claimed to be, in the first place, $KAuCy_2 + HCl = HAuCy_2 + KCl$, thus leading to the formation of aurocyanic acid. When the latter is subjected to the action of hydrogen phosphide gas, the following reaction is stated to take place: $HAuCy_2PH_3 = AuP + 2HCy + H_2$, a phosphide of gold being precipitated, which, according to analysis, corresponds to the formula $AuP4H_2O$. The hydrocyanic acid formed is stated to be reconverted into potassium cyanide by treatment with caustic potash solution. The precipitate is heated at a low heat, in order to drive off the phosphorus. The gaseous precipitant is produced by putting calcium phosphide into water and collecting the hydrogen phosphide resulting therefrom in a gasholder. The precipitating tank is lined with cloth or canvas of fine mesh, so as to collect the precipitate. As an example of the practical operation of the process it is stated that a cyanide mill solution corresponding to 1 ounce of gold per ton of solution was used, in which 10 ounces of copper and iron had been dissolved. The fine gold recovered was equal to .989 ounce, and no trace of iron or copper could be found in it. The consumption of raw materials was as follows: 6.24 pounds of calcium phosphide per ton of solution and 0.75 pound of hydrochloric acid for the same quantity; 1.5 pounds of caustic potash being used in the regeneration of potassium cyanide. An extremely dilute mill solution of 1 ounce of gold to 25 tons of solution gave a recovery of 96.5 per cent of the gold.

Manufacture of Chlorates and Perchlorates.—M. Couleru, Geneva, Switzerland. Patent 788,631, May 2, 1905. Application filed Nov. 12, 1904.

The process devised by the author depends essentially on the use of salts of certain metals, and especially the chlorides of lead, manganese, etc., in order to neutralize the alkali formed in the electrolysis of a chloride solution for the purpose of producing chlorate. The ensuing reaction is as follows: $2KOH + PbCl_2 = Pb(OH)_2 + 2KCl$. As the liquid resulting from the electrolysis always contains more or less hypochlorites, the latter immediately transform the $Ph(OH)_2$ into peroxide of lead, according to the formula $Pb(OH)_2 + KClO = PbO_2 + KCl + H_2O$. The peroxide is heavy, can be easily separated from the liquor and forms a by-product of the process. The solution supplied to the electrolytic cell consists of a saturated solution of sodium or potassium chloride, with a small proportion of a chromate, such as chromate of sodium. The electrolyzed solution is transferred to another receptacle, and chloride of lead in solid form is added, preferably in the form of a very fine powder, and at a temperature of about 80° C, the action beginning at about 40° and continuing up to 90° or more, but being most effective at 80° C.

Plant for the Electrodeposition of Metals.—A. G. Betts, Troy, N. Y. Patent 789,353, May 9, 1905. Application filed April 11, 1904.

The invention relates to an arrangement of electrodes which permits the use of very large tanks, its object being to reduce either the size or the cost of an electrolytic metal depositing plant, or both; particularly of an electrolytic refinery. The depositing plant is more in the nature of a flooded floor, on which the operations are carried out, than the usual large number of individual tanks, the elimination of a large number of tank sides and ends and the intervening spaces, varying in cost of construction and in space. The invention is stated to be of special advantage in an electrolytic lead refinery, as a metallic lining, which prevents leaks, is expensive, and other methods of making tight tanks require thick sides and bottoms. One arrangement of the electrodes is shown in Fig. 3. A tank 1 of any desired length, with the middle part removed, contains the electrolytic solution from which metal is deposited on the anodes 2 and cathodes 3. The current enters through the positive bus-bar 4, passes to the anodes connected to it, flows through the solution to the cathodes and thence to the conductor 5. From there it goes to the next series of anodes, and continues passing in that manner through the whole solution, until it leaves the tank by the negative bus-bar 6. It is stated that by leaving a double space between the anode of one block of electrodes in multiple and the cathode of the next block of electrodes in multiple, and so doubling the local resistance, there is obtained an equal flow of electric current from both sides of all electrodes, with the exception of an anode at one end and a cathode on the other.

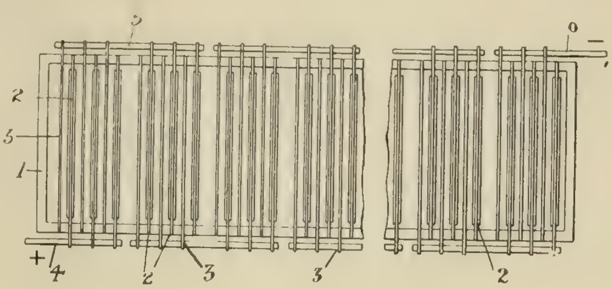


FIG. 3.—ELECTRODE ARRANGEMENT FOR ELECTROLYTIC PLANT.

Process of Electrolytically Refining Copper Alloys. A. G. Betts, Troy, N. Y. Patent 789,523, May 9, 1905. Application filed April 11, 1904.

The first step of the process consists in electrolytically dissolving the copper-nickel alloy, by using it as anode in an acid solution of sulphate of copper, copper being deposited on the cathode. The nickel dissolved from the anodes remains in solution as sulphate, and as the latter therefore increases constantly in nickel, some of it is occasionally withdrawn and replaced by fresh copper sulphate solution. The nickel sulphate solution is freed from copper by well-known electrolytic or chemical methods, and the pure solution is electrolyzed with an anode of spongy lead, such as is used in storage batteries. Nickel is deposited on the cathode, and the lead is partly converted into lead sulphate. A solution of sulphuric acid is electrolyzed, using the spongy lead electrode as cathode and a lead peroxide anode, which is partly converted into sulphate. As a result the solution becomes enriched in sulphuric acid, the lead sulphate of the spongy lead electrode being reconverted into spongy lead for further use in depositing nickel from the solution of its sulphate, and the lead sulphate of the lead peroxide electrode being converted into lead peroxide, as in charging a storage battery. If the lead peroxide electrode is used, after it is thoroughly peroxidized, it is brought into a solution containing sulphuric acid, in which is also suspended an electrode of copper or a nickel copper alloy or fragments, such as may be supported on a conductor. On closing the circuit, electric energy is generated, with the conversion of a part of the lead peroxide into lead sulphate and solution of the copper or copper-nickel anode. It is stated that the electric energy thus produced may be most easily utilized by placing the cell in the refining circuit, when it operates to raise the voltage of the current passing through instead of depressing the voltage, as in the case of most electrolytic tanks. The copper sulphate or nickel-copper sulphate solution produced goes to the electrolytic copper-depositing and copper-nickel dissolving tank. The process thus described is regenerative as regards the sulphuric acid and partly as regards the electric energy used; a plain lead anode, or one of other insoluble material, may, however, be used.

Process of Producing Tartaric Acid and its Salts.—C. Ellis, Boston. Patent 789,269, May 9, 1905. Application filed Dec. 3, 1903.

The process aims at carrying out the electrolytic oxidation of saccharine matter up to the point where tartaric acid is formed and then converting the latter into an insoluble product. In the gradual oxidation by electrolysis of saccharine material various monobasic and dibasic acids are produced, such as gluconic, saccharine, tartaric, lactic, formic and carbonic, The relative quantity of these higher acids is, however, small in the electrolyte on account of this gradual oxidation. In the carrying out of this invention the electrolyte is a 10 per cent solution of grape sugar, made slightly alkaline by the addition of 1 per cent carbonate of potash. Electrolysis is carried out at a temperature of 10° C., the current strength being kept at about 2 amps (current density is not disclosed), while the electromotive force ought not to exceed 10 volts. The electrolyte is gently stirred, and gradually a crystalline deposit of acid potassium tartrate is said to form and to deposit. More alkali is added from time to time, in order to supply the necessary potash, and the electrolysis is continued as long as the tartrate continues to form. It is stated that from 2 to 10 per cent of tartaric acid may thus be obtained. Various salts, such as potassium sulphate, may be added to the electrolyte in order to increase its conductivity. Potassium acetate may be used in place of the potassium carbonate as a source of potash, for the formation of insoluble potassium hydrogen tartrate, in which case the reaction of the electrolyte may be kept acid. The tartaric acid compounds thus obtained may be purified from traces of saccharates by crystallization.

Apparatus for Electrolyzing Liquids.—L. Dion, New York. Patent 789,146, May 9, 1905. Application filed June 10, 1904.

The apparatus, claimed to be especially useful in the treatment of mine waters for the recovery of the metal dissolved therein, consists essentially of an electrolytic chamber, provided with electrodes formed of triangular-shaped bars or tubes. Several series of concentric rows of electrodes are provided, alternately positive and negative. The liquid, after being electrolyzed in this chamber, passes through two filtering chambers, which remove the metals, which are stated to have been separated by the action of the electric current in the electrolytic vessel.

STORAGE BATTERIES.

Gas Separator for Storage Batteries.—T. A. Edison, Orange, N. J. Assignor to Edison Storage Battery Co. Patent 785,297, March 21, 1905. Application filed Aug. 16, 1904.

The invention applies to an improved construction in Mr. Edison's storage battery, for the purpose of separating mechanically entrained globules of the electrolyte and to get over certain difficulties apparent in his previous construction, described in ELECTROCHEMICAL INDUSTRY, Vol. III., p. 000. The valve is made of glass, with a hollow spherical head and a projection for centering it in its seat. It is so proportioned as to float on the solution accumulating about it, but with so little buoyancy that it does not begin to float until the accumulation of liquid is considerable. The excess of liquid then flows back into the cell, which it could not do with the previous construction, as the latter closed too quickly.

Storage Battery Plate.—C. H. Whiting, New York. Patent 789,514. May 9, 1905. Application filed June 25, 1904.

The plate, as shown in Fig. 4, is of rectangular form, and is preferably made of thin metal, cast, stamped or otherwise shaped into form. It has an upper end piece *a*, which furnished a support for the strips *b* which extend from it Each strip has a number of perforations, which serve for applying the active mass, which, when spread on the two faces of each strip, passes through the openings and is thereby held in place. This is furthered by the arrangement of ribs *d* around the end piece *a* and each strip *b*, which, moreover, strengthens the plate and adds to its rigidity when it is made of thin material. The depending strips are fully separated from each other by an intervening slot *e*, which extends from the lower edge of the end piece *a* the full length of the strips. On account of this sub-division of the plate the expansion and contraction of the active material on one strip will not be transmitted to another, but will extend only over each individual strip. The expansion and contraction is, therefore, greatly reduced on such a plate, as compared with a plate in which the body is in one piece. Cross-ties *f* connect the various strips to each other at suitable intervals, thus preventing the plate from curving or bending, so as to destroy the parallelism of the various strips.

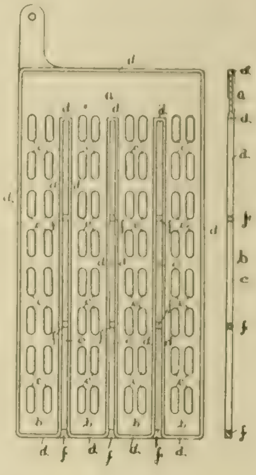


FIG. 4.—BATTERY PLATE.

Protective Sheath or Envelope for Storage Battery Plates.—A. Meygret, Paris, France. Patent 789,557, May 9, 1905. Application filed May 9, 1904.

The inventor intends to prevent the loosening of the active material on the plates by enveloping it in a protective sheath in the nature of an impermeable film. This idea he has developed in several patents, which have been described in ELECTROCHEMICAL AND METALLURGICAL INDUSTRY. In the present process he produces the enveloping sheet by dipping the plate, when complete, with the active material on it, into a bath consisting of castor oil, pyroxyline and flexible collodion. The solution is formed by bringing the pyroxyline, preferably in the form of a jelly, into the solution of castor oil and flexible collodion. The sheath produced by dipping the plates into this solution is stated to be of an elastic nature, and to possess the property of not being attacked by the electrolyte. It has to be provided with a number of preparations or small slits, in order to allow access of the electrolyte to the active mass covered by it.

GALVANIC ELEMENTS.

Galvanic Battery.—D. L. Winters, Chicago. Patent 786,704, April 4, 1905. Application filed Jan. 3, 1905.

The battery is of the zinc-copper oxide type, and comprises a frame for the negative electrode and a holder or clip formed of a plate or bar of resilient material, which holds the electrode securely in its place, so that it can be readily removed and replaced.

MISCELLANEOUS.

Art of Treating and Utilizing Chlorine.—E. C. Paramore, Philadelphia. Patent 786,595, April 4, 1905. Application filed Oct. 5, 1903; renewed Sept. 1, 1904.

The present invention aims at increasing the bleaching properties and destroying the odor of chlorine, by subjecting streams of dry chlorine gas to electric discharges of high tension. The process is carried out by generating chlorine from black oxide of manganese and hydrochloric acid, passing it through a wash bottle containing sulphuric acid and forcing it by means of a pump through the apparatus, where it receives the electric treatment. This apparatus consists of a tube which has walls of dielectric material, which approach each other quite closely at about the middle of the tube, so as to force the gas to flow at that point in a broad and thin stream. Close to each wall, on the outside of the tube, is located an electrode of a current of high tension, which is discharged in sparks through the walls of the dielectric and through the current of gas passing through the channel. Any liquid precipitated in the latter is removed by a trap, and the dry gas passes into a pipe, which carries it down to the bottom of a receptacle filled with the solution to be bleached. Rising above the liquid in this vessel, it is discharged into a pipe by which it re-enters the pump mentioned above, and is again electrified, after which it passes through the same circle of operations. The current employed may be either high tension, direct and intermittent, or alternating of high frequency. It is best derived from the secondary of a Rhumkorff coil. It is claimed that chlorine gas thus treated has greater bleaching power, little or no odor, and bleaches permanently the articles to which it is applied.

Notes on Electrochemistry and Electrometallurgy in Great Britain.

(*From our Special Correspondent.*)

THE PRESIDENTIAL ADDRESS AT THE INSTITUTION OF MINING AND METALLURGY.

Mr. William Freecheville's presidential address to this Institution—a copy of which has been kindly sent to me—is as capable of division into clearly marked sections as any stratified portion of the earth's crust. The first portion is a review of mining development within the British Empire, which deals with the financial assets of the Empire so far as metallic deposits are concerned. The second portion is devoted to some remarks on modern mining methods, in which the working of deep alluvial ground by pump dredges, and the design of compound centrifugal pumps are discussed. The third portion consists of a consideration of the giving of information to shareholders. Mr. Freecheville's opinions should serve to earn popularity among mining investors as distinct from speculators. "It may be stated, broadly, that a manager of a mine, in his periodical letters, gives free expression to his hopes and fears, and that any important change for better or for worse is, as a rule, foreshadowed in them a long time before it happens. Why should not the shareholders have this information, which is obtained at their expense and would be of much value to them to form a just estimate of the value of their property?" * * * "I believe it would be found that the shareholder is nervous and timorous largely because he is kept so much in the dark, and that if he felt that all known information on the subject of his property was at his disposal, and that he could read, if he wished to, the manager's letters at the company's office, he would be much more stolid than many directors imagine."

The remainder of the address deals with the aims and objects of the Institution, emphasis being laid on the educative value of the papers deliberated no less than on the stand which the council of the Institution had taken with regard to defining what is actually comprised by the term "ore in sight," and the active influence it was bringing to bear on the question of mining education.

THE FARADAY SOCIETY.

The meeting held on April 4, was a "three-paper" evening, the first discussed being that of Mr. A. H. Hiorn's, entitled "Alloys of Copper and Bismuth." Professor Huntingdon, who was in the chair, opened the discussion by pointing out that previous observers had not recorded the eutectic point of pure copper and copper bismuth solution at 1,020° C., shown at *D* on the author's cooling curve. He also suggested that Mr. Hiorn's study would have been more complete had there been an inclusion of the results of an examination of chilled alloys. Dr. Desch concurred in this view of Professor Huntingdon's and spoke of the extreme complexity of the problems of crystallization. After some remarks by Professor E. Wilson concerning the effect of bismuth on the conductivity of copper, Mr. Hiorn in reply stated various experiments justifying his belief in the existence of the eutectic at 1,020° C.

Mr. E. Kilburn Scott's paper, which was printed in ELECTROCHEMICAL AND METALLURGICAL INDUSTRY, April issue, p. 140, and to which the author gave a brief introduction at the meeting of January 31, was then discussed. It provoked a lengthy discussion in which different metallurgists spoke of their favorite furnace linings. Mr. Cremer commended sodium silicate as a binding material for use with carborundum. Mr. Morrison favored the use of tar, especially for high temperatures. Mr. Gaster spoke of siloxicon used with either binding material. He deprecated its use as a wash, recommending instead hard bricks made of this substance. Mention was also made of the fact by Mr. Morrison of the lack of exact data in the paper as to the actual temperatures employed, it being scarcely scientific to speak "highest possible" or "very high" temperature. Professor Dains sent a valuable communication to the discussion in the shape of some detailed descriptions and analyses of the various magnesite deposits at present available. Mr. Hutton complained of the absence of information in the paper respecting crystallized carborundum. Magnesite can be very easily fused in the electric furnace when it becomes fluid. Its conductivity, and that of the allied materials, was a most important point in regard to which very little work had been done. In his opinion the only satisfactory furnace lining was the material which the furnace itself produced.

The discussion on the first part of the paper by Mr. R. S. Hutton and Mr. W. H. Patterson, entitled "Electrically Heated Carbon Tube Furnaces," was unfortunately very brief—largely perhaps because the subject of tube furnaces is relatively a new one. Gratitude was expressed to the authors for their suggestion of the use of the water jacket and copper ends to the tubes. (The paper will be printed in full in one of our next issues.)

THE ALUMINIUM OUTLOOK IN THE UNITED KINGDOM.

At a meeting of the shareholders of the British Aluminium Company in April, Mr. J. D. Bonner said they had a surplus available for distribution as dividends on all classes of shares due to increased sales and reduction in cost of production. The profit for the year exceeded £55,400, compared with £34,143 in 1903, and he predicted that the profit for the current year would show an even greater increase. The demand for aluminium was growing, and necessitated the completion of the Loch Leven Power Works, of which the plans had been prepared. The company had a large and increasing business, and there was every indication that they would have a prosperous future.

THE ANNUAL MEETING OF THE UNITED ALKALI COMPANY, LIMITED.

Of the large English chemical manufacturing companies, the United Alkali Company is perhaps the second largest. It possesses works on the Tyne as well as in Lancashire and Cheshire. The annual meeting of the shareholders was held on March 24 last, and the speech of Mr. John Brock, the chairman, was interesting on account of its review of the commercial prospects of the industry generally. Mr. Brock said, "the deliveries of their goods had been somewhat lower in 1904 than in 1903, due mainly to the increase of manufacture in other countries, and to strikes and short time worked in various trades using the goods. Prices had remained in most cases at the low level of 1903, and for several articles had been even lower. In the metallurgical branch of their business, both prices and demand had favored them, and they looked forward to the future with confidence. They would have to spend money in the improvement of machinery and development in linking up the two properties in Spain by a tramway, by improving the means of the transit of the ore to the port of shipment, etc. The action brought against them by the Acetylene Company for alleged infringement of patents had been finally decided in their favor by the House of Lords. The action had involved them in heavy expenses, which they had written off out of revenue. The gross profit was [illegible] or £[illegible] more than in 1903, and the net profit (after payment of interest) £219,019. Adding £80,047 from previous account and deducting £15,000 transferred to debenture redemption fund, there remained £284,066. The dividend on the 7 per cent. preference shares was being paid, £50,000 was transferred to depreciation, and £43,762 carried forward.

The Capitalistic Side of British Electrochemistry and Metallurgy

About the third week in February the writer purchased, as is his annual custom, the 1905 issue of Garcke's *Manual of Electrical Undertakings*, a work which gives the latest return of each electricity supply and each electric traction undertaking in the United Kingdom, together with returns as to capital invested in and dividends in these and in manufacturing concerns. As the *Manual* is divided into the following broad divisions: (1) Light power and traction; (2) telegraph and telephone; (3) manufacturing and miscellaneous, and (4) directory, and as section 3 is not further sub-divided into its respective sub-sections, the writer proceded to note down the names and capitals invested in concerns of electrochemical and metallurgical interest. These may be considered under four groups:

(a) Electrolytic alkali and bleaching powder, makers of.
(b) Batteries, primary and secondary, makers of.
(c) Metallurgical undertakings.
(d) Miscellaneous.

Treating them in the order given, about £1,066,000 sterling is invested in the electrolytic alkali industry in four undertakings. Almost the whole of the capital is accounted for by two firms, the Castner-Kellner Alkali Company, Ltd., and the Electrolytic Alkali Company, Ltd. The former, whose works are at Runcorn, Cheshire, have a share and debenture capital amounting to £700,000. Their dividend for the last financial year amounted to 4 per cent, as compared with 8 per cent for 1898-99, 5 per cent for 1900, and 6 per cent in 1901, 1902 and 1903. The Electrolytic Alkali Co., which works the Hargreaves-Bird process, has not been so fortunate. No dividends have yet been paid on 202,000 ordinary shares of £1 each, nor on the 7 per cent preference shares.

The second group embraces seventeen limited liability undertakings with an aggregate capital of £603,289. Of these the Chloride Electrical Storage Co., Ltd., has a total capital of £115,250. Last year 6 per cent was paid on £62,000 of preference shares, and 8 per cent on £33,250 of ordinary shares. The Electric Power Storage Co., Ltd., has a capital of £118,158. The dividends on the ordinary shares were 5 per cent in each year from 1892 to 1902, and 6 per cent in 1903 and 1904. The only other company in the group which calls for specific comment is the "D. P." Battery Co., Ltd., which, with a share capital of £10,000 and £20,000 debentures, paid a dividend of 12 per cent in 1901, 15 per cent in 1902, and 20 per cent in 1903 and 1904.

In the third group nine undertakings are embraced. These have an aggregate capital of £2,003,173. Over one-third of this sum, or £700,000 is invested in the British Aluminium Co., £400,000 of which represents share capital, and £300,000 debenture capital bearing interest at 5 per cent. An issue of a further £300,000 worth of 5½ per cent debentures has been authorized. Another undertaking with a large capital is the Leeds Copper Works, Limited. The issued capital amounts to £698,400. For the year ending December 31, 1903, which were presented on June 27, 1904, a deficit amounting to £13,885 was incurred, as against a deficit of £5,579 for the preceding year.

The fourth group of miscellaneous undertakings includes makers of electrolytic meters, ozonizers and firms whose work is associated with electrochemistry, such as the Electro-Peat Coal Company, Limited, whose prospectus was so severely criticised last year. The total capital invested in the nine companies of this group is £439,191.

Inasmuch as Mr. Garcke returns the actual aggregate subscribed capital invested in shares, debentures, and loans of the 1225 electrical undertakings particularized in the *Manual* as amounting to about £277,000,000, it will be seen that electrochemical and electrometallurgical investments are not largely patronized by the British investor.

Market Quotations.

The nominal quotations in the chemical trade are still stable, but a slight rise in price may be expected if the cotton mills of Lancashire continue as fully occupied as at present. There are attempts being made in several parts of the country to introduce electrolytically prepared bleaching solutions for textile and paper bleaching as a substitute for chloride of lime. The success of these processes will depend on local costs for power and salt, and their immediate area of commercial possibility would therefore seem to be limited to the vicinity of the salt-producing districts.

Shellac is hardening again in price, being quoted at from 150 to 165 shillings per cwt. Copper sulphate is slightly lower at £21.17.6 per ton, but cyanides are unchanged. Para rubber is in good demand, the prices varying largely with quality; Brazilian varieties fetch 5s. 6d., while the cleaner rubber from Ceylon and the Straits Settlements is in the neighborhood of 6s. 3d.

Platinum is quoted at £4.2.6 per oz. Cleveland pig iron, after remaining under £2.10.0 until the middle of the month, rose to £2.13.0 by May 2. Copper continued to fall, closing at £65.15.0. In sympathy with this decline rod and sheet copper have also fallen slightly. Tin was the subject of a boom in price about the middle of the month, largely owing to market speculations. The high price of £145 per ton was reached on April 12; an abrupt fall of £5 followed in two days. April ended with the price at £138.10.0, an increase of £1 since March 31. Lead rose slightly during the brief days of the tin boom, afterwards receding to £12.17.0 for ingot, and £14.5.0 for sheet. Zinc and quicksilver are unchanged.

London, May 6.

SYNOPSIS OF PERIODICAL LITERATURE.

A Summary of Articles Appearing in American and Foreign Periodicals.

Industrial Applications.

Electric Furnace Work.—The present status of electric furnace working is the subject of a recent paper by C. F. Burgess, published in the April issue of the *Jour. Western Soc'y of Engineers*. The author gives an interesting and enthusiastic [illegible] of what already has been accomplished with the electric furnace, and points out that "while it is only in process of emerging from the laboratory stage it has already reached the dignity of an agent whose services can be measured in terms of that sordid but universal unit—the dollar." He analyzes the inherent advantages of the electric furnace, gives a classification of the different furnace types, and then briefly discusses some details of construction. Concerning the protection from the air of the portions of electrodes projecting outside of the furnace, he mentions encasing in a thin iron covering or artificial water-cooling, and says that in some of his "experiments at the University of Wisconsin it has been found that certain fused substances could be formed on the surface, making a layer of glass, completely excluding the air and protective even at a white heat." The author considers that the ultimate industrial success of the electric furnace depends on continuous operation. He then points out the high efficiency of the electric furnace compared with other furnaces. He emphasizes its possibilities in zinc metallurgy. "The present methods of extracting zinc are notoriously inefficient. Not only is the 80 per cent efficiency of the electric furnace, as compared with the 3 or 4 per cent efficiency of the ordinary zinc furnaces capable of resulting in an economy, but various other advantages may be attained as well. The cost of zinc production under present conditions involves incomplete extraction from the ore, absorption of zinc by the walls of the retorts, rapid destruction of retorts, and losses in the condensation of zinc. Electric furnace methods present opportunities for materially remedying these defects. In laboratory trials, made at the University of Wisconsin, it has been found possible to completely extract the zinc, so that none of it remains in the charge or is retained in the walls of the furnace. No practical form of furnace for this purpose has been evolved, however, the principal difficulty being in effecting continuous operation, a requirement which is almost imperative."

Manufacture of Phosphorus.—It is well known that at the Niagara works of the Oldbury Chemical Co. an electric furnace process is now used for the manufacture of phosphorus. An account of a laboratory investigation on the manufacture of phosphorus is given in two papers by W. Hempel in *Zeit. f. Angew. Chemie*, 1905, Vol. 18, pp. 132 and 401, and abstracted in the *Jour. Soc'y Chem. Ind.*, February 28 and April 15. The author has investigated the Pelletier process (conversion of tricalcium phosphate into monocalcium phosphate by means of sulphuric acid and reduction of the product by charcoal). In his experiments he used an electric tube furnace. Woehler long ago proposed to reduce a mixture of tricalcium phosphate and silica with charcoal, according to the equation

$$Ca_3\,P_2\,O_8 + 3SiO_2 + 5C = 3Ca\,SiO_3 + 5CO + 2P.$$

But the process never succeeded, owing to the difficulty of attaining the requisite temperature or of finding vessels to withstand it. Electric heating, however, has lessened or removed these difficulties, and an experiment was made on this process. Gas began to be evolved at 700° C., and became inflammable at 1000° C.; phosphorus was detected at 1150° C., began to distil in quantity at 1200° C., and the distillation ended at 1450° C. The yield was 92 per cent, and very little was in the "killed" condition. Electric heating has lately been adopted in German works. The retorts are sheet-iron cylinders, lined with fireclay, through the lower part of which carbon electrodes are introduced. The materials are fed in continuously, and the residual slag continuously removed. Perfect dryness of the materials is an important factor in increasing the yield. In the second paper the author replies to some criticisms of Neumann. He has used an electric resistance furnace, not a carbon arc, because the former is more certainly regulated. But he has not suggested that this should be used on a large scale. On the large scale internal arc-heating will be more economical than externally fired retorts. Further experiments in his laboratory have shown that Woehler's process, using arc-heating, is by far the best method of preparing phosphorus.

Yellow Arsenic by Means of the Electric Arc.—A paper, by A Stock and W. Siebert, published in the *Berichte*, 1905, Vol. 38, p. 966, and abstracted in the *Jour. Soc'y Chem. Ind.*, states that yellow arsenic can be obtained by the action of an electric arc between arsenic electrodes under carbon bisulphide. The arsenic volatilizes and is condensed and dissolved by the carbon bisulphide. The anode need not necessarily be of arsenic, but might be a carbon rod. Fused arsenic becomes very brittle in cooling; it is, therefore, difficult to make the anode from the pure metal. The authors employed an alloy of arsenic and antimony. In the arc the antimony is reduced to a very fine black powder, absolutely insoluble in carbon bisulphide. The operation was carried out in a beaker containing 500 cc. of the solvent, the whole placed in a larger vessel filled with water and ice. Using a current of 12 amps., 5 grams of the alloy were decomposed in 5 minutes, when the experiment was interrupted to allow the solution to cool down. Working for a 1 per cent solution gives the best results. The solution, separated from the water and filtered, is colorless, but turns yellow after some time. At — 80° C. all the arsenic crystallizes out from the solution. By evaporation of the solution the black modification of arsenic, almost free from sulphur, is obtained.

Electrolytic Treatment of Electrolytic Slime.—The interesting article of Mr. Anson G. Betts, which appeared on page 141 of our April issue, has been reprinted in form of a pamphlet with some additions. The latter are here noted. To the statement of the particular reactions of the finely divided metals as they exist in slimes, the following additions are made: Selenium is not dissolved from slime by ferric sulphate solution, nor by caustic soda and sulphur, or equivalent solutions; it easily passes into the slag in melting with nitre; in electrolytic parting, at least in the process described on page 145, it remains with the gold. Tellurium is dissolved from slime by the action of ferric sulphate solution, and is reprecipitated by metallic copper, analogously to silver. Copper slimes containing sulphur yield tellurium, but not silver, to ferric sulphate. The reaction holding silver back was given on page 143, top of first column. In melting, even with nitre, tellurium goes mostly into the dore bullion, and in electrolytic parting remains with the gold, from which it can be extracted with hot sulphuric acid. The following analytical data are given in an appendix:

Analysis of electrolytic antimony from trail slime—Ag nil, Pb nil, Bi nil, Cu 0.07 per cent, As 0.47, Sb 99.52. (The arsenic was introduced into the antimony for the most part from the HF used, which almost always contains arsenic and antimony.) Anaylsis of dore bullion from trail slime—Ag 78.94 per cent, Pb 17.56, Au 2.08, Cu 0.81, Sb 0.47, As nil. Treatment of copper slime No. 15 in Table I. (page 142). Extracted by caustic soda, 95 per cent of the antimony, no selenium, and 0.043 tellurium. Analysis of dore bullion—Ag 86.55 per cent, Bi 5.37, Cu 5.99, Au 1.62, Te 0.16, Se trace, Pb nil, S nil, Sb trace. Analysis of upper slag from melting to dore—Ag 0.11 per cent. Analysis of lower slag from melting to dore—Ag 10.76 per cent, Cu 8.79, Sb 1.48, Bi 1.79, SiO 20.40, Fe 17.81, Se 2.25, Te 0.84, Pb nil. Electrodeposited silver, tested for Cu, Bi, Pb, Sb, Se, Te Au and S, none present. Fresh silver electrolyte contains about 7.5 per cent Ag, 15 per cent $CH_3\,SO_3$; exhausted silver electrolyte contains about 1.5 per cent Ag, 15 per cent $CH_3\,SO_3$.

Detinning Tin Scrap.—In the April issue of *Elektrochem. Zeit.*, H. Mennicke continues his article on the winning of tin from tinned iron scrap. He gives brief data on quite a number of processes which have been proposed in recent years, mostly from patent specifications; all of them are considered to be of no value for industrial purposes. He then deals briefly with various patents for details of construction of apparatus, and gives references to articles published on the subject.

Electrolytic Manufacture of Chlorates and Perchlorates.—A French patent of P. Corbin (339,251, Dec. 31, 1903) is mentioned in the *Jour. Soc'y Chem. Ind.*, April 15. The invention relates to the manufacture of chlorates by the electrolysis of neutral alkali chlorides. Resin is added in small quantities in the presence of compounds of metals of the alkaline earths, and, in addition, dilute hydrochloric acid is introduced into the either during, or at intervals in, the electrolytic treatment of the same solution.

Ozone for Water Sterilization at Paris.—In the London *Elec. Eng.* of April 14 E. Guarini describes the water sterilization works recently started at the municipal water works of Paris, at Saint-Maur. The plant treats up to 150 cubic meters of Marne water per hour, taken either from the slow sand filters of the city of Paris or from the river and filtered, without adding any coagulant, by two rapid-pressure filters filled with crushed flint. The ozonizer system used is that of Count de Frise. The results are stated to show that a plant of even much greater size can be worked by two or three men, and does not want more than about 129 hp. in all, pumping included, for the treatment of 1000 cubic meters per hour. No solid dielectric is used in the de Frise ozonizer; they are artificially cooled. Each cell is made of a horizontal brass trough, with a half-cylindrical cross-section fitted with a plate glass cover and a cast iron water jacket. The trough is earthed, and forms one of the poles. Across the trough at regular distances, half-discs of brass, with serrated circular edges and of 60 mm. less diameter than the trough, are suspended from the glass lid; they are connected to the source of high-tension currents through liquid resistances, each half-disc being connected to one liquid resistance. These resistances perform the part of regulators, preventing the tension of the portion of current allowed to each semi-circular pole from rising above the limit at which sparks or arcs would be produced. Silent discharges are produced between the sharp points of the semi-circular high-tension poles and the inner surface of the earth troughs, as shown in Fig. 1. The current of air circulating along the tube passes through the succession of half-annular discharges, which transform part of its oxygen into ozone. After its passage through each discharge the air heated by electrification is partially cooled down by the cool surface of the trough. This cooling is periodically completed by means of surface condensers placed between the elements of each line of ozonizers. The ozonizers are placed in a dark room, so that the superintendent may judge from the blue-violet color of the flames whether the apparatus is working in good condition. The ozonizers are worked with tensions higher than 2000 volts. The ozonized air is used over again and again, the inlet of the ozonizer being connected with the outlet of the sterilizers, arrangements being also provided to free the air from moisture and to make up, by fresh air, the oxygen which has been consumed in the shape of ozone.

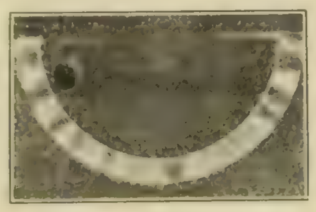


FIG. 1.—OZONIZING DISCHARGE.

Two of the sterilizers erected at Saint-Maur are cast-iron vertical cylinders enamelled inside. They are built up in parts, 50 cm. high, between the flanges of which are fitted horizontal celluloid diaphragms perforated with a great number of small holes of 0.7 mm. diameter. Air and water ascend together from the bottom to the top of the sterilizer, and are intimately mixed at the passage of each finely perforated diaphragm. In a third sterilizer the water is injected in the shape of a fine spray into the mass of ozonized air. The spray collects in the lower part of the column as water, into which the ozonized air is forced through the aperatures of a perforated diaphragm, which makes it bubble through the water before reaching the [illegible]

Another type of ozonizer is described by A. Breydel in *L'Eclairage Electrique*, April 29. This type seems to be adapted [illegible]lly for experimental and therapeutic work. Special [illegible] for artificial cooling and drying are provided. To pre[illegible]ks, the two sets of aluminium plates between which the silent discharge occurs, are separated by glass plates. Every aluminium plate is covered on both sides with glass plates of larger surface.

Production of Ozone.—In connection with the various apparatus for the generation of ozone from air for the subsequent utilization for sterilization of water, bleaching and other purposes, an account of an investigation by Wommelsdorf is interesting, which is given in *Annalen der Physik*, No. 15, 1904, and abstracted in Lond. *Elec.*, Feb. 24. The author studied the "noxious charges" which impair the efficiency of influence machines, and which are, therefore, avoided as much as possible by makers of such machines. These become a minimum at a certain distance between the plates, and that distance, therefore, brings about the maximum efficiency. At a certain smaller distance between the plates the loss by the "noxious charges" becomes very large. The external effect of the machine approaches, in fact, to zero, and the electrical work is all done inside the machine. When that occurs an enormous quantity of ozone is produced by the machine. This indicates a possibility of producing ozone in a much more direct manner than hitherto has been possible. An influence machine can be made into a combined generator, transformer and ozone tube, and it might even be made to act as its own aspirator. It could be enclosed, and the loss of ozone would be reduced to a minimum. It should be possible to generate ozone in a very economic manner.

Calcium Carbide.—The manufacture and consumption of calcium carbide in the world is discussed by L. Marmier in the March issue of *L'Electrochemie.* The consumption during 1904 is estimated as 90,000 tons. Sixty-eight plants are actually in operation, which could produce about 150,000 tons, but they are not working to their full capacity. There are said to be in operation 17 plants in Germany and Switzerland, 14 in France, 11 in United States and Canada, 8 in Austria, 8 in Sweden and Norway, 6 in Italy, 2 in Spain, 1 in England and 1 in Argentine. Germany manufactures 12,000 tons, and consumes 18,000 tons, of which 6000 are imported from Sweden and Norway, Switzerland and Italy. France and its colonies consume about 14,000 tons, made in France. The United States and Canada consume 16,000 tons, and produce 18,000 tons, of which 2000 tons are exported to Mexico, Cuba and South America, the latter producing 1000 tons and consuming 2500. Italy produces more than 20,000 tons, and consumes more than 17,000 tons. Spain produces and consumes 5000 to 6000 tons. England consumes 3500 tons and produces 1000 to 1500 tons. The balance is imported from Sweden and Norway. These countries produce 11,000 tons and consume only 1500 tons. It is estimated that the consumption of calcium carbide in the world will increase by 20 to 25 per cent during 1905.

Calcium Carbide as an Explosive in Mining.—The *American Journal of Science*, March, notices an article by Guedras, from *Comptes Rendus*, Vol. 139, p. 1225, on a method of utilizing the explosive force of acetylene for mining purposes. A sheet-iron cylinder is used as a cartridge. At its bottom are placed about 50 gr. of granulated calcium carbide, and above this in a seperate compartment is a sufficient amount of water to react with the carbide. There is also an air chamber containing an electric fuse. The cartridge is placed in the hole to be exploded, the latter is closed with a wooden plug, and an iron rod attached to the cartridge for the purpose of piercing the water compartment is struck, thus liberating the acetylene. After this has been disengaged for about five minutes, the mixture of air and acetylene is exploded. The explosion causes the rocks to fly about much less than would be expected, but they are thoroughly broken up.

Electrolytic Rectifier Versus Motor Generator.—In the London *Electric Review* of Feb. 10, H. Boot gives an account of some tests conducted with an electrolytic rectifier for charging the accumulators of a large electric car. The amperes required were 50, and the voltage 118 when the cells were fully charged. The total cost of a motor generator plant would have been $775, while the cost of a Nodon electrolytic rectifier was only $340. The latter was adopted and proved very satisfactory in practice. The efficiency was found to be about 65 per cent and does not decrease during use. The rectifier has been in use since October, 1904, and shows no signs of deterioration. There is some evaporation taking place which requires adding distilled water. Two letters with notes on difficulties with electrolytic rectifiers are printed in the same journal of Feb. 17, to which H. Snowdon replies in the issue of Feb. 24, stating that he experienced no troubles. The successful application of an electrolytic rectifier for changing three-phase currents into direct current for use in a telephone station is described by R. Stosberg in *Elektrotech. Zeitschr.*, February 23.

Analysis of Refined Copper.—The principal methods now employed in large technical laboratories for the complete analysis of refined copper are the subject of a concise review by Mr. George L. Heath, of the Calumet & Hecla Smelting Works, published in the March issue of the *Jour. Amer. Chem. Soc'y.* The present tendency is to take separate samples for the estimation of each group of foreign elements by some special method of isolation. Analytical results are usually carried out to 0.0001 per cent. The author first gives exact rules for sampling. The following tests are made upon the finished metal: (1) Mechanical, (2) electrical, (3) chemical. Concerning the first two reference is made to a paper in Proc. Lake Superior Mining Institute, Vol. 7, p. 68 (1901). The author then deals with the chemical tests, assay of gold and silver, electrolytic estimation of pure metallic copper in commercial metal, estimation of oxygen, a rapid and exact method for the estimation of traces of sulphur in refined copper, various methods for determining arsenic, antimony and tin, selenium and tellurium, lead, iron, zinc, nickel and cobalt; bismuth. The author mentions incidentally a design for a rotating anode (Fig. 2), which has been found satisfactory for use with platinum cylinders as cathode. A straight, stiff wire is bent back and forth at one end. A disc, the size of a quarter of a dollar, is cut from a sheet and is slit on radial lines to form ten propeller blades, and a small hole bored in the center, through which the bent end of the wire is passed and clamped by pressure. This is revolved to generate downward pressure, which takes the weight off the spindle and greatly lessens the friction.

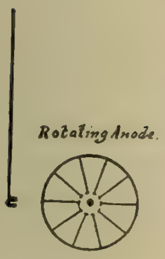


FIG. 2.
ELECTROLYTIC ANALYSIS.

Organic Chemistry.—J. Moeller's serial on electrochemical reactions in organic chemistry is continued in the April issue of *Elektrochem. Zeit.*, while the issue of April 1 of *Chem. Zeitschrift* brings the conclusion of an article by J. Mueller on the application of electrochemical reactions in organic chemistry on a commercial scale. His general conclusions are that primary reactions and, of the secondary ones, the substitution reactions, are at present scarcely of any value for industrial purposes of organic chemistry. There are a few technically useful cases of electrochemical oxidation of organic compounds, and there is no reason to expect much progress in this field in the near future. It is, however, quite different with electrochemical reduction, and the reduction of aromatic nitro compounds seems to represent a very promising field.

BATTERIES.

Nickel-Iron Alkaline Storage Battery and Accumulator Traction.—Jungner's German patents for this battery—which is known in this country as the Edison battery, while abroad it is sometimes called the Jungner-Edison battery—were acquired, as our readers know, by the Kölner accumulatoren Werke Gottfried Hazen in Cologne. The director of this company, Dr. E. Sieg, recently delivered an interesting lecture on the development of this cell by his company, which is published in *Elektrotechnische Zeitschrift,* March 30. The first discussed in a general way the requirements of automobile batteries. The power required for running an automobile is determined by two factors: first, the friction losses of all kinds, together with the efficiency of the gearing devices, etc., and, second, the air resistance. The power for overcoming the friction increases proportionally with load and speed, and amounts to about 80 watthours per ton-kilometer on average roads, so that if the automobile voltage is 80, the current consumption in amperes is approximately equal to the weight of the loaded vehicle in (metric) tons, multiplied by the speed in kilometers per hour. This item of power is of decisive importance for all speeds up to 15 kilometers per hour, so that the power for overcoming the air resistance is negligible except on stormy days. For higher speeds, however, the power consumed by the air resistance cannot be neglected, since it increases proportionally to the third power of the difference of the velocities of air and vehicle. For instance, for a light vehicle for two passengers, this item of power, running at 30 km. per hour, may amount to one-third of the total power; it depends, of course, greatly on the direction and the pressure of the wind. For practical purposes it is of special importance to know the capacity per unit of weight of the battery, for which the cost of operation is a minimum. The author has made such a calculation, with the aid of calculus under certain simplifying assumptions. He assumes a total weight of 250 kg. of vehicle and passengers; a run of 90 km. with one charge; a cost of maintenance of vehicle, excluding the battery, of 0.5 cent per ton-kilometer; and a cost of 5 cents per kilowatt-hours. Under these conditions he finds the following values, *y*, of the most favorable capacity in watt-hours per kilogram, corresponding to the values, *v*, of the speed of the vehicle in kilometers per hour:

v = 10	y = 30.
15	31.2
20	33.3
25	36.
35	37.8

In accordance with these results it has been the endeavor of storage battery manufacturers to increase the capacity per unit of weight, at the expense of life. The company of the author will soon place a lead cell on the market which yields 34 watt-hours per kilogram. He remarks that this type will not be more expensive per ampere-hour than the heavy types, but will be cheaper. The author thinks that the lead cell has not yet reached the end of its devolpment for automobile purposes, "although even now in every case where the electric automobile may be used (*i. e.*, on roads of at least average quality for speeds not above 30 km. per hour and for runs not above 100 km. for one charge) it is cheaper than any other vehicle, especially cheaper than a horse-driven carriage."

The author then gives a long historical review of the development of the alkaline accumulator, and gives some theoretical considerations of the possibilities of the nickel-iron cell. For 1000 ampere-hours the lead accumulator requires theoretically (according to Faraday's law) 8.4 kg. active mass, the nickel-iron cell 3.4 kg. active mass. However, in the latter case the active mass must be mixed with some conducting material (graphite); this makes 5.9 kg. instead of 3.4 kg. Moreover, the average voltage of the lead cell is 2, that of the alkaline cell not more than 1.25. Hence 1 kg. active mass gives 2.38 watt-hours in the lead cell and 2.11 watt-hours in the alkaline cell. Further, in light lead cells the ratio of active mass to total weight of plates is 70 per cent, in the Edison cell 50 per cent. The author thus arrives at 166.5 watt-hours for the lead cell against 114 watt-hours for the alkaline cell for equal weights. (However, in these figures the difference in the weights of the electrolyte in both cases is neglected.)

The author then describes the troubles which his company had with developing the Jungner cell after they had bought the patents. These troubles had to do especially with the preparation of the active materials and the construction of the plates. The difficulties encountered seem to have been so great that from the experiments they became convinced that by their method of construction it would be impossible to develop an accumulator equal to the lead cell with respect to lightness and far less one which would be superior. They therefore restricted their endeavors to the development of an accumulator which would be superior to the lead cell with respect to durability in spite of bad treatment. This cell gives only 10 to 12 watt-hours per kilogram (against 24 watt-hours for the Edison cell). Two modifications of Edison's construction are mentioned: They are sheet nickel instead of nickel-plated sheet steel (since the latter is said to be attacked in time), and they use nickel plates instead of graphite as addition to the active mass.

Edison Accumulator.—In *L'Eclairage Elec.*, of April 29, M. U. Schoop and C. Liagre discuss the question of the best electrolyte for a nickel iron alkaline accumulator. Potassium hydroxide has a maximum conductivity at 29 per cent, sodium hydroxide at 15 per cent, the maximum being greater for KOH than for NaOH. The reasons are given why it is best to use a 20 per cent solution of KOH, as is done by Edison. The authors give some practical rules how to handle alkaline solutions in experimental work in the laboratory. They finally mention the proposal of Michalowsky to use $Al_2O_4K_2$ as electrolyte. In this case it is possible to use zinc, since it is insoluble in this electrolyte. The reaction during charge is given as follows:

$$Al_2O_4K_2 + 2H_2O = 2KOH + Al_2O_3 + 2H + O$$

The two H atoms and the O atom are, of course, not set free, but represent the reduction and oxidation of the two electrodes. The KOH appears at the cathode and Al_2O_3 at the anode. If the electrodes are near enough to each other these two products mix, and $Al_2O_4K_2$ is regenerated. However, the low conductivity of this electrolyte is considered by the authors to be prohibitive for practical use.

Light Storage Batteries.—*L'Industrie Electrique* of Feb. 10, contains a note on a new lead accumulator called "E. I. t." Both plates are pasted, and it is claimed that the novelty rests in the use of special lead oxides. The useful average e. m. f. is stated to be 5 per cent higher than that of the ordinary lead accumulators for the same quantity of active material, and the capacity is said to be 33 to 50 per cent higher than in accumulators in which the ordinary pastes are used. According to tests made in Paris the battery has an energy per weight of 41.2 watt-hours per kilogram with an initial e. m. f. of 2.29 volts and an initial useful potential difference of 2.07 volts, the discharge being continued down to 1.7 volt at an average rate of 5.5 watts per kilogram. The density was 1.32 (35° Baume) at the beginning of the discharge, and 1.16 (20° Baume) at the end. This unusually high density of the electrolyte might explain the result of the tests, and it is very doubtful whether an accumulator with such concentrated acid has any long life. Reference is made to the use of a concentrated solution by Krieger in 1901 on an automobile. He covered 307 km. without recharging with an initial e. m. f. of 21.5 volts, and with an energy of 38 watt-hours per kg. His sulphuric acid had an initial density of 1.285 (32° Baume) which is less than that of the electrolyte E. I. t. Nothing was heard later of his battery. To this later note, Krieger replies in the same journal of Feb. 25. He says his battery was not put out of service by this single experiment. He adds, however, that he would not employ every day such a battery. If very great care is taken, he claims, it is possible to get 50 or 60 charges and discharges with such a battery.

The *Elektrotechn. Zeitsch.* of March 2, gives a review of the electrical exhibits on the International Automobile Exposition in Berlin, the author being C. von Groddeck. In reviewing the accumulator automobiles of the Cologne Accumulator Works of Gottfried Hagen, it is said that the older lead accumulators for automobiles of this firm give watt-hours per kg. The weight of an automobile for four passengers is 1500 kg.; the weight of passengers 400 kg., and the weight of the battery of 1700 watt-hours is 850 kg., hence the total weight 2.75 tons, allowing a single run of 80 km. without recharging on the basis of a consumption of 75 watt-hours per ton-km. on a level road. The newer lead accumulators of the same firm give 29 watt-hours per kg. For the same automobile as before one gets a single run of 120 km. without recharging. On the other hand, if one is satisfied with a single run of 80 km., the total weight of the automobile may be reduced by 350 kg. With the very latest type of this firm it is said that 34 watt-hours are obtained per kg. The lower weight of the newer types is obtained by the use of a very thin plate which is stated to be protected against buckling by mechanical construction. The increase of capacity is obtained at the expense of life. While the older batteries give 150 discharges before they have to be renewed, the newer ones give only 100 discharges. It is stated, however, that this company has found by experiments extending over several years that this shorter life is not prohibitive since the increased cost of maintenance of the battery of the newer type is more than counterbalanced by the smaller cost of maintenance of the rubber tires. Some details are given on mechanical construction of automobiles and on benzine dynamo electromobiles. Gottfried Hagen had also exhibited a battery of the Jungner system (competing with the Edison battery in this country); but it is said that the Jungner battery is still in the experimental stage and has not been so far developed that it can be used in practice. The firm states that the batteries are still more expensive than lead accumulators, and that not the same length of run is obtained with them as with lead accumulators. Some electromobiles exhibited by Ziegenberg were equipped with lead-zinc accumulators which "are claimed to give 50 to 70 watt-hours per kg., and to have an unlimited life, while the time of charging is only one-half hour." No further details are given.

EXPERIMENTAL AND THEORETICAL.

Electrostenolysis.—Electrostenolysis is the name usually applied to the deposition of certain metals in capillary tubes during the passage of the current. The deposition occurs, as a rule, only under circumstances capable of producing considerable endosmose, namely, when the fall of the potential in the capillary is great. After the capillary has been lined or plugged with deposited metal its ends act as minute secondary electrodes within the liquid, giving rise to secondary cathodic and anodic reactions. The phenomenon is most easily demonstrated by using as capillaries the fine cracks caused by the plunging of a hot glass tube into water. Such a device was made use of by T. W. Richards and B. S. Lacy (*Jour. Am. Chem Soc'y*, March), who investigated the fundamental question whether this phenomenon has any effect on the quantity of metal deposited on the cathode according to Faraday's law. This question is especially important, since the interposition of a porous cup (or system of capillaries) has recently been suggested as an essential part of an accurate electrolytic apparatus for measuring electrical quantity. The authors found that the complication of electrostenolytic deposition does not affect in any way the weight of the true cathode deposit or the exact application of Faraday's law.

Liquid Ammonia Solutions.—In the March issue of the *Jour. Am. Chem. Soc'y*, E. C. Franklin and C. A. Kraus publish a second paper giving the results of a very extended investigation of the conductivity of some 37 substances in liquid ammonia solution. The results are given in tables and diagrams. Some special observations are interesting; for instance, the cyanides of the heavy metals and cyanacetamide, when dissolved in ammonia, show the remarkable phenomenon of a decrease of the molecular conductivity with the dilution in the more concentrated solutions. As the dilution increases the conductivity passes through a minimum, and then increases with the dilution in the manner characteristic of salts in gen eral. The uni-univalent salts show a wide variation in conductivity and degree of ionization with the dilution. Tables and curves are given showing these relations and comparing the behavior of salts in solution in ammonia with their behavior in aqueous solutions.

Electrochemical Series of Metals.—The May issue of the *Jour. Am. Chem. Soc'y* contains an account of an experimental investigation by G. McP. Smith, on the reciprocal replacement of the metals in aqueous solutions. An abbreviated electrochemical series of the metals reads as follows: + K, Na, Ba, Sr, Ca, Mg, Al, Zn, Cd, Fe, Co, Ni, Sn, Pb, Cu, Hg, Ag, Pt, Au —. The author formed the metals in the following pairs to be reciprocally replaceable in aqueous solution, the metal first named in each pair being, under ordinary circumstances, more readily replaceable by the second than the second by the first: K — Na, K — Ba, Na — Ba, Zn — Cu, Cd — Cu, (Fe — Hg), (Fe — Ag), Hg — Ag, Hg — Pt, Hg — Au, Ag — Au.

Electrical Thermostat.—A simple electrical thermostat is described by Fred. A. Osborn in the April issue of the *Jour. of Phys. Chemistry,* as a simple, quick and yet accurate means of controlling the temperature in laboratory work in conductivity. The construction is shown in Fig. 3, where AA are platinum points; B, the storage battery of eight cells; B′, a single cell; C, iron wire; D, a stirrer; E, a glass coil (16 cm. long, 9.5 cm. diameter), containing mercury; R, a relay of 20-ohm. resistance. The heating of the 50-liter bath is accomplished by means of one layer of iron wire of five turns, carrying a 3-amp. current from the eight cells. The bath is brought to within one-tenth degree of the required temperature, and then a very little adjustment of the screw P puts the bath into automatic regulation. The bath has many times been made self-regulating within fifteen minutes. The large volume of mercury and the very large surface of the worm combine to make the regulator very sensitive, the relay very often throwing the current off and on, while the Beckmann thermometer showed no change. On continuous runs of six or eight hours no change of 1/100 of a degree has been observed.

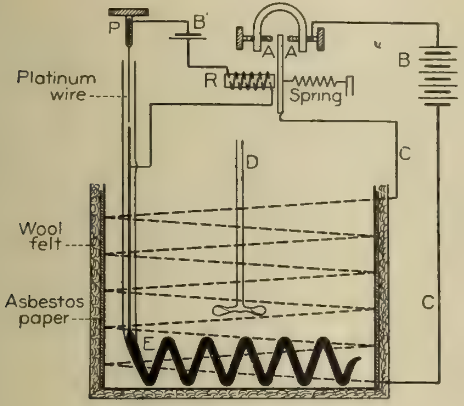


FIG. 3.—ELECTRIC THERMOSTAT.

Atomic Weights of Sodium and Chlorine.—The May issue of the *Jour. Am. Chem. Soc'y* contains the full report on a very extended and extremely careful redetermination, by Theo. W. Richards and R. C. Wells, of the atomic weights of sodium and chlorine. The authors prove that the value for sodium from Stas's classical researches is nearly 0.2 per cent too high. The new values are:

	Ag = 107.920	Ag = 107.930
Atomic weight of sodium.....	23.006	23.008
" " chlorine ...	35.470	35.473

Many other atomic weights are affected, in their second decimal places, by these results.

METALLURGY.

IRON AND STEEL.

Blast Furnace Practice.—A very entertaining lecture was given before the Franklin Institute the early part of this year, and printed in its *Journal* for February (reprinted in the *Iron and Steel Magazine* for May), by Edward A. Uehling, M. E. The subject was "The Fundamental Principles Involved in Blast Furnace Practice," and since the lecturer was in charge of the blast furnaces of the Bethlehem Steel Co. some years ago, he may be taken as speaking with authority. A very brief review of his principal points is as follows: The *ore* should not be too lean or too rich, but above all uniform in quality; the finer it is the easier it is reduced, but the more trouble there is in getting blast through the furnace and the more difficulties there will be from dust in the gases. It should be sized to at least a 2-inch round-hole screen; the worst possible mixture is coarse and fine in such proportions that the latter just fills the interstices between the lumps of the former; all that passes through a 1-25-inch mesh should be briquetted, or else smelted by itself in a specially designed furnace, the latter being probably the best treatment. *Flux* should be well broken up, of uniform size, and dust avoided; calcining before use is impracticable, because of it slaking in the air if not at once used. *Slag* acts as a filter to purify the iron globules passing through it, also as a blanket to keep heat in the iron and to protect it against oxidation by the blast; also as a remover of sulphur and a regulator of the silicon content of the iron. Mr. Uehling's statement that lime is the only ingredient of the slag which is efficient in removing sulphur is believed by many, but it has been shown the magnesia acts similarly in a low-alumina slag; the statement that alumina should be kept below 15 per cent is also open to debate, because slags as high as 25 per cent alumina work satisfactorily if silica is kept correspondingly low, *i. e.,* in basic slags, alumina replaces silica. The *fuel* keeps the charge open and provides the heat; it should be in coarse, rough, uniform pieces. It should possess cellular structure, combined with good crushing strength. Mr. Uehling's opinion is that most of the silicon in the iron comes from the silica in the fuel ash, and that, therefore, a fuel with high silica ash will produce a high silicon pig-iron. *Air* should be regular in quantity, which is best attained by keeping the blowing engines running at constant speed, and allowing the pressure to rise to what it may; this is more desirable than working by constant pressure; small leaks in the blast mains may cause many per cent loss of efficiency in delivery to the furnace; it should be regular in temperature, which is best attained by using "equalizers," which will reduce variations of 100° in the blast to 10°. The blast should be dry; as much as 28 tons of water may be blown in in twenty-four hours in wet, hot weather. Removing this by drying should save some 4 per cent of the fuel consumption, but actually, as shown by Mr. Gayley's experiment, saves 25 per cent. Why, it is difficult to say. In Mr. Uehling's opinion the drying of the blast is second in importance only to the heating of the blast, and it will be generally and promptly adopted.

The lecturer then discusses, in a lucid manner, the chemical reactions and general running of the furnace.

Hardening of Steel.—Paul Lejèune has continued Le Chatelier's experiments on the speed of cooling of steel quenched or hardened in different liquids, and reports his results in detail in the April *Reveu de Metallurgie.* In water, the specimen used cooled from 700° to 100° in 6.6 seconds, in mercury in 17.2 seconds, in alcohol 21 seconds. The duration of cooling varies very closely with the viscosity of the liquid used, for in water, oil, gelatine and glycerine, with relative viscosities of 1, 6, 25 and 36, the relative times of cooling were as 1, 6, 15 and 36.

Forging.—The April number of *Reveu de Metallurgie* contains a valuable memoir on the mode of action of the forging press and a study of the deformations of hot-compressed metal" by M. [illegible]. The tests were made on a 2000 ton press at the Holtzer steel works at Unieux. Numerous curves were taken of the pressures required to effect successive compressions. In general, the energy required is expressible as a function of the cross-sectional area of the bar compressed, the distance through which the press descends, and a coefficient K, which varies with the temperature of the ingot and the number of times the pressure is applied. Curves are given showing the variation of K with these conditions.

Aluminium Steels.—The assiduous metallographist, L. Guillet, has studied the micrography and physical properties of two series of aluminium steels (*Reveu de Metallurgie,* April). The first series is mild steel with 0.08 to 0.16 per cent of carbon, and 0.51, 1.08, 2.05, 3.05, 5.08, 7.18, 9.25 and 15.03 per cent of aluminium; the second series carbon steels with 0.66 to 0.86 per cent of carbon, and 0.05, 1.05, 1.09 2.89, 4.65, 7.0, 9.15 and 14.90 per cent of aluminium. Each steel was tested *au naturel,* forged tempered and annealed. Some twenty-five fine photo-micrographs illustrate the article. The conclusions arrived at are: (1) Aluminium has no important effect on the mechanical properties of steels as long as it is present in small proportion; above 2 to 3 per cent it causes great brittleness. (2) Up to 15 per cent of aluminium there is not found any combination of iron and aluminium; the latter simply goes into solution in the iron. (3) The solution of aluminium thus formed does not dissolve, causing the perlite present to take a special granular form, explaining the brittleness observed, and martensite is found after tempering only where there was perlite. (4) Free cementite is found, even in aluminium steels with less than 0.85 per cent of carbon. (5) The steels low in aluminium have very feeble hysteresis, which fits them for electrical machinery. (6) Aluminium steels with 1 per cent are cemented by carbon only half as fast as pure steels, and with 3 per cent only one-quarter as fast, while a 7 per cent steel could not be cemented; it appears that carbon is no longer soluble in iron containing already-dissolved a certain quantity of aluminium.

Mr. Gayley's Dry-Blast Process.

By the courtesy of the Council of the American Institute of Mining Engineers, we are enabled to give the following abstract from an advance copy of a discussion by Dr. J. W. Richards of Mr. Gayley's celebrated paper on the application of dry-air blast to the manufacture of iron. The full discussion by Dr. J. W. Richards will be published in the July issue of the *Bi-Monthly Bulletin* of the Institute. Dr. Richards' discussion is most instructive and interesting in two respects. First, it gives what appears to be a full explanation of the puzzling question why Mr. Gayley obtained such a far better result than could have been expected from theory or practice. Second, Dr. Richards' discussion is an instructive example of what can be accomplished by exact metallurgical calculations in the analysis of a process. Dr. Richards first states the exact data given by Mr. Gayley as to operation with dry-air blast compared with moist-air blast. He adds a few assumptions as to the composition of the ore, limestone and coke used and of the pig-iron produced, which correspond with sufficient exactness to actual furnace practice. On the basis of these data supplied and assumed he calculates the balance sheets of the materials entering and leaving the furnace per hundred parts of pig-iron produced for operation with moist blast and dry blast respectively. (In this calculation there is a poor agreement between the calculated volumes of air with the stated piston displacements. If the piston displacements are reliable, the delivery efficiencies are 73 per cent in pumping warm, moist air, and 83 per cent in pumping the cold, dried air. It appears more probable that the delivery efficiency is raised in the slower running by the amount just indicated than that the calculations of the volumes received by the furnace are grossly incorrect. The rather startling conclusion is thus reached, that the furnace received nearly as many cubic feet of dried air, measured at 25° F., as it did of moist air measured at 75° F., although the engines were running 15.8 per cent slower. The only explanation Dr. Richards has to offer for this is the increased delivery efficiency; but possibly some other factor may partly account for it.)

Dr. Richards then passes over to the calculations of the heat balance-sheet per 100 kg of pig-iron produced. One figure in this balance sheet enables one to check the calculations; this is the loss of heat by radiation, conduction, etc., which is calculated (by difference) as 77,101 calories with moist blast and 63,025 with dry blast; from the increase of the output from 358 tons to 447 tons per day one would expect a reduction of the radiation loss from 77,101 to 61,830 calories, which agrees well with 63,025, thus giving a good check on the general accuracy of the calculation.

This heat balance sheet is now the basis of the explanation of the whole matter. From it follows that the economy in heat requirements by using dry blast is due to six different causes, which are tabulated below; to understand this table it should be said that the total heat requirement of the furnace, when running on moist blast (the total heat received by it and developed within it), is 383,635 calories.

	Heat Saved.	Per Cent of Requirements.
On decomposition of moisture of blast.	11,287	2.95
On heat in waste gases...............	20,037	5.25
On radiation, etc......................	14,076	3.7
Better combustion of carbon..........	26,395	6.9
Smaller heat in slag..................	480	0.1
Less heat in blast.....................	1,535	0.4
Total	73,810	19.3

Mr. Gayley had found that actually 19.6 per cent less coke per ton of pig-iron made were required. It is thus clear that of the total saving effected, less than 15 per cent is directly attributable to the absence of moisture in the blast; the other 85 per cent of the economy is due to indirect influences, viz.: (1) The heat in the waste gases is less, because the waste gases are less in amount, and therefore pass out cooled to a lower temperature; (2) the radiation loss is reduced, because the furnace smelts faster, and these losses are in actual amount proportional to the time rather than to the amount of material smelted, and (3) the carbon is consumed better, because there is much less carbon dioxide generated, and, therefore, there is less excess of carbonic oxide left unused in the waste gases—in other words, using moist blast, there is an excessive amount of carbon monoxide generated at the tuyeres in the effort to keep up the smelting temperature with the moist blast, and only a certain limited amount of this can be oxidized to carbon dioxide by the oxygen supplied by the reduction of the solid charges, while in using dry blast the temperature of the smelting zone is easily maintained, burning only 77 per cent as much carbon at the tuyeres (58.05 kg instead of 75.3 kg), and the carbon monoxide formed abstracts all the oxygen from the charges, and so is converted more largely into carbon dioxide than in the first case.

The whole question of economy obtained therefore converges towards the discussion of the generation of the heat necessary for smelting in the region of the tuyeres. To generate this amount of heat for the requisite temperature, with the production of no more carbon monoxide than is necessary to achieve reduction of the charges above, is the direction in which economy is to be obtained. The efficiency of hot blast is due to exactly this reason, that it increases the smelting power of the tuyere region without an increase in the carbon monoxide formed, and with a higher temperature of the gases in the tuyere region.

These points are then made the subject of further discussion, with respect to the increased output, and it is shown that the gaseous products of combustion, using dry blast, have 23.4 per cent more of their heat utilizable per unit of time for smelting purposes, and have an additional relative heat potential of 5.6 per cent to assist in imparting this heat more quickly, making a total relative smelting capacity of $123.4 \times 1.056 = 130.3$ to 100, or 30.3 per cent increase if the quantity of oxygen blown in per unit of time were constant. But, since the quantity of oxygen actually blown in per day in Mr. Gayley's experiment was as 100 for moist blast to 96.3 for dry blast, the rapidity of smelting should have been as 100 to $130.3 \times 96.3 =$ 100 to 195.5, or a daily increase of 25.5 per cent.

Since Mr. Gayley obtained an increase of 24.86 per cent the views advanced by Dr. Richards give a satisfactory explanation of the increased smelting power of the furnace: that the heat saved by absence of moisture is directly utilized for increased smelting capacity, aided, furthermore, by a higher heat potential, which of itself alone would increase the smelting rate about 5 per cent, the two factors working simultaneously and necessarily interdependently towards the total effect.

In conclusion Dr. Richards says that the increased efficiency obtained by Mr. Gayley could theoretically have been obtained by an increased temperature of blast alone, viz.: by using the moist blast at 597° C. (1,107° F.), instead of at 382° C. (720° F.). Such increase would produce the effects of quicker running and economizing coke to the quantity noted with dry blast, but would still leave the furnace subject to the irregularities inseparable from using ordinary air with its varying temperature and content of moisture. The increased regularity of running of the furnace and quality of product, due to uniform temperature of air supplied to the blowing engine and uniform temperature before the tuyeres, is the fundamental economic justification for Mr. Gayley's innovation; the increased rate of driving and economy of fuel alone could be obtained more cheaply by increasing the capacity of the stoves.

ZINC.

Zinc.—The decomposition and formation of zinc sulphate by heating and roasting has been the subject of a very extended experimental investigation of Professor H. O. Hofman, of the Massachusetts Institute, the results being given in the January issue of the *Bi-monthly Bulletin* of the American Institute Mining Engineers. The first part of the paper deals with the decomposition of zinc sulphate by heating in air. Dehydration is practically complete at 263° C. Decomposition begins at 532° C., and is complete at 739°. The second part of the paper deals with the decomposition of zinc sulphate by heating with carbon (reducing roast). The reaction of carbon upon zinc sulphate begins at 409° C., but is weak, and lasts only a short time; it grows stronger at 425°, 463°, 475°, 500° C., but weakens perceptibly after a few hours; only at 528° C. does carbon appear to become decidedly active, but even here the effect is at best very slow and imperfect, as it is opposed by the direct reduction of zinc sulphate to zinc sulphide. Under the most favorable conditions the 19.86 per cent of sulphur of zinc sulphate are reduced to 3.47 per cent; of this total 1.41 per cent is sulphide-sulphur, and 2.06 per cent sulphate-sulphur. The third part of the paper deals with the formation of zinc sulphate by roasting. The leading features of the sulphatizing roasts are given by the author in a table. The chief results are as follows: The amount of normal zinc sulphate soluble in hot water obtained in the tests is very small. Based upon 100 parts of zinc in the charge it reaches 9.07 per cent with raw blende, and 15.92 per cent with dead-roasted mixed with a large excess of pure pyrite. Both extractions are too small to be of much practical value. The fundamental difficulty in obtaining a satisfactory percentage of extraction lies in two facts: normal zinc sulphate is rapidly decomposed at the temperature for roasting blende; at the beginning of a roast, sulphur dioxide strongly predominates over sulphur trioxide. In the process carried out in the Harz mountains, the ore is roasted very slowly in heaps, which have to be turned three times; the first heap of 500 tons ore burns from six to seven months, the second fire lasts from six to eight weeks, and the third from four to six weeks. Normal and basic sulphates are formed mainly in the fines, *i. e.*, in the relatively cool cover, and these alone are leashed with water and sulphuric acid. The probable explanation for the formation of these sulphates is that zinc sulphate formed toward the hotter center of a heap is decomposed, and the sulphur trioxide, dioxide and oxygen rising attack the fines in the cool cover and form normal and basic salts, or normal salts alone, which are decomposed in part when the heat creeps up toward the end of the roast. The ratio of sulphur as basic sulphate to sulphur as normal sulphate increases with the temperature. Ferruginous (Warren) blende is more difficult to roast than blende running low in iron (Japlin), but the former gives, under similar conditions of treatment, a higher yield in normal sulphate, owing to the presence of isomorphous sulphide, the basic salts of which have a strongly sulphatizing effect when they are decomposed by heat.

Zinc.—The January issue of the *Bi-monthly Bulletin* of the American Institute Mining Engineers contains a paper by H. C. Meister on the zinc-smelting industry of the Middle West. The paper is mainly of a descriptive character. The author gives statistical data, and deals briefly with the processes of roasting and smelting, several types of furnaces being described. The number of retorts in operation in the coal-burning smelters in Illinois, Missouri and Kansas is 15,606, in the natural-gas-burning smelters in Indiana and Kansas, 37,480, total, 53,086.

Residual Zinc in the Retorts.—A valuable contribution to the subject of the state of combination of the zinc left in the retort residues is given in a letter of E. Schuchard to the editor of *Metallurgie*, and published in the March 8th number of that journal. From analyses of some 500 residues the conclusion is drawn that an average of 2.5 per cent of zinc is present in them; that if sufficient care in firing is observed the residues should never contain over 3.0 per cent of zinc, and that such statements as: "the residues contain 10 per cent and over of zinc," and "this zinc is mostly combined with sulphur," are statements of antiquated practice and kindergarten methods, which have no application at all to modern zinc smelting.

To determine the amount of zinc combined with sulphur in the residues, the sample is first washed with water, to remove the calcium sulphide which is always present. The residue is then treated with acetic acid, to attack and dissolve the zinc oxide, while zinc sulphide remains. In this way it has been determined that from 12 up to 43 per cent of the zinc in the residues is present as zinc sulphide, an average of about 80 analyses giving 31 per cent, a proportion which seems moreover to be independent of the total per cent of zinc in the residues at least from 0.15 to 7.00 per cent.

Furthermore, it is not true that unroasted sulphur in the roasted ore holds back twice its weight of zinc in the residues. Brandhorst has proved that a partly roasted blende, with even 5.8 per cent of sulphur left in it could be so completely distilled that only traces of zinc could be found in the residue. In one residue tested there was present 14.58 per cent of calcium sulphide, 2.12 per cent of zinc oxide, 1.57 per cent of iron sulphide and 1.31 per cent of zinc sulphide, showing that very little zinc sulphide may be left even when the residue is rich in other sulphides. The observations agree very well with the opinions on the same subject by W. McA. Johnson, printed in our issue of January last.

MISCELLANEOUS.

Fire-Clays.—A paper, by H. A. Wheeler in the January issue of the *Bi-monthly Bulletin* of the American Institute Mining Engineers, deals with the fire-clays of Missouri. It may surprise some of our readers to learn that, among the industries based on the mineral resources of the United States, that of clay now ranks third, being exceeded in value of product only by pig-iron and coal. The value of pig-iron produced in 1902 was $372,775,000; coal, $367,032,069; clay products, $122,169,531. About 10 per cent of the latter was firebrick. The author deals with the Missouri fire-clay industry and [illegible] The distinction is made between flint fire-clays and plastic fire-clays, their occurrence, chemical composition and physical properties being discussed.

Plasticity.—We have received a reprint of a paper on the colloid theory of plasticity by Dr. Allerton S. Cushman, chemist of the road material laboratory of the Department of Agriculture. The paper was presented before the American Ceramic Society. The main points brought out by the author are as follows. Both plasticity and binding power are merely manifestations of a colloid modification of matter which exists in rocks and clays. The activity of these useful qualities depends upon the characteristics of the special colloids that may be present, as well as upon their past history, and the modifying effect upon them of saline and organic solutions. Absorptive qualities of clays, such as their ability to stick when touched with the tongue, and as exhibited by their occasional use as "lakes," clarifying agents, etc., are to be ascribed to the same cause. In conclusion the author says that in bringing out the above points, he does not desire to minimize the importance of the effect of the shape and size of the grain on the useful qualities of clays.

Utilizing Waste Heat.—While the utilization of the waste gases from blast furnaces is at present in the front rank of interest, the utilization of the waste heat from coke ovens is also a matter of considerable importance. The February issue of *Mines and Minerals* contains some illustrated notes on the general arrangement of apparatus for this purpose, and on tests made at the Victoria-Garesfield Colliery of the Priestman Collieries Co., of England. A special object in these tests was to compare the results obtained with water-tube boilers in comparison with boilers of the Lancashire type. The figures given in the article show that "the heating surface in the Stirling boiler abstracts more heat from the gases than the heating surface in the Lancashire boilers, and that although the mean temperature of the gases in contact with the surface in the Lancashire boiler is higher than the mean temperature of the gases in contact with the surface in the Stirling boiler, not only was the normal capacity of the Stirling boiler reached, but it gave an evaporation approximately 28 per cent in excess of that obtained with the Lancashire boilers, per pound of coal coked." The heating surface for best efficiency with waste-heat boilers should be greater than for coal-fired boilers.

RECENT METALLURGICAL PATENTS.

ROASTING.

Frederick J. Falding (788,098, April 25) patents details of construction of a roasting furnace of the well-known type comprising a series of superimposed hearths and a series of rotating stirring arms, arranged above the hearths and carried by a central shaft. The object is to so connect the stirring arms with the central shaft as to facilitate the cooling of arms and shaft, and also facilitate any necessary repairs of the arms, [illegible] the necessity of replacing the shaft itself is largely done away with. For each hearth a pair of stirring-arms is provided, secured to the shaft diametrically opposite each other. For this purpose the shaft is perforated at two points opposite each other, and in these perforations are secured fittings 13 in the form of hollow cylindrical sleeves. The simplest construction is shown in Fig. 1, which refers to the case that it is only necessary to cool the shaft D. The illustration shows the annular bosses 14, and the projecting socket ends 17, and the stirring arms F, which are provided with shoulders 29 to abut against the ends of the fittings 13. The arms are held in place by the pins 26. The arrows indicate the flow of the cooling liquid through the shaft around the fittings 13. By the use of the fittings 13, which are held in place in the shaft merely by being tightly fitted and without any positive means, the sleeve, when deteriorated, or when not properly fitting, owing to the deterioration of the shaft, may be readily withdrawn from the shaft by tongs passed through the apertures in the furnace-wall, and another sleeve, having bosses 14, of the proper dimensions inserted. By the employment of such fittings the necessity of cooling down the furnace and withdrawing the shaft for the purpose of inserting a new shaft, is largely obviated. A more complicated construction, comprising partitions in the shaft, in the arms and in the fittings, is described in the patent, for the case that it is necessary to cool both the shaft and the stirring arms.

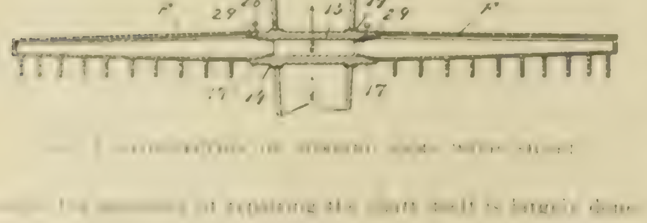


FIG. 1.—[illegible]

Albert C. Johnson (786,905, April 11) patents a rocking, or tilting, roasting furnace of cylindrical shape with a horizontal axis. Within it are provided successive longitudinal and inclined shelves, one above the other. The alternate ends of the shelves are free, so that the ore having passed lengthwise along one shelf to its free end, drops onto the next shelf and passes along the latter back through the cylinder to the other end. Each shelf is curved, being higher in the center, *i. e.*, the cross-section of the cylinder shows the cross-sections of the shelves in form of convex curves. When the furnace is rocked the ore is carried from one side of the shelf over to the other side, and at the same time it is advanced towards the free end of the shelf.

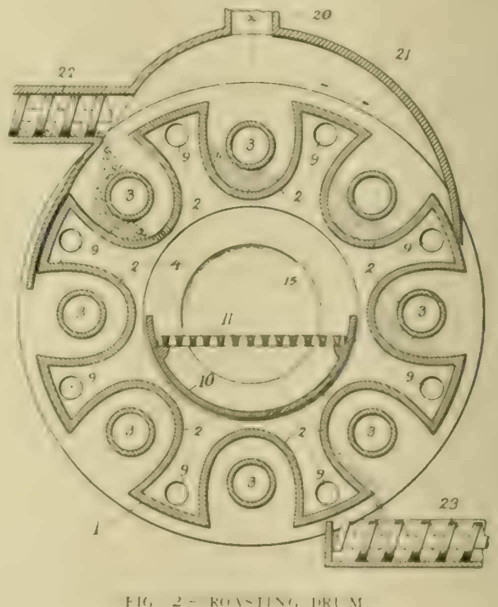


FIG. 2.—ROASTING DRUM.

J. W. R. Laxton (789,371, May 9) patents the revolving roaster drum shown in cross-section in Fig. 2. The drum 1 has a much greater diameter than axial length, and has a large number of pockets 3 in its periphery. The heat is supplied from the fire-pot 10 within the drum, and supported on the outside by a framework. The fire is built upon the grate-bars 11. The ore is supplied through 22 into a pocket, which retains the ore until the drum has made one-half revolution. It is then delivered by gravity into the conveyer 23.

Peter A. Wagner (789,303, May 9) endeavors to prevent the loss of metal, due to the escape of metallic vapor passing with the products of combustion out of the furnace stack. For this purpose he inserts, between roasting furnace and stack, a condensation and collecting chamber, filled with water to a certain level. The products of combustion pass through this chamber before they get to the stack, and are condensed in this chamber.

Seely B. Patterson (787,540, April 18) roasts ore in a stack furnace without solid fuel or fuel gas within the ore. The column of ore is first subjected throughout its length to the action of hot air for a certain time, after that the lower portion of the column is treated with steam for a certain time, then follows again treatment with hot air, and so on. The following equations are given for the action of the hot air:

$$2FeS_2 + 11O = 4SO_2 + Fe_2O_3$$

and for the action of the steam

$$2FeS_2 + 4H_2O = Fe_2O_3 + 4H_2S + O$$

Wet Process.

When a solution is introduced into the bottom of a pulp tank for leaching, a lead conduit is generally used with perforations, through which the solution is passed into the tank under pressure. The lead is sometimes so soft that the orifices are gradually enlarged until a slot is formed in the pipe, and the pipe becomes useless. Chauncey E. Dewey (787,902, April 25) therefore provides the pipe at intervals with orifized nozzles of sufficient length to cause the agitating fluid to pass upwardly into the tank in vertical jets. The tank is V-shaped in cross-section, the arrangement being clearly shown in the above Fig. 3.

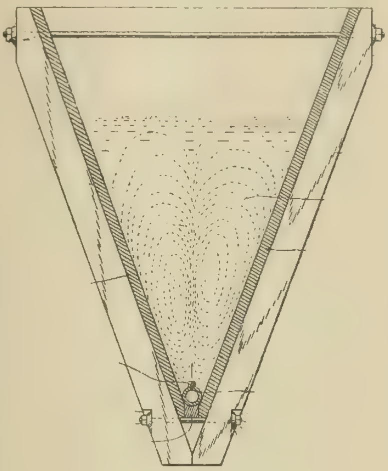


FIG. 3.—LEACHING TANK.

G. H. Waterbury (788,443, April 25) treats copper ore by the following wet process: The pulverized ore is placed in a tank containing water saturated with sulphurous acid, and containing a small quantity of sulphuric acid. Air and steam are then introduced at the bottom of the tank, whereby the copper is dissolved. The solution is then passed into the precipitating tank, filled with particles of aluminium and steel in such a way that the solution can freely circulate therethrough. Air and steam are then introduced at the bottom of the tank, and passed through the solution, whereby the copper is precipitated and settles on the bottom of the tank.

Lead Containers for Acids.

In order to increase the capability of ordinary commercial lead to withstand the corrosive effect of acids, H. E. Miller (786,581, April 4) proposes the following treatment: The lead is oxidized, preferably to the form of litharge, until about 95 per cent are oxidized. The oxide and residum are then separated and the litharge is reconverted into metallic lead, "which will be found to have acquired a capability of resisting the corrosion of acid equal to that of the most satisfactory specimens, regardless of its natural source or previous process of manufacture, so that the metal resulting from this process is of uniform and high excellence in the desired characteristic."

BOOK REVIEWS.

Electric Furnaces and Their Industrial Applications. By J. Wright. New York: The Norman W. Henley Publishing Co. Illustrated. Price $3.00.

That which may be described as the "scissors and paste" type of book is usually looked upon with something akin to contempt. It is regarded by many in somewhat the same light as an index—very useful, but requiring only a low order of ability to turn out. This way of looking at the subject is altogether wrong, for the making of a good index requires no small ability, and the skillful handling of scissors and paste deserve high praise.

"Electric Furnaces and Their Industrial Applications" is a book of the scissors and paste variety, but we regret that the author has failed altogether to produce a useful work, although there cannot be the least doubt that a book covering the subject would be very welcome. Although Mr. Wright has apparently made a most exhaustive catalogue of the technical and patent literature of his subject and has included in his book a large number of furnaces chiefly distinguished by their ingenious impracticability, his information is so inaccurate and the carelessness of his descriptions so marked throughout that it seems difficult to recommend the book. A few examples taken at random will serve to illustrate the general character of the work. In describing the manufacture of carborundum he says:

"The subsequent conversion, which occupies about 36 hours, results in some 2 tons of carborundum, as against a theoretical yield of 4 tons, from which it will be seen that the process is anything but efficient."

A carborundum furnace turns out much more than 2 tons of carborundum in 36 hours, as Mr. Wright might easily have discovered from his own figures, given two pages further on, where we learn that in 1901 with 2000 horse-power the annual output was 1690 tons. A simple calculation shows that a 1000-horse-power furnace turns out about 3.5 tons in 36 hours.

Turning to the manufacture of siloxicon, we are told that it is made from a mixture of "one part carbon to two parts silicon" (*sic*). When siloxicon is heated to 7000° F., "siloxicon vapor" (!) is driven off. In the description of Acheson's furnace for boiling carbon articles we learn that the latter are embedded in a heating resistance composed of "carbon and silicon."

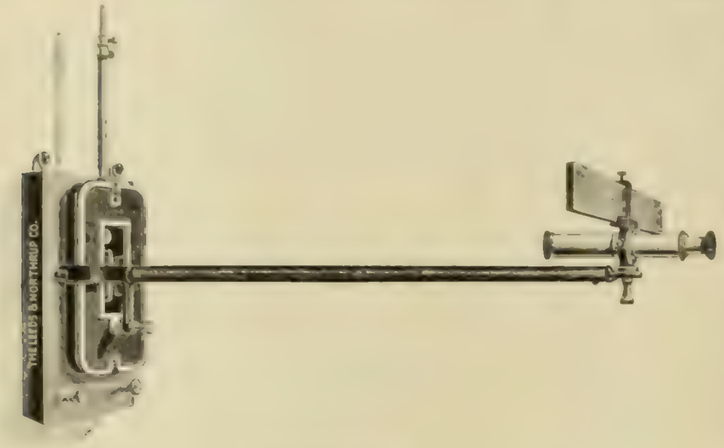
It is interesting to find that of the various electric furnaces used for the manufacture of steel, "the Gin furnace * * * is probably one of the simplest in point of application." The author should explain in detail how in "Mr. Keller's electric blast (*sic*) furnace" the "heating effects" of the electrodes are "equalized by means of an ammeter included in each circuit." The induction furnace of Kjellin, as described at one place, is an astonishing apparatus, consisting of "an annular or helical channel in a refractory base filled with a conducting or semi-conducting medium, which constitutes the furnace charge and has a heavy current induced in it by a surrounding coil of many turns, carrying an alternating current."

Similar examples might be infinitely multiplied; but enough have been given to indicate the inaccurate and careless writing which characterizes the book. The work may be of some use as a catalogue of electric furnaces, practicable and otherwise, though even this is doubtful since references are rarely given which would enable the reader to refer to original descriptions. Certainly the book should not be placed in the hands of students who are desirous of obtaining accurate information.

D'Arsonval Galvanometers.

The Leeds & Northup Co., of Philadelphia, who, at the St. Louis Exposition, received a grand prize for their electrical measuring instruments, exhibited in connection with the exhibit of the National Bureau of Standards, have for a long while paid special attention to the design of the D'Arsonval galvanometers. After having placed several types of this galvanometer on the market they have now constructed an instrument ("type P") which combines the good qualities of their former types with extreme simplicity and compactness of construction. On account of its production in quantity it is sold at a low price. As a result of its very simple design it is very easily set up, easily manipulated and all working parts can be readily gotten at for inspection or repair so as to appeal to the student beginners, while its reliable working qualities commend it to the confidence of experienced investigators. This type of instrument is sold either in tripod form to be placed on a table or for suspension on the wall. In the latter case it is only necessary to drive two screws into the wall and hang the galvanometer up on them by means of two hooks in the backboard. By screwing one or two of the screws in or out it is easily made to hand perpindicular and allow the coil to swing free. The latter form of the instrument is shown in the adjoining illustration.

A great advantage is that the moving system is always entirely visible. Should it become necessary, it is most easy to remove the telescope and scale arm and to take off the front plate, whereupon the coil is directly accessible.



GALVANOMETER

The deflections are approximately proportional to the amperes or for ballastic measurements to the coulombs discharged. The system is well damped, so that it does not oscillate but make its deflection quickly, and comes promptly to rest either deflected or at zero.

A strong suspension is provided which allows no zero shift. The suspensions are consequently not easily broken, and when broken can be easily replaced; the only tools required for the latter purpose are a pair of forceps and a watchmaker's screw driver. The usual suspension is a heavy phosphor bronze strip, rolled from a wire of 0.003-inch diameter, and the coil carries, permanently attached to it, a copper rectangle which makes it nearly dead beat. It is thus absolutely free from zero shift. The coil resistance is about 100 ohms, and the sensibility such that 1 volt through 80 megohms will cause a deflection of 1 millimeter with the scale at 1 meter distance, or one-half millimeter on its own scale, which is one-half meter from the mirror.

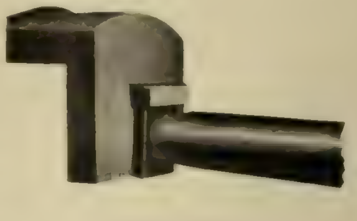
By using lighter upper suspensions (which can be easily done) the sensibility can be very much increased. With an upper suspension of phosphor bronze wire, .0015 inch diameter, the sensibility will be such that 1 volt through 150 megohms will cause a deflection of 1 mm., or more with the scale at 1 meter's distance. The zero shift on full scale reversed, deflections will generally not exceed 1 mm.

With an upper suspension of phosphor bronze ribbon rolled from strip .0015 inch diameter the sensibility will be such that 1 volt through 400 megohms will cause a deflection of 1 mm. or more with the scale at 1 meter's distance. The zero shift on full scale reversed deflections will generally not exceed 1 mm.

There is an ample coil clearance provided so that no delicate adjustment is necessary to get the coil to swing free. The scale is large and clear and very easy to find and to read.

Furnace Door Arch-Plate.

The accompanying illustrations show the Johnson patent arch-plate made by the Wellman-Seaver-Morgan Co., of Cleveland, for the purpose of facilitating the work of repairing and rebuilding furnace-door arches. It is made of cast steel, and it is hence a permanent appliance which can be utilized on short notice. The arch-plate fits on the ram of the charging machine, and can be quickly picked out by it or detached as desired. The furnace door is raised out of the way and the charging machine inserts the plate in the charging opening of the furnace, holding it there in proper position until the repairs are completed, when it is withdrawn and detached from the ram of the charging machine.



ARCH PLATE

The arch-plate serves a double purpose.

First, it forms a shield against the heat from the interior of the furnace, thus enabling the work of repairing the arch to be carried on without shutting down the furnace. In the old method of repairing arches it has been necessary to build a temporary protecting wall or bulkhead of brick, and on account of the heat this must be done with long-handled tools, making the process not only awkward and tedious, but also expensive, as it necessarily delays the furnace. The use of the Johnson arch-plate does away with all this.

Second, the Johnson arch-plate serves as a form on which the arch can be expeditiously relaid, supporting it during construction. The flange serves as a stop for the brick in laying the arch.

Work which would ordinarily require hours in the old way can, by using this arch-plate, be done in a small fraction of the time, while the charge of steel is being melted down and without any delay to the furnace.

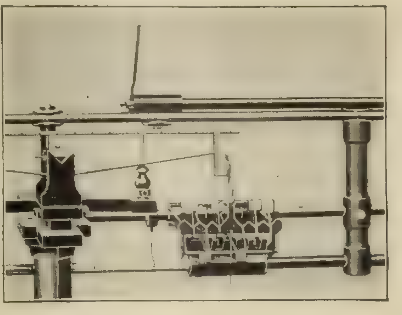
JOHNSON ARCH-PLATE IN USE

An Improved Multiple-Rider Carrier.

The accompanying illustration shows an improved multiple-rider carrier for button balances, just placed on the market by William Ainsworth & Sons, of Denver, Col.

The weights or riders are each carried by an individual arm and cannot become displaced except through gross carelessness.

The arms are numbered in their front surfaces to correspond with the weight of the rider carried, and pivoted at the rear, so as to just swing clear of the comb-shaped bar below the stirrup, and are operated by a single rod with a thumb piece on the end of it extending through the case. When the arms are up the riders are carried by them far enough above the comb-shaped bar so as not to interfere with the oscillations of the beam.


MULTIPLE-RIDER ARRANGEMENT.

In operating, it is only necessary to move the rod to the right or left until the figure corresponding to the weight of the rider to be shifted, stands opposite to the index pointer shown to the left of the frame carrying the arms; then give the rod a slight turn and the arm carrying the rider is swung downward, shifting it on the comb-shaped bar.

When the weighing is completed, a glance at the figures on the arms that are down shows the combined weight of the riders on the bar. A slight turn of the rod resets all of the riders simultaneously.

The carrier can be operated with great speed and certainty, and the riders can be used for an indefinite length of time without perceptible change in weight.

Industrial Notes.

The New York office of the ROBERTS CHEMICAL Co. has been moved from 40 Wall Street to 60 Wall Street, where a more commodious suite has been secured on the twenty-fifth floor, from which is obtained an impressive view of New York harbor and surroundings.

Messrs. P. BLACKISTON'S SON & Co., the well-known publishers of scientific books in Philadelphia, have sent us a list of the international atomic weights for 1905, reprinted from the *Journal of the American Chemical Society,* Vol. 27, No. 1. This list is printed on pasteboard and may be hung up for use in the laboratory.

BLOWING ENGINE.—The Republic Iron & Steel Co., Youngstown, Ohio, has placed an order with ALLIS-CHALMERS Co., of Milwaukee, for one horizontal cross-compound condensing Bessemer blowing engine. The high-pressure cylinder will be 46 inches in diameter, the low-pressure cylinder 88 inches, and the two air cylinders each of 76 inches effective diameter. The stroke is to be 60 inches. The engine will be capable of delivering 24,000 cubic feet of free air per minute at a pressure of 28 pounds per square inch. The enormous size of this machine makes the order worthy of special note.

VANADIUM.—We have received from Mr. J. BAXERES DE ALZUGARAY a pamphlet on vanadium, its recent history, treatment, uses and market. The author first gives some historical notes on his intimate connection with the metallurgy of vanadium. On account of its high melting point it is preferable to use in practice ferro-vanadium alloys of lower melting point; the author recommends a 10 per cent alloy as the best for general use. The iron contained in the ore is not separated, but left to be precipitated together with the vanadium oxide to prepare ferro-vanadium from the reduction of this mixture. The ore is first fused with a suitable proportion of nitro cake, which is easily obtained at a low cost from nitric acid works. This fused mass, containing iron and vanadium sulphates, is lixiviated with hot water, and from the solution a mixture of iron and vanadium hydroxides is precipitated by means of soda lye. The precipitate is collected in a filter, dried, and then reduced in an electric arc furnace with a reducing agent, or reduced by means of aluminium. While vanadium was formerly believed to be scarce in nature, the author states that vanadium ores, such as lead vanadate (vanadinite), lead-cupro-vanadate (descloizite), and the vanadium-aluminium-silicate, known as roscoelite, are by no means scarce, so that the industry may count on a plentiful supply of ore. The author then discusses the applications of vanadium in the iron and steel industries. The introduction of vanadium into all the usual brands of steel, wrought and cast iron is stated to increase the malleability and the breaking strain by 90 per cent, and even further. "Crucible steel of a poor quality mixed with ferro-vanadium has produced a metal with a breaking strain of 61.57 tons (137,915 pounds) per square inch, and with an elongation of 23 per cent. The steel used to produce it broke before the addition of vanadium, under a load of 31.14 tons (69,753 pounds) per square inch, and had an elongation of 16 per cent." Several other examples are quoted. "The introduction of vanadium renders steel very mild when annealed and very hard when tempered. Armor plates can be produced of vanadium steel with an extremely hard surface, and with a soft backing. The superior ductility of this soft backing of vanadium steel would avert the splintering of the hardened plates and shields when struck by projectiles." The author emphasizes the great suitability of vanadium for self-tempering steel tools, but strongly urges to use in all cases pure ferro-vanadium, since impurities are likely to counterbalance the good effect of the vanadium. Vanadium-aluminium is briefly mentioned. "Copper-vanadium is possessed of great resistance and hardness. These alloys are suitable for the manufacture of guns, propeller blades, valves, and for many other uses in which a hard copper alloy possessing great tensile strength is required." The price of a 25 to 30 per cent ferro-vanadium, it is stated, was $8.75 in 1893, and now is $2.50. The author estimates that the vanadium contents of ferro-vanadium is now generally quoted at $10.00 per pound.

The GENERAL STORAGE BATTERY Co., of 42 Broadway, New York, have issued a handsomely illustrated pamphlet, giving description, standard sizes and prices of the Bijur storage batteries, "high-duty" type, for stationary service. A description of this battery was given on page 203 of our last issue.

LARGE PUMPING ENGINES.—The city of Montgomery (Ala.) has recently placed an order with ALLIS-CHALMERS Co., of Milwaukee, for a horizontal duplex double-acting plunger pump, directly driven by a horizontal cross-compound Reynolds-Corliss condensing engine. This engine will have a capacity, against a total net head of 254 feet, of 8,000,000 United States gallons every twenty-four hours, when operated at a plunger speed not exceeding 360 feet, or 60 revolutions per minute, and supplied with steam at a pressure of 125 pounds per square inch at the throttle. This pumping engine is guaranteed to develop a duty of 135,000,000 foot-pounds for each 1000 pounds of dry steam consumed by the engine when operated at its rated capacity under contract conditions. The cylinder will be fitted with Reynolds-Corliss valve gear, especially arranged for this type of engine, and having independently adjustable cut-offs for each cylinder, the high-pressure valve gear being also under the control of an adjustable speed governor, provided with suitable mechanism for varying the speed of the engine between certain limits. The steam and exhaust valves of each cylinder will be operated by independent eccentrics. The total weight of the pumping engine is to be 230,700 pounds. The pumps will have a suction valve area of 297 square inches, and a discharge valve area of 297 square inches for each plunger, or 200 per cent of the area of the plunger.—The American Steel & Wire Co. has ordered for its American Works at Cleveland, Ohio, a 6,000,000-gallon pumping engine, which will also be built by Allis-Chalmers Co. The steam end of this pumping engine is a "heavy duty," horizontal, cross-compound Corliss condensing engine, with jacketed steam cylinders. The engine frames are a combination of the "heavy duty" and "reliance" type, having the slide cast solid with the frame, and a heavy foot extending down and resting with a broad base upon the foundations. The housing is also carried outside of the crank, as on the "heavy duty" frame, and a second slide housing is cast on to the frame for the attachment of the pump chambers; this pump end of the frame having a bored slide for the plunger cross-head. This design provides for the attachment of each steam cylinder and the corresponding pump cylinder to one and the same engine frame at opposite ends, providing an ideal and simple transmission for all the working stresses. The pumps are double-acting, with outside, center-packed plungers. The suction air chambers of the pumps are of special design, and insure absolutely the prompt and complete filling of the pumps on the suction stroke, even under very high suction lifts. The valve area and all waterways through the pumps are made very large, in proportion to the amount of water to be handled. The discharge air chambers are very large, and their effectiveness is greatly increased by large equalizing pipes, which connect them.

On the occasion of the recent International Railway Congress in Washington the various **Westinghouse Companies** have issued an extremely neat and profusely illustrated booklet on the "Westinghouse Companies in the Railway and Industrial Fields," with an excellent portrait of Mr. George Westinghouse as frontispiece. While the booklet appeals in the first line to the railway and tramway engineer, it should be of great interest to every engineer on account of the historical notes contained in it on the evolution of the different Westinghouse Companies.

Personal.

Dr. **Chas. F. Chandler**, of Columbia University, widely known for many years as one of the most prominent representatives of chemical and electrochemical science in this country, was married on May 24 to Miss Augusta Polhemus Berard, of New Hartford, Conn.

Mr. **Woolsey McA. Johnson** has accepted a position as superintendent of the Lungwitz Reduction Co. This company has a large zinc blast furnace installed at Warren, N. H., to operate under the Lungwitz patents. The process in which the shaft is placed under a pressure of 45 pounds per square inch, was noticed in Dr. Franz Meyer's article on zinc metallurgy in our January issue. The furnace is designed for the treatment of ore from the Warren mine.

Dr. **Henry Cook Boynton**, instructor in metallurgy and metallography in Harvard University, was awarded with a Carnegie Research Scholarship at the recent annual meeting of the Iron and Steel Institute in London. Dr. Boynton has already done much valuable research work in the field of the metallography of iron and steel.

Dr. **Charles A. Doremus**, chairman of the New York Section of the American Electrochemical Society, sustained a severe bereavement in the death of his mother, Mrs. R. Ogden Doremus, wife of Prof. R. O. Doremus, and for many years a distinguished leader in the social and intellectual life of New York city.

Dr. **Edward F. Kern**, who has been associated during the last three years with Mr. Anson G. Betts in his well-known research work on electrolytic refining of lead, electrolytic treatment of slimes, etc., is now connected with the metallurgical department of the Columbia University. Dr. Kern is a graduate of the University of Pennsylvania and of Columbia Uni-[illegible] having received the degree of Ph.D. from the latter [illegible]

We are obliged to Mr. **Alois von Isakovics**, secretary of the New York Section of the American Electrochemical Society, for four most interesting photographs, taken at the recent general meeting of the Society in Boston. Two of them were taken by flash-light after the banquet, and are excellent photographs, although, unfortunately, unsuitable for the making of half-tones. One of the other photographs is herewith reproduced, showing a group of distinguished members, with the two gentlemen in the center, to whom as chairman and secretary of the Boston committee, much of the great success of the meeting is to be credited, while the first two presidents of the Society are seen on the left.

FROM LEFT TO RIGHT: J. W. RICHARDS, H. S. CARHART, W. H. WALKER, H. P. TALBOT, E. R. TAYLOR, S. S. SADTLER, G. DROBEGG.

Digest of U. S. Patents.

Compiled by Byrnes & Townsend, Patent Lawyers, National Union Building, Washington, D. C.

Electric Smelting and Reduction Processes.

(*Continued from page* 208.)

590,673, September 28, 1897, Francis H. Soden, Chicago, Ill.

Supplements the ore-purification treatment of United States patents 290,213-4, and, in general, purifies ores of iron, gold, silver, copper, lead, etc. The ore, either before or after other treatment, is placed in an airtight receptacle having walls of firebrick or plumbago, through which pass horizontal resistance rods of platinum or carbon, arranged in two vertical series. The rods are connected to a dynamo, in multiple or in series, and the ore is thereby raised to a temperature of from 1200-1500° F. During the heating, superheated air or steam is first introduced, and the water, etc., in the ore is driven off. Superheated hydrogen is then passed into the vessel. The vessel is provided with a safety valve to relieve excess pressure.

591,355, October 5, 1897, Henri Moissan, Paris, France.

Produces titanium containing from 2 to 6 per cent of carbon by smelting a mixture of rutile 330 parts and carbon 96 parts, in a carbon crucible, with an arc of 1000 to 2000 amperes at 60 to 70 volts. The product, when reduced to a powder, is attacked by dilute hydrochloric acid, thus distinguishing it from titanium carbid and titanium nitride. If the mixture be smelted with the current from a dynamo of 45 horse-power only, or with an arc of 400 amperes at 80 volts, the product is a reddish brown titanonitrogen compound, containing from 79 to 80 per cent of titanium and having a density of 5.02. To decarburize the titanium, it is broken into pieces and resmelted with titanic acid, by an arc of the specified intensity. If the quantity of carbon in the mixture is increased, titanium carbid, TiC, is produced, having the color of galena and the density of 4.3. The titanium when alloyed with iron or nickel is soluble in molten iron, cast iron or steel. Titanium steel may also be produced by electrically melting the iron in a crucible and adding briquettes of a mixture of rutile, carbon and a glutinous binder.

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NEW YORK, JULY, 1905.

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Chemical Power Transmission.

In a recent address before the American Chemical Society on the electrochemical industries of Niagara Falls—a full account of which will be found elsewhere in this issue—Mr. FitzGerald mentions an interesting series of new commercial compounds now being made by the Roessler & Hasslacher Co. It is a series of peroxides with energetic oxidizing properties: calcium peroxide for the sterilization and preservation of foods, for ageing wines and for therapeutic purposes, magnesium peroxide for sterilizing water, zinc peroxide for antiseptic dressings, and fused sodium peroxide—or "oxone"—stable under ordinary conditions in air, but giving off oxygen in a most convenient manner when in contact with water. This commercial development is the direct outcome of systematic research work carried out in the research laboratory of the Roessler & Hasslacher Co.—certainly a gratifying result which should encourage our chemical manufacturers to establish more research laboratories. But this is a point on which we comment in a separate note in this issue. The development is also interesting in another respect. All the compounds mentioned have a high chemical energy content, and it is on this account that they are useful. A large amount of chemical energy is stored in them, it is "latent" under ordinary conditions, but it may be gotten out of them whenever or wherever necessity arises. By transporting compounds of high chemical energy content to a distant place, we transmit simultaneously the energy to that place. Chemical energy transmission, as we may term this, has some features in common with other systems of energy transmission and other features distinctly characteristic for chemical energy transmission alone.

The principle of the conservation of energy is at the bottom of all modern theories of physics and chemistry. The transformation of different forms of energy into each other is at the bottom of our modern civilization. All big engineering accomplishments of the last hundred years—from the steam engine up to electric power transmission, electric lighting, electric traction—are examples of a change of one form of energy into another. Under given conditions we may advantageously consider the energy as a product of two factors—the capacity factor and the intensity factor, to use the terms of Ostwald. The total amount of energy is the same, if the product of capacity factor and intensity factor is the same. But for the engineering applications it makes an enormous difference whether we make the capacity factor small and the intensity large, or the former large and the latter small. Thus in water-power developments we may consider the amount of water passing during a certain time as the capacity factor and the difference of level—the height of the fall—as the intensity factor. If we have two falls, one with a height 100 times greater than the other, but with a 100 times smaller amount of water, then the energy is the same in both cases, but the difficulties of developing the higher fall will be considerably smaller. With electric energy we have the current passing in a certain time—the coulombs—as capacity factor, the pressure—the volts—as intensity factor. Here, again, it is important for transmission purposes to use a high intensity factor, a high voltage, since the loss in transmission will then be a minimum With chemical energy we may consider the mass or weight of the reagent as the capacity factor and its specific chemical potential (which determines the activity of the reaction) as the intensity factor. For transmission purposes it is again preferable to have the capacity factor, the mass, small, to save cost of transportation. Thus for chemical power purposes those chemical compounds are chiefly of importance which have a very high energy content; and this is the reason why electrochemical engineering is so intimately connected with chemical power transmission problems. In many cases only electrochemical methods can cheaply produce those high-energy compounds, and for this reason many chief reagents for chemical energy transmission are products of electrochemical industries.

Chemical power transmission has two distinct advantages over other forms of power transmission. First, it is not bound to a fixed route; it requires no line wires. Second, it is not bound to a certain time; it is not necessary to use the transmitted energy at once or to provide special means for storing it. Chemical energy is in itself essentially stored, latent energy, and is used, therefore, in connection with other systems of power transmission for storing purposes, as in the case of storage batteries in electric power systems. Chemical energy storage and transmission is indeed capable of a surprisingly large number of useful applications, of which probably only a small number have so far been developed. The whole development is apparently only in the beginning, being an outcome of the development of electrochemical industries. Thus, the aluminothermic reaction with its steadily increasing number of practical applications is an example. Whether tramway rails are welded by thermit or the broken sternpost of a steamer is repaired by thermit, in any case the energy which is so conveniently developed there by the chemical reaction, has been transported with the aluminium to that place from the works of the Pittsburg Reduction Co. at Niagara Falls. If only thermit is on hand, we are enabled to produce molten soft steel at a very high temperature, wherever it is wanted and whenever it is wanted. Analogous remarks will be true of the silicothermic reaction when once introduced into practice. Any acetylene lighting plant, wherever it may be situated, utilizes the energy which was stored in the calcium carbide at the electric furnace plant. The calcium carbide gives the acetylene where it is wanted and when it is wanted. In the same way "oxone" will be found useful for the convenient production of oxygen. All these examples show the great convenience of the general method, and there are no reasons why a great many other examples of this general principle should not be found. But this convenience in use and this practical simplicity should not mislead anyone to think that the development of these methods has been a simple matter. Since all these reagents have a high energy content they are always willing to give their energy up—even when this should not be desired. Careless handling and transport of such reagents involves dangers which must be avoided in practical use. In all examples mentioned above much work has been necessary to bring the methods to such perfection as to become simple in practice and free from danger for general use.

Metallurgical Research Laboratories.

Metallurgical research laboratories are comparatively uncommon in this country. They, however, do exist, and their presence is encouraging. Years ago, when the iron business was called upon to furnish iron of regular composition for Bessemer plants, analytical laboratories were few and insignificant. Now, however, few iron furnaces or smelting plants of any size are without a laboratory for analyzing the samples of the works. This change has come about in spite of the opposition of the so-called "practical men." It is believed the research and experimental laboratory will win a similar victory. Wisely and liberally administered by a progressive man, such a laboratory is most valuable. Foolishly and extravagantly managed, it is worse than useless. Here, as in all else, the personal equation determines the fate of the enterprise.

Arguments for a research department are not many, but these few arguments are irresistible. A metallurgical laboratory has a splendid chance to test the methods of analysis and samplings. This is an important branch of metallurgy, as important and vital as are foundations in civil engineering. Such work should be removed entirely from the "works-laboratory," and can best be done by a well-trained and well-paid specialist. Next, the laboratory can make cheaply and accurately determinations of the physical constants of the reactions, worked on a large scale in the plant. The application of physical chemistry to metallurgy is too important to be discussed here, but using the abbreviated phrase—"physical metallurgy"—of one of our correspondents, we can state our opinion that the field will be found most prolific for hard, systematic original work in the next decade, and that from this scientific work will come results that can be measured only in millions of dollars. For instance, in a lead furnace, which is often as flirtatious as a coquettish woman, slags low in silver and lead are the essential. This fact depends largely on the temperature of formation, rate of formation, and temperature of fusion of these slags, and their resultant physical properties. If the slags were tested from time to time, and it were found by experiment which slags had the requisite physical properties, a large financial gain might easily ensue.

Again, there are often numerous new proposals that come up in the daily routine of work. To try these out on a large scale is expensive. With a regular research department this can be accomplished cheaply and accurately. Experimenting is the price of progress. The question, "will it work," is best answered, "try and see." To do this, requisite experimental work with certainty is hard without the regular organization. With an organization it is easy. Such other uses as the calibrating of pyrometers, checking scales, and so forth, need not be elaborated, for they are but minor advantages. The essence of the whole matter is the perfect welding of theory and practice. The research department should be in closest touch with the actual operations. Its head should be above all else a man of tact, a man who knows how to deal with young college graduates who are full of energy and theoretical knowledge and workmen who are conservative and have practical experience. He should have a good theoretical training, and he must have been "through the mill" himself. Needless to say, only a high-priced man is worth his pay in this place. He must be on the warmest and most sympathetic terms with the chief of the operating department and the chief of the business department, and should be subject to the former's orders. Such a man is the ideal, and hard to secure. Perhaps his rarity is the cause of distruct on the part of the directors of most concerns for anything called an experimental laboratory. We know of one instance where the laboratory was called the "experimental shop," to disabuse the directors' mind of this delusion. But it will not always continue thus. The cause is found in the fact that in the past experimenting has been done inefficiently. The results have been bad and the cost excessive. When once business men are convinced that a course is wise and productive of financial returns they are only too anxious to follow that course. This the experience in the electrical field proves beyond a doubt.

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Genesis of Chemical Elements.

In our April issue of last year we commented on J. J. Thomson's remarkable and beautiful scheme of an atom being a structure of shells of corpuscles, or negative electrons, within a sphere uniformly charged with positive electricity. Starting from a few fundamental suppositions, Thomson arrived, by strict mathematical analysis, at an explanation of the properties which chemical elements are known to possess, and especially of the periodic system of elements. While in this investigation Thomson's mathematical genius contributed largely to his triumph—one of the greatest ever achieved in theoretical physics—yet it should not be forgotten that Thomson did not follow the easy path of those who before him put forward speculations on the constitution of atoms. Thomson first proved the existence of his corpuscles in his researches on the nature of cathode rays, he determined their charge, their mass and properties. He proved them to be identical when produced from different kinds of matter and under different conditions, and then he devised the scheme of the rôle which they play in the constitution of matter. If his theory is right, if the atom is no longer indivisible, but represents itself a complicated system of corpuscles or electrons, and if the atoms of the different elements are all made up of the same kind of corpuscles and differ among themselves only in the number and configuration of corpuscles, then it is no longer absurd to consider the possibility of the tranmutation of chemical elements. As a matter of fact we are forced to such considerations by the established phenomena of radioactivity. The only satisfactory explanation, or at least adequate coördination, of the mass of facts found concerning radioactivity is that proposed by Rutherford and Soddy, that the atoms of the radioelements suffer spontaneous disintegration, and that each disintegrated atom passes through a succession of well-marked changes (metabolons), accompanied in most cases by the emission of alpha rays. In radioactive processes we are dealing with changes in the atoms themselves, and are watching the true gradual destruction of the radioactive atom. This itself is a metabolon; this means, it must have a parent element, and much circumstantial evidence has been produced in favor of the view that uranium is the parent of radium, especially by Boltwood. In any disintegration series, equilibrium will be attained and the relative quantities of the various members of the series, expressed in terms of the quantity of the parent element, must have a definite value, representing the inverse ratio of the rates of change of the transition form and parent element respectively. Boltwood, in analyzing a great many minerals, containing very varying proportions or uranium, has found in each case the expected proportionality to exist between the quantities of uranium and radium. Now Soddy, in a paper in the June issue of the *Philsophical Magazine,* produces direct evidence of the paternal relation of uranium towards radium. A quantity of uranium, originally free from radium, was proven to develop later the unmistakable characteristics of the radium emanation. This, taken together with the older spectroscopic proof by Ramsey and Soddy, of the production of helium from radium, enables Soddy to sketch the general outline of this disintegration series.

This disintegration series is as follows, the figures in parenthesis representing the atomic weight: The parent is uranium (238), metabolons are radium (225) and polonium (212); the final element is either lead (206.9) or bismuth (208.5). The alpha particles emitted during the different changes and causing the lowering of the atomic weight of the residue, form helium. In favor of the view that lead is the final product, Boltwood has cited the fact that lead persistently appears as a constituent of the uranium-radium minerals. Certainly, we are here in the midst of a most important phase of the development of natural science. There is much promise of important results from further conscientious and systematic experimental work, while with respect to the theory we may expect to see Thomson follow up his own explanation of the periodic system by a scheme of the evolution of chemical elements, just as Ernst Haeckel followed Darwin's work, and determined the pedigree of living nature. To those who judge everything from its immediate and direct commercial value, we may point out that radioactive processes are spontaneous. With none of our strongest egencies, like application of heat, etc., are we able to influence in the least the rate of change of radioactive processes. From the experimental evidence now available, there is no hope that the modern chemist will solve commercially the problem of the old alchemist of making gold from dirt. This will remain, as in the past, within the domain of the cyanide engineer.

American Chemical Society.

The thirty-second general meeting of the American Chemical Society was held in Buffalo, N. Y., from June 22 to 24, according to the programme published in our last issue, and turned out to be a full success in every respect. There was a very satisfactory attendance. The last printed list of members and guests who registered showed an attendance of 148.

The sessions for the reading and discussion of papers were held according to the programme, and were well attended, while the social functions were highly enjoyed.

On Thursday afternoon a large number of the members enjoyed a drive through many interesting parts of the city in carriages provided by the committee, while others visited the Gratwick laboratory. The same evening Mr. F. A. J. FitzGerald, of Niagara Falls, read a paper in one of the parlors of the Iroquois Hotel on the electrochemical industries of Niagara Falls, which brought out a large attendance. The paper, an abstract of which is printed on another page of this issue, was received with much enthusiasm. The new materials now made on a commercial scale in Niagara Falls—acetyvone, peroxides and sodium perborate—excited a great deal of interest; the fused sodium peroxide ("oxone") came in for the lion's share of attention. After the conclusion of Mr. FitzGerald's address an informal luncheon was served with the compliments of the management of the Iroquois Hotel, the convention headquarters.

Directly after the afternoon session on Friday, which concluded at 3 o'clock, the majority of the members took advantage of the opportunity to take a trip on Buffalo harbor in the largest and most modern of the fire boats. An excellent opportunity was afforded of seeing the Lackawanna Steel Co's plant from the water side, after which the boat was moored to one of the breakwaters, and an exhibition given of its fire-fighting qualities. A few of those present were unfortunate enough to be too close to the fire nozzle when the water was first turned on, and suffered somewhat in consequence. After this a short run was made up the Buffalo River, and the boat returned to her mooring at the foot of Main Street. About 100 of those present participated in the subscription banquet at the Hotel Iroquois on Friday evening.

Abstracts of the papers presented will be given later, when they have been published in the Journal of the Society. Besides Mr. FitzGerald's suggestive address, a paper, presented by Mr. Chas. B. Jacobs, should be of interest to our readers, since it deals with some observations on the *electrolytic deposition of alloys from mixed solutions.* The method described by Mr. Jacobs is now successfully used on a commercial scale by a lithograph company.

In the simultaneous deposition of two metals from a mixed solution of their salts, the difficulty is met that the solution attacks and dissolves the more electro-positive metal after deposition. This may be prevented by the use of two anodes, one of the more electropositive and the other of the more electronegative metal, connected to separate generators running at different voltages, the current returning through the cathode in the bath by a common third conductor to the generators.

Alloys of zinc and nickel and zinc and copper were deposited in this manner from neutral sulphate solutions. With cyanide solutions of copper and zinc a great variety of brass work was plated from the same, by changing the voltage on either anode in order to throw down a brass running high in copper and low in zinc, or vice versa.

Visit of American Members of Society of Chemical Industry to Great Britain.

On the evening of June 17, a joint meeting of the Chemists' Club, the Society of Chemical Industry, the American Chemical Society and the American Electrochemical Society was held in the convention hall of the Chemists' Club in New York City, the object being to wish God speed to those who were about to leave for England to attend the annual general meeting of the Society of Chemical Industry and participate in the trip through the country.

The meeting was very well attended, nearly 200 members being present. Mr. T. H. Kiessig, as chairman of the meeting, expressed its good wishes to the departing members and Dr. Wm. H. Nichols, the president of the Society of Chemical Industry, answered in a forceful and humorous speech. At the conclusion of his speech he made a remark which will certainly be applauded by chemical engineers all over the country; namely, that "in the autumn when we come back from England, we will direct our energies toward securing a fine new home for the Chemists' Club and for chemical engineers in the United States."

The programme for the week in London from July 10 to July 16, has already been published in our June issue, page 212. The party will then proceed on July 17 to Nottingham; July 18 to Rowsley, Chatsworth, Baslow and Manchester; July 20 to Chester and Liverpool; July 22 to Newcastle; July 23 to Durham and back to Newcastle; July 24 to Tynemouth and Edinburgh; July 26 to Glasgow.

From the programme as issued it is evident that the greatest endeavors are being made to make the trip a most memorable one, never to be forgotten by those who are lucky enough to participate in it. An alphabetical list of those participating in the visit to England is herewith given.

Mr. William A. Alpers and Miss Alpers, New York City; Mr. and Mrs. Theo. Armstrong and son, Philadelphia Pa.; Mr. F. E. Atteaux, Boston, Mass.; Dr. Leo. H. Baekeland, Yonkers, N. Y.; Mr. and Mrs. Eugene Barry, Ayer, Mass.; Dr. Charles Baskerville, New York City; Mr. Victor G. Bloede, Baltimore, Md.; Mr. Walker Bowman, New York City; Dr. E. Burkard, Garfield, N. J.; Dr. Nicholas Murray Butler, New York City; Dr. and Mrs. Eugene A. Byrnes, Washington, D. C.; Mr. and Mrs. Walter M. Chadwick and Miss Chadwick, Bayonne, N. J.; Dr. and Mrs. Charles F. Chandler, New York City; Mr. Alan A. Claflin, Littleton, Mass.; Dr. Virgil Coblentz, New York City; Dr. W. A. Converse, Chicago, Ill.; Mr. and Mrs. M. L. Davies, Bay City, Mich.; Dr. A. R. L. Dohme, Baltimore, Md.; Dr. William Dreyfus, New York City; Mr. B. P. Ducas, New York City; Mr. and Mrs. Chas. E. Eager, Boston, Mass.; Mr. Peter Fiebiger, New York City; Mr. Herman Frasch, New York City; Mr. and Mrs. Wm. S. Gray, and Miss Gray, New York City; Mr. Martin L. Griffin, Mechanicsville, N. Y.; Mr. and Mrs. Wilson Hanna, Boston, Mass.; Mr. Henry Hochstetter, Cincinnati, Ohio; Mr. William F. Hoffmann, Newark, N. J.; Mr. and Mrs. Henry Howard, Boston, Mass.; Dr. and Mrs. Alex. C. Humphreys, Hoboken, N. J.; Mr. Clarence M. Joyce, Arlington, N. J.; Mr. Alex. S. Kirkman, Brooklyn, N. Y.; Mr. H. J. Krebs and Miss Krebs, Wilmington, Del.; Professor Leonard P. Kinnicutt, Worcester, Mass.; Professor W. R. Lang, Toronto, Canada; Mr. Ashton Lee, Lawrence, Mass.; Dr. E. G. Love, New York City; Mr. and Mrs. Houston Lowe, Miss Lowe, Dayton Ohio; Mr. W. E. Lummis, Boston, Mass.; Miss Mary S. Lynne, Pittsburg, Pa.; Mr. and Mrs. Hugo Lieber, New York City; Professor and Mrs. C. F. Mabery, Cleveland, Ohio; Mr. F. Grant Marriott, Toronto, Canada; Mr. H. A. Metz, New York City; Mr. Ambrose Monell, New York City; Dr. Russell W. Moore, New York City; Mr. and Mrs. St. Clair McKelway, Brooklyn, N. Y.; Mr. William A. Nash, New York City; Dr. and Mrs. William H. Nichols and Miss Nichols, New York City; Mr. and Mrs. E. Remington Nichols and Miss Nichols, New York City; Dr. William A. Noyes, Washington, D. C.; Mr. and Mrs. W. F. Oburg and son, Boston, Mass.; Dr. T. J. Parker, New York City; Mr. Albert Plaut, New York City; Mr. and Mrs. G. G. Pond, State College, Pa.; Mr. C. Propach, Chicago, Ill.; Mr. Herman S.

Riederer, Newark, N. J.; Mr. B. E. Schlesinger, Boston, Mass.; Dr. H. Schweitzer, New York City; Professor A. W. Smith, Cleveland, Ohio; Mr. James Speyer, New York City; Mr. I. F. Stone, New York City; Mr. and Mrs. Geo. H. Taber, Pittsburg, Pa.; Mr. and Mrs. Henry D. Tichenor, New York City; Dr. and Mrs. Maxmilian Toch, New York City; Mr. W. H. Van Winckel, Toronto, Canada; Mr. I. D. Varshnei, Boston, Mass.; Mr. and Mrs. S. S. Voorhees, Washington, D. C.; Mr. Louis I. Waldman, Albany, N. Y.; Dr. Max Wallerstein, New York City; Dr. and Mrs. Martin Waldstein, New York City; Dr. Ida Welt, New York City; Mr. Clarence B. White, Youngstown, Ohio; Dr. H. W. Wiley, Washington, D. C.; Mr. R. C. Woodcock, New York City; Mr. and Mrs. C. B. Zabriskie, New York City.

American Foundrymen's Association.

The tenth annual convention of the American Foundrymen's Association was held in New York City, from June 6 to 8. The attendance was beyond the 200 mark, and ran up on the last day to almost 250. There was no local committee or entertaining committee, and nothing had been arranged on the official programme for the entertainment of the members on the evenings. Nevertheless, the convention was a most enjoyable affair, perhaps just on account of the informal nature of its social features. There is a very healthy professional sentiment which holds the members of the Association together, while not enough praise can be given to the splendid work of the indefatigable secretary, Dr. R. Moldenke, who is the real soul of the Association.

Mr. Thos. D. West (T. D. West Foundry Co., Sharpsville, Pa.) was elected president for the coming year. The next convention will be held at Cleveland.

The sessions were held partly in the Grand Central Palace and partly in Columbia University. A great many papers were presented. Those of metallurgical interest are abstracted below.

Fan Blowers vs. Positive Pressure Blowers.

A paper, by Mr. H. E. **Field**, on this subject was based on a report made to the Pittsburg Foundrymen's Association last year by a committee appointed by that association. Four tests were made with two Sturtevant fans and one Connersville positive-pressure blower, supplying a cupola which was lined to 54 inches. The blower and the fans were electrically driven. The electrical power, the motor speed, and the blast pressure were measured during four regular cupola heats.

The main point of difference between the two types of machines is this: With the positive-pressure blower the amount of air delivered is not dependent upon the size of the opening. With the fan this is the case. A fan will deliver a maximum amount of air only as long as there is a free opening, and when this opening is made smaller the fan will deliver less air. As the opening grows smaller, and consequently the amount of air delivered grows less, there will be a corresponding less amount of horse-power required to run the fan. The horsepower consumed then in driving a fan is in a measure proportionate to the amount of air delivered by the fan. The action of a positive pressure blower is exactly opposite, since it insists upon a given amount of air being delivered per revolution. The larger the opening the less horse-power will be required to deliver a given amount of air through that opening.

With a fan blower the piping should be as short and free from curves as possible. With the positive-pressure blower isis is not so important.

The fan consumes the greatest power at the beginning, and least power when the blast pressure has arisen to its maximum; after that to the end of the run the power required by the fan increases again slightly. On the other hand, the blower consumes its least power at the beginning of the run; the power required then rises very rapidly, and reaches a maximum over twice as great as the initial, at the same time as the blast pressure reaches its maximum. In the last part of the heat the blower power decreases, ending with about 70 per cent of the maximum.

The electrical energy consumed by the motor for the blast in kw-hours per ton of iron melted was found to be in the four tests as follows:

With fan (two tests), 3.41 and 2.76; with positive-pressure blower, 2.86 and 2.63.

In the second fan test very light scrap was used, of such shape that it kept the cupola open and gave freer passage for the air. Under these circumstances the performance of the fan was practically equal to that of the blower.

Mr. Field concludes that an escape valve, or relief valve, should not be used with positive pressure blowers, since this destroys their ability to keep the cupola open, and makes the blower equivalent to a fan.

The paper elicited considerable discussion, in which especially Messrs. F. W. Stickle, H. E. Field and D. Spence participated.

Aluminothermics.

Three papers were presented on practical applications of the aluminotherimic reaction. Mr. W. M. **Carr** gave a demonstration of the properties of thermit, and its use in welding and the like, while Mr. J. F. **Webb** discussed the use of thermit in a railroad shop, and Mr. G. N. **Prentiss**, the repair of driving wheels by thermit. It was reported by Mr. Webb that a broken locomotive frame was in one instance repaired in about 20 hours, counting from the time the engine came into the shop until the time it left to take out a train.

In the discussion of Mr. Carr's paper the use of "nickel-thermit" was brought out for introducing nickel into molten cast-iron, in order to increase the resistivity of cast-iron tanks against chemical action and to improve the strength, density and fineness of grain. Mr. E. Stütz stated that nickel thermit is now being used successfully in Germany in connection with the manufacture of alkali pots, the amount of nickel thus introduced into cast iron being about one-half of 1 per cent. Nickel thermit is now being introduced in this country for the same purpose.

A practical application of nickel thermit and titanium thermit was made on the occasion of the visit of the Association to the works of the International Steam Pump Co., in Harrison, N. J. A wide range of castings is produced at this plant, both in cast iron and brass. A feature of the visit was the pouring of a 1000-pound casting after nickel thermit had been introduced into the ladle. About 1 per cent, or 10 pounds of nickel thermit was used, this being poured from a small ladle into the ladle containing the cupola iron, when the latter was about half filled. Then the remaining iron was tapped out, and, lastly, there was added to the ladle of molten iron one-sixteenth of 1 per cent of titanium-thermit (a mixture of titanic oxide, aluminium and iron oxide). The object of the latter is to cleanse the iron, particularly to remove nitrides, and thus increase the density of the iron; and above all to stir the molten bath so thoroughly that the nickel which was introduced before is uniformly distributed through the molten bath.

Properties of Alloys.

A paper by Mr. P. **Longmuir**, Sheffield, England, on "Variations in the Properties of Alloys," was contributed by the *Metal Industry*, and was read by Dr. G. P. Scholl. It was illustrated by lantern slides.

The author gives an account of a series of tests on the influence of the casting temperature on the strength and ductility of brasses and other alloys. The highest tensile strength and ductility were obtained when the alloy was cast at a moderate or "fair" temperature, *i. e.*, one which is neither too high nor too low.

The chemical analyses in all cases showed exactly the same composition for all bars cast from a single ladle, although the first bars were cast very hot, while the last were cast when the brass was quite dull; this shows that there is no burning-out of zinc during the process of pouring. Microphotographs of sections, however, showed the cause of the difference in strength ductility, since the finest and best interlocked grain results when the casting temperature is fair. From the experience of the author it would seem that when once an error has been made in using too high or too low a temperature of casting, no subsequent heat treatment will bring up the ductility and tensile strength to the possible maximum

Determining Silicon in Pig Iron and Cast Iron.

Mr. H. E. **Diller** presented the report of the metallurgical section of the Association. The committee proposes the following method of determining silicon in iron:

"Weigh 1 gram of sample, add 30 c.c. nitric acid (1.13 specific gravity); then 5 c.c. sulphuric acid (conc.). Evaporate on hot plate until all fumes are driven off. Take up in water and boil until all ferrous sulphate is dissolved. Filter on an ashless filter, with or without suction pump, using a cone. Wash once with hot water, once with hydrochloric acid, and three or four times with hot water. Ignite, weigh and evaporte with a few drops of sulphuric acid and 4 or 5 c.c. of hydrofluoric acid. Ignite slowly and weigh. Multiply the difference in weight by 0.4702."

The committee recommended this method as one that sacrificed shortness to some extent to insure accuracy, and that left little to the judgment of the chemist. While it was recognized that it is almost impossible to get chemists to use a standard method in their daily work, the one proposed would serve as a check method in case of dispute between laboratories or between buyer and seller.

Effect of Manganese in Low-Silicon Cast Iron.

Mr. H. C. **Loudenbeck** presented results of tests on the effect of manganese on the chill and fraction of cast iron having a low percentage of silicon. He reached the following conclusions. Manganese may be used to an advantage in low silicon and chilling iron in the following cases: In mixtures where the percentage of scrap is large and the sulphur necessarily high (this will occur in a car-wheel mixture where usually a large portion of old metal is used), the result of this increase in manganese would be lower sulphur, lower combined carbon, less chill and greater strength.

Very often chilled plates are required, having hard-chilled faces and soft backs suitable for planing. Manganese, added in the right proportion, will reduce the tendency to mottle, and make a comparatively soft graphitic back.

In all cases where chilling irons are melted in a cupola and the sulphur is over 0.07 per cent, the iron can be strengthened by the use of ferromanganese or pig iron having a high percentage of manganese.

There are, however, some cases where the manganese should be kept low. In the manufacture of large hydraulic cylinders it is necessary to have a close, mottled iron to withstand the pressure and prevent leakage. If the manganese is too high this mottled structure is replaced by a coarse graphitic structure, which is not satisfactory for this class of work.

American Institute of Electrical Engineers.

The twenty-second general meeting of the American Institute of Electrical Engineers, held from June 19 to 23, at Asheville, N. C., had attracted not less than 250 members from all parts of the country. It was the first meeting held by the Institute in the South. The technical programme was excellent; twenty-four papers were presented and elicited good discussion. The social features of the convention were a reception and ball on the evening of June 20, a trolley ride to the Asheville Golf Club on the afternoon of June 21, and on the same evening a banquet, at which nearly 200 were present. Mr. T. C. Martin acted as toastmaster. President Lieb spoke in very complimentary terms on the South, and Mayor Barnard, of Asheville, replied in terms equally complimentary to the Institute. The president-elect, Dr. Schuyler Skaats Wheeler, responded to the toast "The Institute," Dr. C. P. Steinmetz to "Theory and Practice," Dr. S. Sheldon to "Technical Institutions," Mr. J. H. Finney to "Industrial Development of the South," and Dr. F. A. C. Perrine to "The Utilization of Natural Resources Through Electricity."

The only two papers of chemical or electrochemical interest are briefly abstracted below:

Constant-Current Mercury Arc Rectifier.

A paper, by Dr. C. P. **Steinmetz**, dealt with the use of a constant-current mercury arc rectifier system for operating direct-current series arc circuits from an alternating-current constant potential supply. The series of arc lamps is connected at one end with a tap brought out from the middle of the secondary of a constant-current transformer. The two terminals of the secondary coil are connected through two induction coils with the two graphite anodes of the mercury rectifier tube. The common mercury cathode is connected through an induction coil with the second terminal of the series of arc lamps. Two auxiliary mercury anodes near the cathode are used for starting the discharge (as described in the recent paper of E. Weintraub, May issue, p. 184).

A system of this kind has been in regular service for over a year, supplying a constant current of 3.8 amps. to twenty-five arc lamps in Schenectady. In this small plant of only 3.9-kw output an efficiency of 80 per cent and a power factor of 70 per cent is reached, between the constant-potential alternating current received from the primary distributing mains and the constant direct current sent into the arc circuit.

A New Carbon Filament.

A paper on this subject was presented by Mr. **John W. Howell**. It is well known that an ordinary incandescent filament in an evacuated lamp may be heated to excessive temperatures by the application of high current without producing any beneficial changes in the filament. To produce the effects described in the paper, the heat is externally applied to treated filaments at atmospheric pressures. The simple application of the highest attainable temperatures to the plain carbonized cellulose fiber, *i. e.*, the so-called "filament base," is also incapable of producing any considerable changes in the filament. This high temperature treatment of the base is, however, a beneficial step in the process described.

In attempting to graphitize carbon filaments it was discovered that treated filaments undergo very remarkable changes when subjected to the highest possible temperature of an electric resistance furnace. The furnace consisted of a carbon tube, held at the ends by large water-cooled copper clamps; the tube was imbedded in powdered carbon to prevent its combustion. The filaments were packed in small, cylindrical carbon boxes, which were fed into the heated tube. The temperature was usually between 3000° and 3700° C.

Under such firing the ordinary treated filament changes in appearance; its graphite coating looks as though it had been melted, and its specific resistance is greatly reduced. The resistance of one of these filaments, measured at ordinary temperature, may be reduced as much as 80 per cent by the firing.

The changed nature is also indicated by toughness and flexibility. It seems that the change is practically all in the treated coating and not in the base.

The most effective carbon filament now in general use operates at 3.1 watts per candle, and under accurate test conditions gives a useful life (to 80 per cent of its initial candle power) of about 500 hours. At the present time graphitized filaments that give the same useful life at about 2.5 watts per candle power, are being produced with a fair degree of uniformity.

The Electrochemical Industries of Niagara Falls.

An address on this subject, delivered on June 22 by Mr. FRANCIS A. J. FITZGERALD, of Niagara Falls, at the Buffalo meeting of the American Chemical Society, was highly interesting and suggestive in two respects; firstly, for the reason that Mr. FitzGerald treated the subject from an evolutionary point of view, observing the effects of those most important factors in evolution, the struggle for existence, the influence of environment, etc., in the development of the Niagara electrochemical industries, and secondly, because several new electrochemical products made commercially at Niagara Falls were here publicly discussed for the first time.

ALUMINIUM.

The aluminium industry is now about ten years old in Niagara Falls, and Mr. FitzGerald discussed briefly the enormous growth of the Pittsburg Reduction Co. in this period, which necessitated the erection of further large plants at Massena and Shawinigan Falls. Now more aluminium is manufactured in America than in all the rest of the world combined.

Mr. FitzGerald emphasized the effect which is produced on certain details of the process of manufacturing aluminium by the environment of cheap electric power. An expensive part of the manufacturing process is the preparation of the raw materials. Thus the bauxite used must undergo an elaborate chemical treatment to remove objectionable impurities such as iron, silicon, etc., as these would enter into the metal. Mr. C. M. Hall, in looking for an improved method of purifying bauxite, proposes to use an electric furnace where the bauxite is fused and aluminium is added to reduce the impurities, which then sink to the bottom of the molten bath in the elementary state, leaving the bauxite pure.

Another detail of great importance and no little expense in the manufacture of aluminium is the carbon electrodes. Here again the influence of cheap power is important, for Mr. Hall has developed a method of baking carbons in an electric furnace. The process has been described in his patents (see our Vol. I., pp. 30 and 467), and briefly consists in moulding the carbon electrodes from a mixture of carbon powder and pitch, then placing the articles in a furnace around a central carbon resistor, through which the current is passed. The heat thus generated in the resistor is conducted to the surrounding carbon electrodes, and they are raised to a higher temperature than can be economically obtained in an ordinary furnace using fuel. In this way an important advantage is gained, since the electrical conductivity of carbon and its resistance to oxidation are functions of the temperature to which it has been raised.

CARBORUNDUM AND ALUNDUM.

In passing over to carborundum Mr. FitzGerald gave a review of Mr. Acheson's work and pointed out how the large scale operation, beginning at Niagara Falls with a 750-kw furnace, permitted the cheaper production of carborundum, so that it could be put on the market at a price somewhat near to that of other abrasives.

At first the production of carborundum had little effect on the abrasives market, this being due in part to the costly experimental work necessary before the new material could be put into marketable form and in part to its price as compared with other abrasive materials, such as emery. Finally, however, the manufacturing difficulties were overcome and the consumer learned of the advantages of carborundum. The important growth of the carborundum industry is illustrated by the figures showing the annual production for the last fourteen years.

The production in 1891 and 1892 combined was ½ ton; 595 tons in 1896, and 2380 tons in 1903. The carborundum plant has recently been increased in size by the addition of another unit of 1500-kw capacity, so the future annual production may be estimated at 3970 tons.

Keeping in mind that carborundum must do quicker work and more work in order to compete successfully with other abrasive materials, the large production of recent years must have had a profound effect on the market for materials like emery, and thus the struggle for existence among abrasive materials has become more severe. The severity of the competition has undoubtedly resulted in a general improvement in manufactured abrasives, but in addition to this it has stimulated the development in this country of the industrial manufacture of other artificial abrasives.

Until the advent of carborundum, the hardest abrasive known, with the exception of the diamond, was corundum. But the corundum deposits were not very large and were difficult to work, and so it was that in Europe the manufacture of artificial corundum by fusing bauxite in the electric furnace developed. This manufacture was not developed in America until the competition of carborundum was felt. Now, however, artificial corundum is manufactured on a large scale at Niagara Falls by the Norton Emery Wheel Co. (see our Vol. I., p. 15), and put on the market under the name of "alundum." Mr. FitzGerald mentioned here that another artificial abrasive called "adamite" is also used in this country. It may be described as an artificial emery.

In preparing carborundum for the market it is crushed, and this results in the formation of a large quantity of powder, far more than is demanded in the abrasive trade. It was found that this powder might be used in the manufacture of steel as a substitute for ferrosilicon. The commercial material contains about 62 per cent of silicon, and this is much higher than can be obtained in blast furnace ferrosilicon. In this way a market for carborundum powders was opened, and probably a good market until the arrival of electric-furnace ferrosilicons. These are now manufactured so cheaply that it is improbable that the market for carborundum is at all as good as it was, for carborundum has the disadvantage of containing about 30 per cent of carbon. A probable effect of this new condition will be the development of the manufacture of ferrosilicons, using carborundum as a raw material, since it requires a large amount of energy to reduce silica and form high-grade ferrosilicon, so that it may be found more economical to start with carborundum instead of silica.

At a lower temperature than that of the formation of carborundum, an amorphous substance is formed. This has the same composition as carborundum, but is not crystalline, and was first described by Schützenberger, who made it in a wind-furnace from a mixture of silicon and carbon. Amorphous silicon carbide is formed in the carborundum furnace outside the crystalline zone. As might be expected, it is a very refractory material, for as the temperature is raised beyond that of the temperature of its formation, it does not fuse, but is simply converted into carborundum. This by-product of the carborundum furnace has recently been put on the market under the name of "carborundum fire sand," and forms a very good refractory under certain conditions. It is not suitable, however, for cases in which a powerfully oxidizing flame is used, since in this case the carbon is oxidized to carbon monoxide and the silicon to silica.

ARTIFICIAL GRAPHITE.

Mr. FitzGerald discussed the growth of the artificial graphite industry. It was first mainly used for electrodes in electrolytic cells, and this use is still increasing. More recently the manufacture of artificial graphite for other purposes, such as paint, dry batteries, stove polish, lubricants, etc., has assumed large dimensions. The fact that carbon articles which have been converted into graphite are so far changed in physical structure that they can be readily machined, while their resistance to oxidation is increased and their electrical resistivity decreased, makes them a very useful article in the construction of electric furnace terminals and for other purposes in electrical engineering.

The growth of the manufacture of artificial graphite was shown by Mr. FitzGerald by the following figures: In 1897 the production of graphite amounted to 81 tons, in 1900 it was 440 tons, and in 1904 it was 1333 tons; but the sales of graphite last year amounted to 1624 tons, so that an increase of the plant has become necessary. For this purpose the International Acheson Graphite Co. is installing a 1500-kw unit, which will give an output of 2666 tons of graphite per year.

Calcium Carbide and Acetylene.

Mr. FitzGerald briefly pointed out that since the validity of the Willson patents has not been seriously attacked, the manufacture of calcium carbide is not exposed to competition in this country. This is quite different from the situation abroad. On the other hand, in the use of calcium carbide the competition is severe, if we may judge by the number of factories making acetylene generators of various types and from the voluminous patent literature dealing with the subject. Mr. FitzGerald gives some interesting notes on a factory at Niagara Falls which has recently started, or is about to start, on the preparation of calcium carbide in a special form for use in acetylene generators. The factory is owned by the Acetyvone Manufacturing Co., and will turn out carbide in the form of sticks or blocks of a certain size. These are made by grinding the carbide to powder, mixing with a suitable binding agent and then moulding the mixture in the desired form. Mr. FitzGerald exhibited some specimens of acetyvone.

The advantages of this form of calcium carbide are that it can be exposed to moist air and handled without suffering appreciable decomposition; it can be caused to give off gas with great regularity when used in a suitable generator, and when the generation of gas is to be stopped, this can be done practically instantaneously.

The generator used with this form of carbide is simple, consisting of a water reservoir, a cylinder, which acts as a reservoir for the gas, and the holder for the carbide stick. The holder is movable and connected with a small lever, which lowers the carbide stick so that it comes in contact with the surface of the water and at the same time opens the gas pipe. The acetylene gas then passes into the reservoir and through the pipe to the burner. When the lever is lowered the gas pipe is closed and the carbide stick raised out of the water, when the generation of gas stops almost instantly.

Ferro-Alloys.

The fact that Europe is a long way ahead of this country in the production of ferro-alloys by electric furnace methods was shown by Mr. FitzGerald to be due to curious circumstances. The great boom in calcium carbide in Europe about ten years ago resulted in the erection of an astonishing number of plants and in great overproduction. Then followed the celebrated Bullier patent suit, which ended, so far as France was concerned, in the establishment of the validity of the Bullier patent. This resulted in many of the calcium carbide factories stopping the manufacture of that material. The manufacture of ferro-alloys was then taken up, and later on the manufacturers turned to the production of steel in the electric furnace.

This brings out clearly the important influence of environment on the evolution of industries, for in spite of the pioneer work of the Cowles brothers, between 1880 and 1890, little advance has been made here in this particular respect. Some years ago there was a plant at Niagara Falls, manufacturing ferro-chrome, but this was burnt down, and the manufacturers sought a more favorable environment, because, in spite of the great power development at Niagara Falls, the cost of power is still very high compared with prices elsewhere, especially compared with European plants.

Recently at Niagara Falls, Messrs. Becket and Saunders have done some excellent work at the Niagara Research Laboratories on ferro-alloys. Mr. FitzGerald exhibited some specimens of ferro-alloys made at these laboratories (analyses of some specimens of alloys made by this company were given on page 133 of our April issue).

Vanillin.

In passing over to this subject Mr. FitzGerald remarked that a great deal of work has been done on the use of the electric discharge for bringing about various reactions, one of the most interesting being the oxidation of nitrogen. This problem, in spite of much study, does not as yet seem to have resulted in tne development of a satisfactory commercial process for the manufacture of nitric acid from the air. The general result of the researches seems to show that the two important conditions of the reaction are temperature and pressure. In his discourse on "Vitrified Quartz" at the Royal Institution, March 8, 1901, Mr. Shenstone said: "It is interesting to record the fact, first observed by Mr. Lacell, that nitric peroxide may be produced by heating a mixture of oxygen and nitrogen above the melting point of platinum in tubes of silica."

Another interesting reference of Mr. FitzGerald related to the "Proceedings of the Royal Society," Vol. 60, where there is a paper by FitzGerald and Wilson, read December 17, 1896, on the temperature of the electric arc. The arc was formed inside a closed vessel so that its behavior under high pressure could be observed. The authors say: "We found that at any rate some of the radiation, and possibly a great deal of it, was cut off by the formation of what appeared to be red fumes of NO_2. We found no absorption from this cause so long as the pressure was nearly atmospheric, but at about 100 pounds pressure this gas was formed with great rapidity. We easily confirmed our belief in the presence of this gas by its well-known absorption spectrum.

"Lest heat dissociation might cause an apparent increase in the amount of NO_2, we tried heating some of the gas in a flask. We observed that when hot the brown fumes became a golden yellow, and the absorption bands nearly disappeared; so that the heating could not have been the cause of the apparently enormous production of NO_2 at high pressure."

It seems probable that if a commercial process of making nitric acid from the air is eventually worked out it will depend upon the combined effects of high temperature and pressure. In this connection Mr. FitzGerald called attention to a description in the *Gazzeta* for April 3, by Mr. E. Rossi, of an apparatus in which nitric oxide is formed by heating oxygen and nitrogen under high pressure with a heating resistor similar to the glower in a Nernst lamp.

However, if the use of electricity for the synthesis of oxides of nitrogen has not yet reached the industrial stage, the production of ozone for commercial purposes certainly has. This is illustrated in Niagara Falls at the works of the Ozone-Vanillin Co., which manufacture vanillin. The raw material, oil of cloves, contains a high percentage of eugenol, and this is readily converted into the isomeric compound iso-eugenol, which in turn is transformed to vanillin by oxidation.

At the works of the Ozone-Vanillin Co. the oxidation of the iso-eugenol is accomplished by means of ozonized air. For this purpose high-tension step-up transformers are used, and the ozonized air produced in the usual way by means of the silent discharge. The advantage of using a process of this kind for oxidizing purposes is obvious, avoiding, as it does, troublesome chromium or manganese residues.

Chlorine and Caustic Alkalies.

The first chlorine and caustic soda factory was that of the Castner Electrolytic Alkali Co., which owns the Castner patents and now uses 700 hp. Mr. FitzGerald remarked that there is no process he knows of which turns out a purer form of caustic soda than that working with the Castner cell, and for most analytical work it is preferable to the caustic soda specially prepared for that purpose.

In the Acker process fused sodium chloride is electrolyzed. In the Roberts Chemical Co's process a solution of potassium chloride is electrolyzed with the production of caustic potash and chlorine. The most interesting fact, however, in connection with the two last named companies is that the struggle for commercial existence has had a very stimulating effect on the development of new uses of electrolytic chlorine, since the manufacture of bleaching powder is no longer very remunerative on account of overproduction. Thus, Mr. Charles E. Acker, of the Acker Process Company, has worked out processes for the manufacture of carbon tetrachloride, tin tetrachloride and tin oxide. At the Roberts Chemical Co's works the hydrogen and chloride from the cells are combined to form hydrochloric acid. (In connection with this the two articles by Dr. F. Winteler and Dr. O. Nagel, in our Vol. II., p. 339, and Vol. III., p. 16, should be of interest.)

"Oxone."

An interesting use of caustic soda at Niagara Falls is for the production of sodium. The process used was developed by Mr. H. Y. Castner, and is carried on at the works of the Niagara Electrochemical Co. This industry has grown considerably, and gives promise of extensive growth in the future through the development of certain important derivatives recently made by the Roessler & Hasslacher Chemical Co.

An important compound made from metallic sodium is sodium peroxide. Dr. Hans Foersterling and Mr. Herbert Philipp conceived the idea of fusing sodium peroxide in any desired form, so that the solid cakes of this material might be used as generators of oxygen, much as calcium carbide is used for the generation of acetylene. This was found to be practicable, and Mr. G. F. Brindley, of the Niagara Electrochemical Co., worked out a process for the commercial manufacture of "oxone," as the new material is called.

Mr. FitzGerald exhibited several specimens of oxone. These were in the form of square blocks, weighing 70 grams each, and were specially made for generating oxygen in stereopticon work. According to Dr. Foersterling, it is the intention of the Roessler & Hasslacher Co. to manufacture also a round cake weighing about 180 grams. When one of these cakes is put into water, pure oxygen is immediately generated, 1 pound of oxone yielding 59 litres, or 2.08 cubic feet of gas. If the oxone is used in an ordinary gas generator, such as those made for laboratory purposes, oxygen gas is generated in a very convenient way. Oxone seems to be remarkably stable when exposed to air under ordinary conditions. Thus, according to Dr. Foersterling, in a test in which a cake of oxone, weighing 178 grams was exposed in the open air on a glass dish, the consumption of the oxone was at the rate of 0.618 grams per hour which corresponds to an hourly generation of 90 cubic centimeters of oxygen.

On account of the absorption of carbon dioxide by the sodium hydrate formed on the decomposition of sodium peroxide, it is proposed to use oxone for supplying oxygen and absorbing carbon dioxide in cases such as arise in the problem of supplying oxygen and absorbing carbon dioxide in submarine boats. While sodium peroxide in the ordinary form might be used for this purpose it would not be so satisfactory on account of its relative instability.

In addition to oxone the Roessler & Hasslacher Chemical Co. is now manufacturing an interesting series of peroxides, specimens of which were exhibited. Calcium peroxide, as manufactured, contains from 60 to 65 per cent of CaO_2. It is insoluble in water, but liberates oxygen in the presence of moist organic matter or by heating. It is proposed to use calcium peroxide for the sterilization and preservation of foods, since its presence would have no harmful physiological actions. It may also be used for ageing spirits and wines. Finally, it may be used in therapeutics for various purposes.

Magnesium peroxide contains from 25 to 32 per cent of MgO_2, and may be used in a similar manner to calcium peroxide. It may be employed also for sterilizing water. Thus, according to Dr. R. von Foregger, a tablet containing 0.1 gram of magnesium peroxide and 0.5 gram of citric acid dissolves readily in a litre of water and completely sterilizes it in a few minutes.

Zinc peroxide contains from 50 to 60 per cent ZnO_2. It may be used as an antiseptic dressing. Sodium perborate contains 100 per cent of $NaBO_3 + 4H_2O$. This salt, when dissolved in water, forms hydrogen peroxide, and Dr. Foregger has been able to make a 50-volume solution of hydrogen peroxide in this way. The great advantage of using a substance of this kind for bleaching purposes is obvious.

Mr. FitzGerald concluded his address by pointing out how all these electrochemical industries at Niagara Falls have been greatly stimulated by the struggle for existence and the influence of environment, and how these industries give strong evidence of vitality in the continuous adjustment of internal to external relations.

The Development and Present Status of Cyanidation in the United States.

By William H. Davis.

The cyanide process is one of the greatest achievements ever realized in the metallurgy of gold. Its influence has been felt in almost every mining region of the world. Its material addition to the world's wealth has been instructive, beneficial and stimulating to a large number of other industries. The process has, like other chemical operations, a large number of points of nicety; some have been developed and others have not, and in some respects the process is still in an unfinished state. Undoubtedly lower costs will be realized in ore treatment and a higher percentage of values extracted, but sufficient results have been realized at this time to place the process far in advance of all competitors in matters of cost of operation and bullion production.

Historical.

As early as 1867 a patent was issued to J. H. Rae, covering a proposed process for the extraction of gold and silver from their ores by the use of potassium cyanide to be aided by the use of the electric current which was to simultaneously precipitate the gold and silver upon a copper plate. Other patents were issued in 1880 and 1881, but no commercial success was achieved by these attempts. In 1885 a United States patent was obtained by J. W. Simpson, the proposed scheme being to use a solution containing 3 per cent potassium cyanide and 0.2 per cent ammonium carbonate. This treatment was set forth for ores carrying gold, silver and copper. It is not recorded that working tests were ever made, though experimental work was done, and in fact is still being done along this line. United States patents were applied for and finally granted to Messrs. MacArthur and Forrest in the year 1889, some two years after the issuance of patent by the government of Great Britain. These patents called for the use of a dilute solution of potassium cyanide, and without any other chemically active agent. While the process was anticipated before the issuance of these patents, and although other reagents were used from almost the beginning of the work, namely, alkali, still this was the dawn of the new era in gold metallurgy, and modern practice depends upon the general principles promulgated by MacArthur and Forrest.

The process was introduced into this country, as a working method, soon after the American patents were allowed, by the agents of the Cassel Gold Extraction Company, and was in some respects, pertaining to the recovery of the zinc box products, a very immature method. This was, however, soon overcome and we find the first applications being made in the years 1891 and 1892. The first successful cyanide plant to be operated in the United States was in the year 1892, and was erected and operated by Mr. Alonzo Coan in Boulder County, Col. for the Livingston mine. The experimental tests were made during the latter part of 1891 in the city of Boulder, and the following spring work was inaugurated on the mill. This mill was operated for the greater part of ten years by this party.

A few words as to these operations will be of interest and therefore will be given.. The Livingston mine lies in the telluride belt of Boulder County, and the ores therefrom are telluride ores. The ore was, however, oxidized for some distance below the surface. This natural process of roasting leaves the gold (for the values were almost wholly gold) in the so-called "rusty gold" condition, *i. e.*, with a minute layer of oxides of base metals occurring on the surface of the gold. It is presumed that the whole gold content of this ore was originally of the telluride formation. This ore was treated to the depth of 120 feet. At this depth live (telluride) ores were encountered and the farther cyaniding of the ores would require roasting. At this time the property passed into other hands. The operations of this mill comprised crushing the ore dry, with rolls to 12 mesh, then loading into vats, a five-day treatment with cyanide solution of 0.25 per cent strength, principally maceration, then washing. The average value of this ore is given me as approximately $10.00 per ton (though some ore of $4.00 value was treated at a profit); the average value of the discharged tailings were $0.80, which gives the extraction as 92.00 per cent. This plant treated 10 tons of ore per day, and was in every respect a successful plant.

Following these operations comes the great Mercur plant which treats, when in full operation, over 1000 tons per day, and is the largest dry crushing plant in the world. From this time new plants came into existence so rapidly that no attempt is made at this time to enumerate them. In fact cyanidation is so well established, so successful, so fruitful that frequently stock jobbing schemes are floated where there is no ore to treat.

CRUSHING.

There are two general methods of reducing the ore to the required fineness in vogue in this country, one dry by breakers and rolls, the other wet and usually (though perhaps not wisely, since better methods are now practicable) by the stamp, followed first by concentration, and last by cyanidation where sufficient values are present in the tailings to warrant the additional expense. Where the ore is to be treated as a whole by either cyanidation or chlorination, the first method is used, but where pyrite is contained in greater amounts than, say, 3 per cent (depending upon conditions of the gold), wet crushing is usually employed, followed by concentration as above stated. The American practice has been to ship the concentrates to smelters. The tailings are, for the greater part, treated by cyanide for the reason that few concentration mills make quantitative separation of the values from the quartz. There are some objections to the day crushing method from the amount of slimes or dust made, although this seldom amounts to more than 2 per cent of the total ore crushed, and in well regulated mills much below this. Dust losses, whatever they may be, are insignificant, when compared to the slime losses with wet crushing, especially when the stamp is used, as in this case the slime seldom falls below 10 per cent, and often exceeds 40 per cent. However, this matter of slimes has received special treatment and will be discussed later.

The secret of crushing dry ore with the least amount of slimes or dust is frequent and thorough screening. For this purpose the vibratory or impact screen offers advantages over the trommel type for the reasons that the smaller particles seem to adhere to the larger ones, which attraction is broken by the impact, by the momentum being added to gravitation. By this method a better separation is effected, and American practice tends toward the impact screen. In wet crushing this result could be realized through the use of a greater amount of water; but this is objectionable when concentration is employed, as a large amount of water intercepts efficient results. The proper size for crushing is usually determined by sizing the product before and after treatment, assaying separately the different sizes and plotting curves for the values and extraction.

ALKALINITY.

This important topic received but little consideration from the early workers and some indifference is manifest to-day, but this is fast dying and the matter is becoming a considered factor in the economic treatment of ores by the cyanide process. Perhaps no part of the process is of such vital importance to successful work as this one. That a protective alkalinity should be maintained is a matter that will appeal to the chemist upon first consideration. He has only to remember the low acidity of hydrocyanic acid, and therefore the ready displacement by even the carbonic acid radical, which is ever present, to suggest the necessity of free alkali to retard this action. Other reasons would follow with experience, namely, the action of the double salt for which the compound KCy has great tendency. In actual work, with cyanide solutions applied to ores, perhaps the first avenue of escape is the double salt formation which, being soluble in excess of free alkali, calls for protective action.

To the physical chemist the matter of hydrolysis becomes important, which also calls for protective alkalinity, for, according to the present theory of solutions, there is always a fixed relation between the ions and the undissociated portion of the solution. Therefore, if a compound, having in common an ion with the first, be added, the dissociation of the compound is driven back. Now as there is in common between the alkaline cyanides and free alkali, the alkali radical this would prevent, at least to some extent, the dissociation of the cyanide compound which is fatal to the economic use of the process. As evidence of this it has been observed by the writer that the odor of cyanide from solutions (which is hydrocyanic acid) is much more pronounced when no free alkali is present. Without doubt the ingredients of the ore treated would have a marked effect upon this, but still the relative difference would be present.

SELECTION OF ALKALI.

In practice I have found the use of lime to have advantages over that of the more caustic alkalies. The latter tend to foul the solutions, thereby decreasing the solvent power of the solution for values sought. There are other reasons, though obscure to the writer, which are illustrated by the following experiment: Make a solution of c. p. cyanide to 0.25 per cent strength, divide into three parts. To one add a little caustic soda, to the second a corresponding amount of lime and use the third without the addition of any alkali. Now treat the same weight of gold separately by the three solutions—other conditions being the same. Finally remove the gold, wash same and evaporate the total solutions and assay, when it will be found that the solution having the alkalinity due to lime has dissolved a much greater amount than either of the remaining two. These results have been verified by the writer on a large scale.

TREATMENT OF COARSE SANDS.

The general treatment of coarse sands in this country remains practically the same as it was when the process was introduced and as it was established in South Africa. The treatment of telluride and oxidized ores consists in dry crushing, followed by roasting when necessary, and then the usual treatment, which is leaching the ore without agitation. The dust resulting from dry crushing has offered some difficulties. It was formally added to the ore before roasting with the hope of remixing and final treatment as a whole. This, however, has not proven successful, and later practice has been to treat the dust by special methods which will be considered later. When wet crushing is followed there must in every case be used some method of sizing. This is accomplished either by the spitzkasten or overflowing a filling vat where the coarse sands are allowed to settle. When the vat is loaded the surplus water is drained off and the moisture left in the ore displaced by one of the mill solutions, the treatment being continued usually by the strong solution, then weak and this is followed by wash water.

The solution schedule of different workers varies somewhat, but the general practice is to load into strong solution when possible, macerate for a short time, 6 to 24 hours, then leach with the solutions until the ore is finished. As diffusion plays an important part, when changing solutions, it is customary to leach the offgoing solution below the surface of the ore before application of the changed strength of solution.

The time of treatment varies from 4 to 15 days, depending upon the physical condition of the values and the character of the ore. From my experience with several cyanide plants, operating upon as many types of ore, ranging from complete oxidized surface ores to the treatment of tailings from stamp mills, working upon pyritic ore, including live telluride ores which were roasted—I must say, it appears to me that not sufficient time is given in general practice for the greatest returns. For example, take a cyanide plant which, we will say, is treating 100 tons per day, the time of treatment being 8 days. The cost of cyaniding *per se* in this plant should not exceed $0.55 to $0.60 per ton on the average ore. This cost is made up, for the greater part, from consumption of chemicals, labor, power, superintendence, assaying, taxes, insurance and interest on the original investment. Now suppose the time of treatment was to be extended "N" days, what would follow? We would have practically no greater consumption of chemicals, for this occurs in the first few hours; the labor item would not be necessarily increased; all fixed charges would be the same, but N tanks must be added and the investment increased by that amount which would raise the taxes, interest and insurance. In the above case let N equal 4, then the increased cost per ton should not exceed approximately $0.10, and the increased extraction would be in excess of this amount. I predict that if this matter was properly determined (which may readily be done by the ordinate method) the average extraction would be near 90 per cent.

SLIMES TREATMENT.

In reducing ore to the required fineness, an appreciable amount of fines are always made. In dry crushing some of these fines are lost as dust. There is, however, at present in well-governed mills little or no loss during the crushing of the ore, as efficient dust collectors are on the market, which not only collect the dust made by the rolls, screens, etc., but will deliver the same, continuously and in a manner suitable for the work. The initial efforts were to reincorporate this with the coarser sands, but such practice has not been all that could be desired, and at present this part of the ore receives special treatment.

The filter-press treatment is considered both burdensome and expensive, and has not found much favor in this country. A modification of the filter-press method, making use also of decantation, has been in use for the past year at the plant of the Dorcas Reduction Company, at Florence, Col., and is described by the management as follows. Four large 70-ton agitation tanks, 13 feet 6 inches by 20 feet high with conical bottoms, are used. Into one of these tanks from 10 to 12 tons of dust are charged, solution sufficient to make a thin fluid is added, and then the whole mass is agitated with compressed air from the lower part of the bottom until the extraction is complete. The air simply is then discontinued and the charge allowed to settle for some time. The liquor is then decanted off through a vacuum pan 20 feet by 5 feet 6 inches high, the same pan being subsequently used for washing and finally for draining the slimes. The washing is extended only to a point necessary to maintain the volume of solution. The cyanide losses are given as being about 2 pounds per ton of dry slimes treated and the extraction as 93.50 per cent. This treatment is made on the dust without roasting, the dust coming from the telluride ores of Cripple Creek district. The total time of treatment is stated to be five days over all. These operations are worthy of special notice for the reason that, a hitherto considered stable compound (telluride of gold) toward cyanide solutions is brought within the field of commercial possibilities. The method is said to be simple and effectual.

In 1903 a process was set forth by Mr. George Moore, then at the Mercur plant in Utah, for the treatment of slimes. The process is a departure from both the filter-press and decantation methods. He uses a light framework, some 20 feet long, 4 feet high, and of sufficient width to permit the passage of pipes inside the framework to evacuate the filtered solutions (which is done with vacuum pump). The frames are covered with canvas, and several are grouped together, securely suspended from a traveling crane under which are arranged three tanks: one to contain the slimes which are agitated by compressed air to aid extraction, also to prevent settling. After the extraction is complete the filters are lowered into the slimes and a vacuum is created within the filters to aid the filtering of the solution. The same is continued until the slime prevents further filtration, when, with vacuum maintained, the filters are raised and carried to the second tank in which a dilute solution is contained.

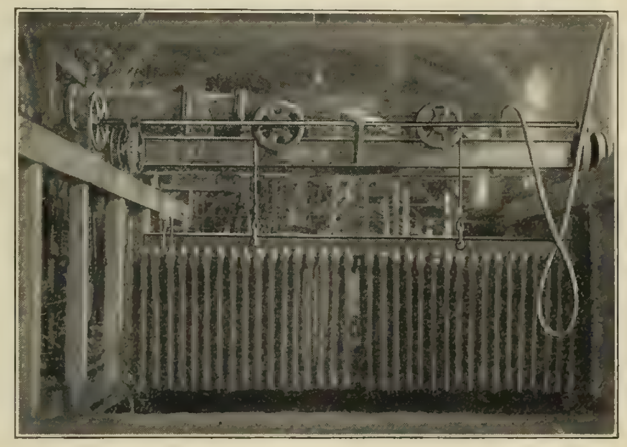
FIG. 1.—FILTER FRAMES, MOORE SLIMES PROCESS.

Here the filters are lowered and the strong solution displaced, the same followed with wash water. Finally the filters are carried to a fourth position, where the vacuum is released, and compressed air is then used to free the canvas from the slimes, the latter falling into cars for removal from the plant.

The process has been described by the inventor in the *Mining Reporter* of November 12, 1903. The first successful application of this process, so far as I am informed, was made at the Lundberg, Dorr & Wilson mill at Terry, S. D., after some modifications were made by Mr. Dorr. In a recent communication to the writer, Mr. Dorr gives a résumé of their experience with the process and such costs of operations as are available. He states: This process has now been in use a little over a year, during which time many changes and improvements have been made, and the difficulties incidental to the working out of a new process overcome. The plant has been described in Bulletin No. 7, of the South Dakota School of Mines, and in the transactions of the Black Hills Mining Men's Association. The changes made consist in an alteration in the hydraulic cranes, whereby the plates are carried from four points of support instead of one, the remodeling of the frames, and the arrangement for discharge by means of water instead of air, which gives perfect discharge, and in many other small details.

As there have been so many changes and periods when, owing to experimentation, the process was not worked to advantage, it is difficult to give correct operating costs, but some figures can be given.

Labor.—On the night shift the process is operated by the solution man, who, in addition attends to the regular duties of the mill, such as looking after the classification of the ores, packing the zinc boxes when necessary, drying and grinding all mine and mill samples, so that it is difficult to tell how much of his time to charge to the process. On the day shift, owing to some defects in our installation that would be difficult to remedy, a slime man is employed, who, with the solution man, handles the slimes from both shifts. With arrangements for sluicing out, and a good installation, the solution man could readily handle the process on both shifts, as the work will not exceed that in a dectantation plant of like capacity.

Repairs.—The repair cost on all pumps connected with the process amounted to $0.0146 per ton of slimes treated, for an extended period of time, and may be taken as an average for this installation. It may also be considered as a maximum, since our air pumps, which have caused the major part of the repairs, are a cheap type that is not satisfactory. The total costs would remain the same if the capacity of the plant was doubled.

Renewals.—The life of the frames should be almost indefinite, as they are made of wood and pipe and are subjected to no wear. It is impossible to say what the life of the canvas will be. The original bags lasted eight months under very hard usage. The cost for renewal here was 2 cents per ton of slimes treated. The ones in present use should materially exceed the life of the original ones.

Power.—We have not been able to make satisfactory measurements of the power required, but calculating from the work done and allowing liberally for losses in the pumps, it would not exceed 4 horse-power for both the agitation necessary, and for the Moore process. This at 40 cents per horse-power day would equal 5.3 cents per ton. This cost would not increase proportionally with an increased tonnage, as the vacuum pump used is much larger than required, and the cost of keeping the slimes in suspension in the loaded vat would be about the same with a larger vat, or with two sets of plates used in that vat.

Chemicals Used.—The consumption of cyanide at the mill is about 11.7 cents per ton of ore, which is slightly under the average consumption at dry crushing plants in the district operating on similar areas, so we can assume that the loss due to the Moore process is very slight. Tests have shown practically no consumption of cyanide from the agitation with air, and losses in the discharged tailings will not exceed 0.15 pound per ton.

Extraction.—The Moore process is only intended to recover the dissolved values with the slimes, but in this plant, owing to unexpected difficulty encountered in dissolving the last of the gold in the slimes, it has had to do a part of the dissolving as well. There has been a period when precipitation was bad, and changes were made in the plant that interfered with getting the best results. For a period of about eight months the average loss in dissolved gold has been 8 cents per ton in handling $8.00 slimes, and during this period the gold dissolved during the passage of the slimes through the Moore process has averaged $0.32. This fact and an inspection of the daily record which shows considerable losses due to causes that can be avoided, explains most of that loss. The following results from a 15-day consecutive run will show the possibilities of the process.

Valuation of the slimes entering the process (laboratory
[illegible]) .... $[illegible]
[illegible] of the discharged slimes $0.084
[illegible] of the discharged slimes (laboratory washed) $0.070
Tailings cyanided again in the laboratory (discharge) .. $0.956

Fig. 1 (p. 257) shows the general construction of the Moore process as applied at the Lundberg, Dorr & Wilson plant.

The decantation method has, perhaps, found more followers in the United States than other methods, and from the simplicity of the method it would seem that it was the better method where no talc is contained in the ore. But where talc is contained and where the ore makes a considerable amount of extremely fine slime, the decantation method becomes too slow. There have been a great number of schemes promulgated for the treatment of slimes, but none have become commercial factors and, therefore, will not be mentioned here.

Treatment of High Grade Ores

The cyanide process has been regarded as appropriate for the treatment of low-grade ores only, This conception has come from the early use of the process, the original application being made to tailings. This would intimate a limitation as to value of ore which could be economically treated. Simple logic, together with acquaintance with a few facts, should be all required to clear this question. Thus, if all the gold contained in a unit weight of ore was in one particle, then it would follow that the size of this particle would increase with increasing values of the ore, but this is not true—commercially, at least. We are, therefore, licensed by nature to assume that with increasing values an increasing number of particles ensue. This premise is enforced by fine crushing.

Now if it takes T time to dissolve one particle of gold in position P in the ore, other things being equal, it would require no greater time to dissolve another particle in position P_2. In fact, this would be true if the number were extended to P_n—where n is less than infinity. This is not intended to mean that coarse gold is not present in ores of higher values—this may or may not be true, since coarse values are peculiar more to types of ore rather than to grade. We see, therefore, that there is no *a priori* reason why ores of any value should not be successfully treated by the cyanide process where the ores are amenable to any chemical process.

Let us now look to general practice. The Dorcas Reduction Works is a custom mill, treating Cripple Creek ore, which is purchased upon full assay value and paid for. It is then crushed, roasted and cyanided. The average value of the ore treated during the past three years is given me by the management as being $35.00 per ton, and frequently for months as high as $60.00, the highest average for a single month being $86.02. These values are above the average treated by other chemical mills working on the same ores. The average extraction for the greater values is given as being from 95.00 to 96.50 per cent, while for the lower grades assaying approximately $25.00 per ton it is between 93.50 and 94.50 per cent, which means that the unavailable gold in the higher grade ore is not greater, practically, than in the lower grade, and confirms the above conclusions. This plant treats normally 125 tons per 24 hours, and has been in operation for some four years.

Precipitation

In the American practice of precipitation the zinc shavings method has found the greatest number of followers. Perhaps one reason is that this was the method introduced with the process, but a much more important reason is that efficient results attend the use of this method. In general, about 1 pound of shavings, costing $0.12, ready for the mill, will precipitate 1 ounce of gold, while in the electrolytic method of precipitation the cost of the energy necessary to precipitate this amount of gold would be theoretically much below this (assuming 100.00 per cent efficiency for electrolysis, about 65 watt-hours per ounce). Still, in American practice the electrolytic precipitation has not found favor, and it is difficult to obtain reliable data. It is, however, the general belief that the zinc method is the cheaper method, and this is in all probability true. The objection usually met with is the anode action and the excessive alkalinity required for complete precipitation.

It is doubtful that the electrolytic method will ever displace the zinc method in this country.

CLEAN-UP AND REFINING.

With the zinc method the clean-up consists in the removal of all the zinc from the boxes and a careful washing of the same with water. This is accomplished by placing the zinc on a screen—6 to 12 mesh—and scrubbing in water. The zinc remaining on the screen is returned to the zinc box for further use, the product passing being added to the mass for reduction. This product is sufficiently washed to remove all cyanide and alkali, and is then treated with dilute sulphuric acid to separate the values from the zinc. After this action is complete, the product is washed to remove excess of acid, then dried to a pasty mass with air, either on a vacuum pan or filter press, and finally to complete dryness by heat, after which it is mixed with suitable flux and melted to bullion, the finness of which is from 850 to 975, depending upon the care with which the operation is made. The bullion is then shipped direct to the mint, and settlement is made by the government upon a basis of $20.67 per ounce of fine gold. A great many special methods have been and are in use for both precipitation and refining. In precipitation the zinc dust method offers some attractions. The zinc consumption is perhaps less than with the shavings method. But the dust costs more than the shavings, and filter presses are necessary, also compressed air for agitation of the solution, which is run into tanks with conical bottoms. This solution is segregated and zinc dust is added, and then the whole mass is agitated with air, afterwards decanted and run through a filter press and returned to the general stock. The method seems more applicable to large plants as more attendance is required. In small plants the greater cost would overbalance the gain in zinc savings. The product may be refined in the usual manner by the use of acid, or the excess of zinc may be removed by lead acetate, sufficient lead added and the mass cupelled, which yields a high-grade bullion, but calls for a special furnace.

A special method of refining, which, when properly conducted, will give an exceedingly high-grade bullion—from 990 to 997 fine—was patented in 1894 by Messrs. Alonzo Coan and Hugh F. Watts, the procedure for which Mr. Watts describes as follows: Treat the zinc first with dilute sulphuric acid in the same manner as the general treatment, wash and dry to about 10 per cent moisture, then digest with concentrated sulphuric acid, heating the same for from thirty minutes to one hour in cast-iron vessel, the acid to be used only in slight excess. After the action is over the mass is cooled and diluted, then allowed to settle. The values subside very quickly to a perfectly clear solution, after which decantation may be used in washing, and in this way a thorough washing perfected, which is followed by drying and melting in the usual manner. It will be seen from the foregoing that the silver will be attacked, and if proper conditions were present the gold and silver would be parted. The silver, however, may be recovered from the acid liquor as a chloride, melted separately or added to the gold as desired. This method is extremely simple and gives excellent results.

REGENERATION.

The regeneration of the alkaline cyanides from the double salts has received some attention in this country, and was for some time considered to be a momentous question, since, when solved, it would make possible the treatment of all ores by the process. There are at present two methods, either of which will recover, as alkaline cyanide, the cyanogen in the double salts of working solutions. While it has been found in practice that a material saving is made in the cyanide consumption it has also been realized that the value of regeneration was overestimated, and that by regeneration the treatment of all ores has not been made possible. A brief review of these methods will be given here, together with such results as are available.

The Orr Methods.—In 1901 William Orr, of the Gold & Silver Extraction Co. of America, Ltd., patented a method which has been applied to several mills, it is claimed, with marked success. This method is largely a chemical one, and is described in United States patents Nos. 687,258, 689,017 and 689,018. In general, the process consists in a segregation of a fractional part of the solution, treating the same with caustic soda to decompose the double salts, and then with sodium sulphide to remove the base metals, which is presumed to be largely zinc. If copper is present in the solution the same is removed by electrolysis before the solution is passed to the zinc boxes for precipitation of the gold values, zinc cathodes being used, whereby the cyanogen is transformed into a zinc cyanide with precipitation of the copper. The solution is then run through the zinc boxes, where the gold is removed, and the solution is then treated as above. The cyanide is recovered from the waste solutions by adding an excess of zinc chloride and precipitating the cyanide as zinc cyanide. The same is removed either by filtration or decantation and returned to the working solutions as desired; recovery being made in the above manner. The technical modus operandi has not been made public by Mr. Orr, and therefore detailed results are meager.

Davis Method.—This method of recovery of the cyanogen depends upon the use and action of an alternating electric current. The solution is treated as a whole, without segregation, which is accomplished by placing suitable electrodes in the solution storage vat, the same to contain approximately the volume, in tons, which is used in the plant per 24 hours. By this the time of threatment of the solution is 24 hours. Slacked lime is added to the solution, each time a pumping from the sump occurs, and in the same manner as the cyanide is added, *e. g.*, by placing in some receptacle and allowing the pumped solution to flow over the alkali whereby the alkali is dissolved and added to the storage solution.

FIG. 2.—CYANIDE PLANT SMUGGLER-UNION MINING CO.

The electrodes are preferably lead, as slight action occurs, although a set of lead electrodes will last but little more than half a year. They then become honeycombed and lose the metallic property of lead to some extent. The distance of the electrodes apart, is, with wooden tanks, the diameter of the tank, the object being to get equal distribution of the current throughout the whole solution mass. Therefore the electrodes should be as long as the tank is high, and from 6 inches to 1 foot wide.

A suitable transformer giving a potential sufficient to produce a current from .05 to .10 amperes per ton of solution contained in the storage tank should be used. The frequency should be 60 periods per second, and there should be no ebullition of gas at either electrode. To deliver the energy to the solution, care should be taken that no pipe or other metallic conductor prevent this. The current should be maintained continuously and free alkali present, no mixture of alkalies used.

This process was applied, as described, to the cyanide plant of the Smuggler-Union Mining Co., (Fig. 2) about May 15, 1902, some four and one-half months after that plant was put into commission. This plant used, when in full operation, a total of some 2000 tons of solutions. Therefore some time elapsed before the solution had made its circuit and become treated. When this was done a marked difference in the KCy consumption was realized. Previous to this some 400 to 500 pounds of cyanide was necessary each day to maintain the standard strength of the solution. After the process was in full operation for two or three days, and the maximum effect reached, no cyanide was needed, then a few pounds, and finally as the double salts were regenerated and used the addition of fresh cyanide came up to a point some 25 per cent below the consumption before the process was installed. The solution in the mean time titrated as a c. p. solution with a pure opalescent end point (the Liebeg method being the one in use) and had all other indications of being purified. The results of the first four months were tabulated as being typical of the work of the plant without the process, and, neglecting May, the four following as that representing the improvement due to the process, the following are the results:

Total tons ore treated first four months was.... 38,190.00
Total pounds KCy used first four months was.. 42,600.00
Total KCy per ton of ore...................... 1.11 lb.
Average extraction first four months was Au 58.46 per cent, Ag 33.16 per cent.
Total tons ore treated last four months was.... 49,098.00
Total pounds KCy used last four months was.. 38,600.00
KCy per ton of ore was......................... 0.78 lb.
Average extraction last four months was Au 75.55 per cent, Ag 37.97 per cent.

Reduced consumption of KCy was 29.73 per cent.

Increased extraction was Au 17.09 per cent, Ag 4.81 per cent; total, 21.90 per cent.

After the process had been operated some six months, the solution was removed from the storage tank, and what was estimated to be 75 tons of precipitate was removed from the tank which contained calcium (as carbonate), also some oxide, arsenic, antimony, zinc, iron, aluminum and a trace of copper. No change in the character of the ore could be observed during the time of this comparison.

The writer is aware that these results are not akin to the present theory held by electrochemists, but after due consideration and in view of the above facts there can be but one conclusions, namely, that the theories are, at least, incomplete. Whether the action of the alternating current is a direct one on the compounds, or but a secondary, I have no means of determining.

Boulder, Col.

On Autogenous Lead Soldering.

By M. U. Schoop.

There exist at present quite a number of methods by which autogenous lead soldering (*i. e.*, welding without any flux or any foreign metal) may be performed. Of these methods, the following are of special value for storage battery factories, and are used to a greater or smaller extent:

(1) *Air-hydrogen flame;* the hydrogen is either produced from zinc and sulphuric acid, or is taken from steel bottles in which the hydrogen is compressed up to 125 atmospheres.

(2) *Oxygen-hydrogen flame;* both gases are obtained by electrolytic decomposition of water, and are used in the factory at a pressure of 500 to 600 mm. water column, while for installing batteries outside the factory, the gases are compressed in steel bottles.

(3) *Oxygen-illuminating gas flame;* according to the circumstances, this method may often be used in factories as a substitute for method (2). The heat of the flame is, however, considerably lower than in (2).

(4) *Electric resistance heating;* this method has been developed and is in use especially in French storage battery factories.

Finally soldering with the *"Pollak solder"* may be mentioned. This method makes very quick work possible, and consists in treating the heated edges with an alloy of hard lead, tin and mercury.

The use of the oxygen-hydrogen flame has such distinct advantages over the air-hydrogen method that especially in German factories the latter method has been completely replaced by the former. The two largest German works (Kölner Accumulatoren Werke and Accumulatoren Fabrik A. G., Berlin) have their own electrolytic plants for the production of oxygen and hydrogen. On the other hand, English factories use, as far as I know, exclusively the old "wet method" (zinc-sulphuric acid and air); this is somewhat astonishing, since compressed oxygen and hydrogen are probably obtainable in commerce in England.

The great disadvantage of the air-hydrogen flame, in comparison with the oxy-hydrogen flame, is mainly due to the fact that on account of the content of nitrogen in air, the economy of the burning process is very poor. In other words, much less work can be done in the same time than with the oxy-hydrogen flame with its absence of nitrogen. For large types of plates the loss of time may amount to 300 per cent; moreover, there is a great difference in the quality of the work.

Another disadvantage of the "wet method" is its dangerousness. Whoever has been working for years in storage battery factories knows that the handling of the reservoirs for developing the gas and of the air containers is by no means without danger, and I know myself several cases in which the engineer who made the installation, or his helper who has generally to attend to the apparatus, were killed; on the other hand, I do not know of a single accident with steel containers, although they have been used in the two German factories mentioned above for the last four and six years, respectively. It may sound paradoxical, but the statement is true that the use of compressed oxygen and hydrogen is absolutely dangerless, while the use of hydrogen and air is not free from danger By the way, I may call attention to the fact that sulphuric acid, sold as technically pure, and often also raw zinc, contains arsenic, and that for this reason objections may be raised against this method, especially if the installation is made in a poorly ventilated cellar or vault, and the development of troublesome acid vapors cannot be avoided.

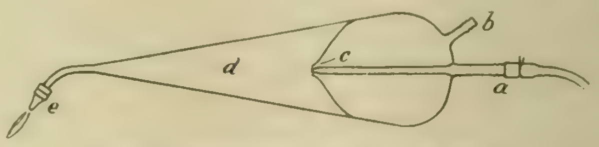

FIG. 1.—BURNER FOR LARGE SCALE WORK.

I have already described in various articles the oxy-hydrogen method, as well as the commercial electrolysis of water, so that I may refer the reader to these articles.[1] To many readers, however, the following construction of a burner may be novel, the principle of which may be applied with advantage to the use of oxy-hydrogen flame, as well as to hydrogen-air flame, and which should be specially useful in the latter case. The principle of burners of this kind is that in the same the one gas sucks the other gas, so that it is impossible that one gas might pass into the supply pipe of the other gas.

Fig. 1 shows a burner for more extended work. The hydrogen is supplied at *a*, and as soon as its velocity is beyond a certain minimum value, there is a sucking action at the aperture *c*, so that an explosive mixture is formed in the chamber *d*. If one wants to use the illuminating gas-oxygen flame, the oxygen is supplied under a certain pressure at *a*, and the gas hose is fastened at *b*. In both cases it is important to provide the chamber *d* for mixing the gases, since otherwise the gases have not sufficient time for intimate mixture, so that a non-homogeneous mixture passes out at *e* and the temperature of the flame is lowered.

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[1] L'Eclairage Electrique, 1899, 16, XII.; Zeit. f. Elek., 1900, No. 37, and 1901, No. 18. Die Industrielle Elektrolyse des Wassers (Enke, Stuttgart, 1901.)

Another form of this burner, especially suitable for work on a small scale, is shown in Fig. 2. Here the air is sucked in through the two holes in close proximity to the terminal; one of these two holes may be partly or completely closed by means of the regulating ring *a*. This burner contains no chamber for mixing the gases, but this slight disadvantage is counterbalanced by the neatness of the instrument.

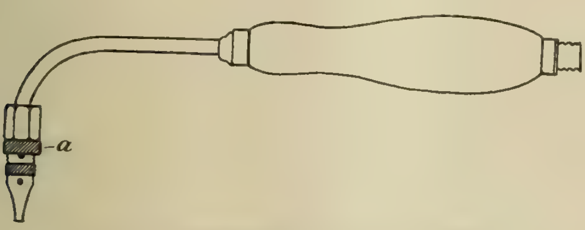


FIG. 2.—BURNER FOR SMALL SCALE WORK.

It is at once evident that this burner will render good service in all cases in which it is desired for some reason to use the air-hydrogen flame. In cases of small repairs outside of the factory it is often preferable to give the man only one small hydrogen bottle, instead of two steel bottles with oxygen and hydrogen. If the hydrogen is produced at the place of the repairs from zinc and sulphuric acid, there is no need for the otherwise necessary and dangerous air-gasometer, so that there is a saving in the cost of transport and attendance. If work is to be done in the factory itself, the use of this burner makes a compressed-air supply unnecesary.

If the electric resistance heating method is used, care must be taken that the applied voltage is not more than 14 to 18 volts, so that no arc can be formed; further, the carbon pencil is to be connected to the negative pole of the supply. If the latter point is neglected, the joint gets dirty, on account of carbon particles. The electric welding method can be completely learned by unskilled workingmen in one or two days, while the use of the air-hydrogen flame, or of the oxy-hydrogen flame requires considerable practical experence; this explains why good men for this work demand good wages.

Paris, France.

Metallurgical Calculations.—V.

By J. W. Richards, Ph.D.
Professor of Metallurgy in Lehigh University.

THE THERMOCHEMISTRY OF HIGH TEMPERATURES.

(*Continued.*)

In order to apply the principles explained in the preceding discussion, two sets of data are necessary; first, the thermochemical data, such as are ordinarily obtained by laboratory experiments at laboratory temperatures; second, physical data concerning the specific heats and latent heats of fusion, and volatilization of elements and compounds. The first have been given, at least for all important compounds met with in metallurgy, in a previous paper (April), the latter will now be discussed and the data presented as far as they have been determined.

Specific Heats of the Elements.

Dulong and Petit's law announces the fact that the specific heat of atomic weight of a solid metal is nearly constant, the value varying between 6 and 7, and averaging 6.4. This generalization was made chiefly upon the basis of the specific heats of the metals, as determined in the range 100° C. to 10° or 15° C., such as in Regnault's accurate experiments. About the only notable exceptions to this rule are carbon, boron and silicon, and it has been naively remarked by more modern physicists that these exceptions to the rule disappear if we find the specific heat of these three elements at high temperatures, that, for instance, the specific heat of carbon above 1000° C. is 0.5, making its atomic specific heat $0.5 \times 12 = 6$, and therefore the exceptions to the rule are all accounted for. Now, the rule is an important one, and has done good service, but the exceptions just noted and their behavior at high temperatures really prove that the rule must be made more general, or else abandoned altogether. The fact is, that the specific heats of almost all the solid elements increase with the tempearture at a rate equal to an increase of about 0.04 per cent of their value for each degree centigrade, so that the atomic specific heat of the majority of the elements, which is about 6.4 at ordinary temperatures, becomes about 40 per cent greater at 1000° C. for such elements as are not melted at that temperature. Therefore, while we may say that at ordinary temperatures the specific heat of atomic weight of a solid element is 6.4, and its specific heat per unit of weight is 6.4, divided by the atomic weight, yet it will be more accurate, if actual determinations have not been made, to assume that the actual specific heat increases 0.04 per cent for every degree rise in temperature, and mean specific heat to zero half that fast.

The specific heat in the liquid state has not been determined for many elements. It is in general higher than the specific heat of the solid at ordinary temperatures; in fact, it appears to be more nearly equal to the specific heat of the solid element just before fusion, and may be so assumed if no determinations have been made. It is found, furthermore, not to change perceptibly with rise of temperature, so that it may be assumed constant.

The specific heat of the gaseous elements has been determined only for those which are gaseous at low temperatures. For the metals which, as far as known, have monatomic vapors, *i. e.*, vapors in which the atoms exist alone and uncoupled with each other, the specific heat of atomic weight, occupying 22.22 cubic meters at standard conditions, should be theoretically 5.0 Calories at constant pressure, or 0.225 per cubic meter. We may thus estimate the specific heat of metallic vapors which have not been determined.

Latent Heats of Fusion of the Elements.

The passage from the solid to the liquid state is in all cases accompanied by an absorption of heat, which in amount varies from one or two Calories up to 100 Calories per unit of weight. This quantity has been most frequently determined by finding calorimetrically how much heat is given out by unit weight of the melted element just at its setting point, in cooling to ordinary temperatures, and subtracting from this the heat in unit weight of the solid substance at the melting point, as determined most accurately by exterpolating the value of mean specific heat of the solid up to the melting point. In this manner the latent heat of fusion for a number of elements has been directly determined.

If a crucible full of melted metal is allowed to cool, the temperature falls regularly until the melting point is reached, and then stays constant, or nearly so, for some time, while the metal is setting. A comparison of the rate of cooling before and after setting, with the length of time during which the temperature was constant, gives the relative value of the latent heat of fusion in terms of the specific heat of the melted metal and of the solid metal near to the melting point.

While the latent heat of fusion per unit weight shows no perceptible regularities, it is found that as soon as the latent heat of fusion is expressed per atomic weight of the element (analogous to specific heat of atomic weight) that notable regularities appear. The elements with high melting points have high atomic heats of fusion, and *vice versa*, so that if the elements are arranged in the order of their melting points their latent heats of fusion per atomic weight of each are in the same order, and very nearly proportionately so. If, for instance, a chart or diagram is made, using the melting points as abscissas, and latent heats of fusion of atomic weights as ordinates, the latter will lie very nearly in a straight line. Numerically, if the melting points be expressed in degrees of absolute temperature (centigrade temperatures plus 273), the latent heats of fusion of atomic weights average about 2.1 times the temperature of the melting point. This rule may be used to predict an undetermined latent heat of fusion.

In addition to the above general rule, another one bearing on the same question was also discovered and applied by the writer. (See *Journal of the Franklin Institute*, May, 1897.) According to this observation, the continued product of the latent heat of fusion of atomic weight and the coefficient of expansion and the cube root of the atomic volume (atomic weight divided by specific gravity) is a constant. If the coefficient of linear expansion between 0° and 100° C. is used, the constant is 0.095, or if the actual linear expansion of unit length from 0° to 100° C. is used, the constant is 9.5.. This rule, applied to all elements whose latent heat of fusion is known, gives satisfactory agreements, and enables us therefore to predict the latent heat of fusion of nearly a dozen other elements for which the coefficient of expansion is known.

LATENT HEATS OF VAPORIZATION OF THE ELEMENTS.

This datum has been determined for but a very few elements. Some of the metalloids, like sulphur, phosphorus and arsenic, are known to become complex vapors immediately above their boiling point, corresponding to such formulæ as S^6, P^4, As^4; the metals, as far as they have been tested, pass into monatomic vapors, such as Na, K, Hg, Zn and Cd, in which each atom represents a molecule. In the latter cases the following generalization may be made: The latent heat of vaporization of atomic weight is proportional to the absolute temperatures of the boiling point at atmospheric pressure, and is numerically equal to about twenty-three times that temperature (twenty-one times, if the outer work of overcoming the atmospheric pressure be not included). From this rule it is possible to estimate the amount of heat necessary to vaporize any metal whose boiling point under atmospheric pressure is known.

Examples: The boiling point of carbon under atmospheric pressure is 3,700° C., and if its vapor is monatomic, the latent heat of vaporization is, for an atomic weight of carbon ($C = 12$), $23 \times (3700 + 273) = 92{,}080$ Calories, equal to 7,673 Calories per kilogram of carbon. If the vapor is diatomic, and its formula C^2, then the above latent heat is for twenty-four parts of carbon, and for one part by weight is 3,808 Calories. Other considerations, from thermochemistry, make the latter value the more probable one.

The boiling point of cadmium is 772° C., and its vapor is known to be monatomic, what is its latent heat of vaporization? The atomic weight being 112, the latent heat of vaporization of this quantity is $23 \times (772 + 273) = 24{,}035$ Calories, which is 215 Calories per kilogram.

In making such calculations it must be strictly observed that the boiling point *under atmospheric pressure* is to be used, and not any temperature at which vapors may appear at partial tensions which may be only small fractions of atmospheric pressure.

THERMO-PHYSICS OF THE ELEMENTS.

Having laid down the laws and the empirical rules which appear to govern these phenomena, we will now discuss the data for the common elements, giving both what is known and what may be assumed as probably true wherever actual determinations have not been made. The elements will be taken up in the order of their atomic weights,—the only scientifically logical order.

In all cases, the actually measured *mean* specific heats, Sm, will be given. In the case of gases, these will be the mean specific heats under constant pressure; if they are desired under constant volume the amount of outer work must be calculated in Calories (two Calories per degree for a molecular weight of a gas, 0,09 Calories per degree for 1 cubic meter, and 0.09 ÷ weight of 1 cubic meter for a kilogram of gas), and subtracted from the specific heat at constant pressure. The specific heat of 1 cubic foot is, in pound Calories, the specific heat per cubic meter divided by 35.32 and multiplied by 2.204, or, in brief, multiplied by 0.0624; in British thermal units it will be nine-fifths as much, or the specific heat per cubic meter multiplied by 0.1123.

If, from the data given, it is desired to find the mean specific heat between any two temperatures, t and t′, instead of the mean specific heat from t to 0°, as given directly by the formula, it need only be observed that Sm from t′ to t is obtainable by finding Sm from 0° to (t′ + t). If, for instance, Sm (0 to t) = 0.303 + 0.000027 t, then Sm (t to t′) = 0.303 + 0.000027 (t′ + t). Furthermore, if the actual specific heat at any temperature t is desired, it is equal to the mean specific heat from 0° to 2t; *e. g.*, in the above case, S (at any temperature t) = 0.303 + 0.000027 (2t) = 0.303 + 0.000054 (t).

Temperatures will be always given and represented in centigrade degrees, except where the specific heat is given in British thermal units, in which case *t* represents, of course, Farenheit degrees, and will also be printed in italics, *t*, to further distinguish it from t in centigrade degrees. Absolute temperatures, if used, will be designated as T, and are equal to t + 273.

HYDROGEN.

From the experiments of Mallard and Le Chatelier we deduce:

Sm (0 — t) 1 kilogram (up to 2000° C.) =
3.700 + 0.0003t Calories.
1 pound (up to 2000° C.) =
3.700 + 0.0003t pound Calories.
1 pound (up to 3600° F.) =
6.660 + 0.0003*t* B. T. U.
1 cu. meter (up to 2000° C.) =
0.303 + 0.000027t Calories.
1 cu. foot (up to 2000° C.) =
0.0189 + 0.0000017t pound Calories.
1 cu. foot (up to 3600° F.) =
0.0341 + 0.000017*t* B. T. U.

For higher temperatures, such as electric furnace heats between 2000° and 4000° C. (3600° to 7200° F.); Berthelot and Vielle have made experiments which give us:

Sm (0 — t)	1 kilogram	= 2.75 + 0.0008t Calories.
	1 pound	= 2.75 + 0.0008t pound Cal.
	1 pound	= 4.95 + 0.0008*t* B. T. U.
	1 cubic meter	= 0.2575 + 0.000072t Calories.
	1 cubic foot	= 0.0161 + 0.0000045t pound Cal.
	1 cubic foot	= 0.0290 + 0.0000045*t* B. T. U.

LITHIUM.

Sm (26° to 100°) (solid)	= 0.9408 (Regnault).
Melting point	= 180° C.
Latent heat of fusion (1 kilo.)	= 73 (calculated).
S (liquid)	= 0.975 (calculated).
Boiling point	= 500° (estimated).
Latent heat of vaporization	= 2,540 Cal. (calculated).
S (gas) per kilo.	= 0.714 (calculated).
per cubic meter	= 0.225 (assumed).

BERYLLIUM.

Sm (solid) = 0.3756 + 0.00106t (Humpidge)

BORON.

Sm (solid) = 0.22 + 0.00035t (Weber).

CARBON.

Sm (amorphous) t under 250° = 0.1567 + 0.00036t.
(250° to 1000°) = 0.2142 + 0.000166t.
(for t over 1000°) = $0.5 - \frac{120}{t}$
Sm (graphite) = Approximately the same.
Boiling point = 3700° C.
Latent heat of vaporization = 23 T = 92,080 Cal. per molecule.
= 3,837 Cal. per kilo. if vapor is C^2.
= 6,906 B. T. U. per pound.
S (vapor) per cubic meter = 0.27 if diatomic (assumed).
per kilogram = 0.25 if diatomic.

NITROGEN.

Sm (o—t) 1 kilogram (up to 2000° C.) = 0.2405 + 0.0000214t Cal.
1 pound (up to 2000° C.) = 0.2405 + 0.0000214t pound Cal.
1 pound (up to 3600° F.) = 0.4329 + 0.0000214*t* B. T. U.
1 cu. meter (up to 2000° C.) = 0.303 + 0.000027t Cal.
1 cu. foot (up to 2000° C.) = 0.0189 + 0.0000017t pound Cal.
1 cu. foot (up to 3600° F.) = 0.0341 + 0.0000017*t* B. T. U.

For temperatures between 2000° and 4000° C. the following are the values of the mean specific heats to zero:

Sm (o—t) 1 kilogram = 0.2044 + 0.000057t Cal.
1 pound = 0.2044 + 0.000057t pound Cal.
1 pound = 0.3679 + 0.000057*t* B. T. U.
1 cubic meter = 0.2575 + 0.000072t Cal.
1 cubic foot = 0.0161 + 0.0000045t pound Cal.
1 cubic foot = 0.0290 + 0.0000045*t* B. T. U.

OXYGEN.

Sm (o—t) 1 kilogram (up to 2000° C.) = 0.2104 + 0.0000187t Cal.
1 pound (up to 2000° C.) = 0.2104 + 0.0000187t pound Cal.
1 pound (up to 3600° F.) = 0.3837 + 0.0000187*t* B. T. U.
1 cu. meter (up to 2000° C.) = 0.303 + 0.000027t Cal.
1 cu. foot (up to 2000° C.) = 0.0189 + 0.0000017t pound Cal.
1 cu. foot (up to 3600° F.) = 0.0341 + 0.0000017*t* B. T. U.
Sm (o—t) 1 cu. meter (2000° — 4000° C.) = 0.2575 + 0.000072t Cal.
1 cu. foot (2000° — 4000° C.) = 0.0161 + 0.0000045t pound Cal.
1 cu. foot (3600° — 7200° F.) = 0.0290 + 0.0000045*t* B. T. U.
1 kilogram (2000° — 4000° C.) = 0.1788 + 0.00005t Cal.
1 pound (2000° — 4000° C.) = 0.1788 + 0.00005t pound Cal!
1 pound (3600° — 7200° F.) = 0.3218 + 0.00005*t* B. T. U.

SODIUM.

Sm (solid) (—38 to + 10° C.) = 0.293 (Regnault).
Melting point = 96.5° C.
Latent heat of fusion = 31.7 Cal. (Johannis).
= 730 Cal. per atomic weight.
S. liquid = 0.30 (estimated).
Boiling point = 742° C.
Latent heat of vaporization = 23T = 23,345 Cal. per Na.
= 1,015 Cal. per kilogram.
S of vapor per kilogram = 0.2174 Cal. (calculated).
per pound = 0.2174 pound Cal.
per pound = 0.3914 B. T. U.
per cubic meter = 0.225 Cal.
per cubic foot = 0.014 pound Cal.
per cubic foot = 0.0253 B. T. U.

MAGNESIUM.

Sm (solid) (22° — 98°) = 0.25 (Regnault).
Melting point = 750° C. (approximate).
Latent heat of fusion = 58 Cal. (calculated).
Boiling point = 1100° (Ditte).
Latent heat of vaporization = 31,580 Cal. for Mg. = 24 (calculated).
= 1.315 Cal. per kilogram.
S (vapor) per cubic meter = 0.225 Cal. (assumed for Mg.).
per cubic foot = 0.014 pound Cal.
per cubic foot = 0.0253 B. T. U.
per kilogram = 0.2084 Cal.
per pound = 0.2084 pound Cal.
per pound = 0.3752 B. T. U.

ALUMINIUM.

Sm (o — t) solid = 0.2220 + 0.00005t (Richards).
Melting point = 625°.
Heat in solid at 625° = 158.3 Cal. (Richards).
Heat in liquid at 625° = 258.3 Cal. (Richards).
Latent heat of fusion = 100.0 Cal. (Richards).
S in liquid state = 0.308 (constant up to 800°) (Pionchon).
Boiling point (estimated) = 2300° C.
Latent heat of vaporization = 61.480 Cal. for Al. = 27 (calculated).
= 2.277 Cal. per kilogram.
S (vapor) per cubic meter = 0.225 Cal.
per kilogram = 0.1852 Cal.
per pound = 0.3334 B. T. U.

SILICON.

Sm (o — t) (solid, up to 234° C.) = 0.17 + 0.00009t (Weber).
Melting point = 1430° C. (Elihu Thompson).
Latent heat of fusion = 3,576 Cal. for Si. = 28 (Calculated).
= 127.7 Cal. per kilogram.
Boiling point = 2800° C. (estimated).
Latent heat of vaporization = 70,680 Cal. for Si. = 28 (Calculated).
= 2,524 Cal. per kilogram if vapor is Si.
= 1,262 Cal. if vapor is Si^2.
S (vapor) per cubic meter = 0.225 Cal. (assumed for Si.).
per cubic meter = 0.205 Cal. (if vapor is $Si.^2$).
per kilogram = 0.107 Cal. (if vapor is $Si.^2$).

PHOSPHORUS.

Sm (solid) (— 20 to + 7°) = 0.1788.
Melting point = 44°
Latent heat of fusion = 156 Cal. for P = 31.
= 5.03 Cal. per kilogram.
S (liquid) (44° to 51°) = 0.2045.
Boiling point = 287°.
Latent heat of vaporization = 23T = 12,880 Cal. per molecule = P^4.
= 104 Cal. per kilogram.
S (vapor) estimated = 0.36 Cal. per cubic meter.
= 0.064 Cal. per kilogram.
= 0.115 B. T. U. per pound.

SULPHUR

Sm (solid) (15° to 97°) = 0.1844 (Regnault).
Melting point = 114° C.
Latent heat of fusion = 9.37 Cal. (Person).
S (liquid) (119° to 147°) = 0.2346 (Person).
Boiling point = 444.5°.
Latent heat of vaporization = 72 Cal. per kilogram.
= 13,825 Cal. for S^6 = 192.
= 130 B. T. U. per pound.
S (of vapor) per cubic meter = 0.50 for S^6 up to 500°.
per cubic meter = 0.315 for S^2 above 800°.

CHLORINE.

Sm (constant pressure)
1 kilogram = 0.1241 (from 13° to 202°) (Regnault).
1 cubic meter = 0.40 Cal. (from 13° to 202°).
1 cubic foot = 0.025 pound Cal.
1 cubic foot = 0.045 B. T. U.

POTASSIUM.

Sm (solid) — 78° to + 23° = 0.1602 (Schüz).
Melting point = 58°.
Latent heat of fusion = 15.7 Cal. (Johannis).
= 612 Cal. for K = 39.
Sm (liquid) 98° to 58° = 0.25 (Johannis).
Boiling point = 730° (Carnelley and Williams)
Latent heat of vaporization = 23 T = 23,069 Cal. for K = 39.
= 592 Cal. per kilogram (calculated).
= 1065 B. T. U. per pound.
S (vapor) per cubic meter = 0.225 Cal. (assumed).
per cubic foot = 0.014 pound Cal.
per cubic foot = 0.025 B. T. U.
per kilogram = 0.128 Cal.
per pound = 0.230 B. T. U.

CALCIUM.

Sm (solid) 0° to 100° = 0.1704 (Bunsen).
Melting point = 780°.
Latent heat of fusion = 2.1 T = 2,106 Cal. for Ca. = 40
= 52.6 Cal. per kilogram.

TITANIUM.

Sm (solid) 0° to 400° = 0.0978 + 0.000147t (Nilson and Pettersson).

VANADIUM.

Sm (0° to 100°) = 6.4 for V = 51 (assumed).
= 0.125 Cal. per kilogram.

CHROMIUM.

Sm (15° — 99°) = 0.100 (Kopp) (uncertain).

MANGANESE.

Sm (14° to 97°) = 0.1217 (Regnault) (uncertain)

IRON.

Pinchon's determinations on the soft iron of "Berry" are the [illegible]

Sm (o to t) (for t up to [illegible]°) = 0.11012 + 0.000025t + 0.00000000547t²
Latent heat of change of state = 5.3 Cal. absorbed between 660° and 720°
Sm (o to t) (for t = 720° to 1000°) $= 0.218 - \frac{39}{t}$
Latent heat of change of state = 6.0 Cal. absorbed between 1000 and 1050
Sm (o to t) (for t = 1050° to 1160°) $= 0.19887 - \frac{23.44}{t}$
Melting point = 1600° (Roberts-Austen).
Latent heat of fusion = 70 Cals. (calculated from 2.1 T rule
= 69 Cal. (calculated from second rule).
Heat in solid iron at M. P. = 300 Cal. (calculated).
Heat in liquid iron at M. P. = 370 Cal. (calculated).
S (liquid) estimated = 0.22.

NICKEL.

Pionchon obtained the following results:

Sm (o to t) (for t up to 230°) = 0.10836 + 0.00002233t.
Latent heat of change of state = 4.64 Cal. absorbed between 230° and 400°.
Sm (o to t) (for t = 440° to 1050°) $= 0.099 + 0.00003375t + \frac{6.55}{t}$
Melting point = 1450°.
Latent heat of fusion = 62 Cal. by 2.1 T rule.
= 68 Cal. by second rule.

COBALT.

Sm (o to t) (for t up to 900°) = 0.10584 + 0.00002287t + 0.000000022t² (Pionchon)
Sm (o to t) (for t over 900°) $= 0.124 + 0.00004t - \frac{14.8}{t}$ "
Latent heat of fusion = 68 Cal. (calculated).

COPPER.

Sm (o to t) = 0.0939 + 0.00001778t (Frazier and Richards).
Melting point = 1085°.
Heat in melted copper at M.P. = 162 Cal. (Frazier and Richards)
Latent heat of fusion = 43.3 Cal. (observed) Frazier and Richards).
= 44.8 Cal. (calculated by 2.1 T rule).
= 46.2 Cal. (calculated by second rule).
S (liquid) estimated = 0.133 per kilogram.
= 0.24 B. T. U. per pound.

ZINC.

Sm (o to t) = 0.0906 + 0.000044t.
Melting point = 420°
Heat in solid metal at M. P. = 45.2 Cal.
Heat in liquid metal at M. P. = 67.8 Cal. (Person).
Latent heat of fusion = 22.6 Cal. (observed).
Latent heat of fusion = 22.4 Cal. (calculated by 2.1 T rule).
Latent heat of fusion = 24.6 Cal. (calculated by second T rule).
S (liquid) (calculated) = 0.1275 (from specific heat before fusion).
Boiling point = 930°.
Latent heat of vaporization = 23 T = 27,670 Cal. for Zn = 65.
= 425 Cal. per kilogram.
= 765 B. T. U. per pound.
S (vapor) per cubic meter = 0.225 Cal. (assumed).
per cubic foot = 0.014 pound Cal.
per cubic foot = 0.025 B. T. U.
per kilogram = 0.076 Cal.
per pound = 0.136 B. T. U.

GALLIUM.

Sm (12° to 23°) = 0.079 (Bettendorf).
Melting point = 30°.
Latent heat of fusion = 19.1 Cal. (Berthelot).
Sm (liquid) (30° to 119°) = 0.0802 (Berthelot).

ARSENIC.

Sm amorphous (21° to 65°) = 0.0758 (Bettendorf and Wulner)
crystallized (21° to 65°) = 0.083 (Bettendorf and Wulner)
Sublimation temperature = 450° to As^4.
Latent heat of sublimation = 25 T = 18,075 Cal. for As^4 = 300
= 60 Cal. per kilo. (calculated).

SELENIUM.

Sm (66° to 250°) = 0.084 (Bettendorf and Wulner)
Melting point = 125°.
Latent heat of fusion = 13 Cal. per kilogram (second rule).
Boiling point = 690° (Berthelot).
Latent heat of vaporization = 22,150 Cal. for Se^2 = 158 (calculated).
= 140 Cal. per kilogram.

BROMINE.

Sm (solid) (— 10° to — 77°) = 0.0843 (Regnault).
Melting point = — 7°.
Latent heat of fusion = 16.2 Cal. (Regnault).
Sm (liquid) (— 6° to + 58°) = 0.105 + 0.0011t.
Boiling point = 58°.
Latent heat of vaporization = 43.75 Cal. (Berthelot and Gier)
S (vapor) per kilogram = 0.0555 Cal.
per pound = 0.0999 B. T. U.
per cubic meter = 0.80 Cal.

STRONTIUM.

Sm (solid) (0° to 100°) = 6.4 Cal. per Sr = 87 (assumed)
= 0.0735 Cal. per kilogram.
Melting point = bright red heat.

ZIRCONIUM.

Sm (0° to 100°) = 0.0662 (Mixter and Dana).

COLUMBIUM.

Sm (0° to 100°) = 6.4 Cal. per Cb. = 94 (assumed)
= 0.068 Cal. per kilogram.
Melting point unknown.

MOLYBDENUM.

Sm (5° to 15°) = 0.0659 Cal. (De la Rive et Marcet).

PALLADIUM.

Sm (o to t) = 0.0582 + 0.00001t (Violle).
Melting point = 1500°.
Heat in solid metal at M. P. = 109.8 Cal. (Violle).
Heat in melted metal at M. P. = 146.1 Cal. (Violle).
Latent heat of fusion = 36.3 Cal. per kilo. (observed).
= 36.1 Cal. by calculation, rule 2.
= 65.3 B. T. U. per pound.

SILVER.

Sm (o to t) (up to 400°) = 0.0555 + 0.00000943t.
(over 400°) = 0.05758 + 0.0000044t + 0.000000006t^2 (Pionchon).
Melting point = 962°
Heat in solid metal at M. P. = 64.8 Cal. (Pionchon).
Heat in melted metal at M. P. = 89.15 Cal. (Pionchon).
Latent heat of fusion = 24.35 Cal. (Pionchon).
= 23.5 Cal. (calculated by rule 2)
S (liquid) (962° to 1020°) = 0.0748 Cal. (Pionchon).
= 0.0782 (calculated from solid state at M. P.).
Boiling point (approximtely) = 1600° (V. Meyer).
Latent heat of vaporization = 23 T = 43,080 Cal. for Ag = 108
= 397 Cal. per kilogram.
= 715 B. T. U. per pound.
S (vapor) per cubic meter = 0.225 Cal. (assuming it monatomic).
per cubic foot = 0.014 pound Cal.
per cubic foot = 0.025 B. T. U.
per kilogram = 0.046 Cal.
per pound = 0.082 B. T. U.

CADMIUM.

Sm (o to t) = 0.0546 + 0.000012t (Naccari).
Melting point = 321.7°
Heat in solid metal at M. P. = 18.81 Cal. (Naccari).
Heat in melted metal at M. P. = 31.83 Cal. (Person).
Latent heat of fusion = 13.02 Cal. per kilogram.
S (liquid) (calculated) = 0.0623 (from S of solid at M. P.)
Boiling point = 778° (Berthelot).
Heat in liquid metal at B. P. = 50.26 Cal. (calculated).
Latent heat of vaporization = 23 T = 24.173 Cal. for Cd = 112
= 216 Cal. per kilogram.
= 389 B. T. U. per pound.
S (vapor) per cubic meter = 0.225 Cal. (assumed by rule).
per cubic foot = 0.014 pound Cal.
per cubic foot = 0.025 B. T. U.
per kilogram = 0.0446 Cal.
per pound = 0.0802 B. T. U.

TIN.

Sm (o to t) = 0.0560 + 0.000044t (Bede and Regnault).
Heat in solid metal at M. P. = 14.34 Cal. (by above formula).
Heat in melted metal at M. P. = 28.16 Cal. (Richards).
Latent heat of fusion = 13.82 Cal. per kilogram.
= 13.7 Cal. (by calculation, rule 2)
Sm (o to t) (for t = 232° to 1000°) = 0.06129 — 0.000010474t + 0.0000000010345t^2 + $\frac{14.37}{t}$ (Pionchon)
S (liquid) (250° to 340°) = 0.0637 (Person).
Boiling point approximate = 1550° (Carnelly).
Heat in liquid metal at B. P. =
= 159 Cal. (Pionchon's formula)
Latent heat of vaporization = 23 T = 31,930 Cal. for Sn = 118
= 271 Cal. per kilogram.
= 487 B. T. U. per pound.
S (vapor) per cubic meter = 0.225 Cal. (assuming it monatomic).
per cubic foot = 0.014 pound Cal.
per cubic foot = 0.025 B. T. U.
per kilogram = 0.0424 Cal.
per pound = 0.0764 B. T. U.

ANTIMONY.

Sm (o to t) = 0.04864 + 0.0000084t (Naccari)
Melting point = 632°.
Heat in solid metal at M. P. = 34.1 Cal. (by above formula).
Heat in melted metal at M. P. = 74.3 Cal. (Richards).
Latent heat of fusion = 40.2 Cal. per kilo. (Richards).
= 15.8 Cal. (calculated by 2.1 T rule).
= 16.0 Cal. (calculated by second rule).

There is a very large discrepancy here, which may be due to the value 74.3, determined by the writer, being too high, or Naccari's results being too low. Further experiments by the writer have confirmed Naccari's results, and leave the latent heat of fusion of antimony anomolous. It should be, by two rules, about 16, and it appears to be 40. Possibly the water oxidizes it as it is poured melted into the calorimeter.

S (liquid) (632° to 830°) = 0.0605 (Richards).
Boiling point = 1600° (Biltz and V. Meyer).
Heat in liquid metal at B. P. = 131.4 Cal. (calculated).
Latent heat of vaporization = 23 T = 43,080 Cal. for Sb = 120
= 359 Cal. per kilogram.
= 646 B. T. U. per pound.
S (vapor) per cubic meter = 0.225 Cal. (assuming vapor monatomic).
per cubic foot = 0.014 pound Cal.
per cubic foot = 0.025 B. T. U.
per kilogram = 0.0416 Cal.
per pound = 0.0749 B. T. U.

TELLURIUM.

Sm (solid) (15° to 100°) = 0.0525 Cal. (Fabre).
Melting point = 455° (Richards).
Heat in solid metal at M. P. = 27.3 Cal. (Richards).
Heat in melted metal at M. P. = 46.3 Cal. (Richards).
Latent heat of fusion = 19.0 Cal. (Richards).
= 17.0 Cal. (calculat'n from rule 2)
= 34.2 B. T. U. per pound.

IODINE.

Sm (solid) (9° to 98°) = 0.05412 (Regnault).
Melting point = 114°.
Latent heat of fusion (1 kilo.) = 11.7 Cal. (Person).
Boiling point = 183°.
Latent heat of vaporization = 23.95 Cal. per kilogram (Fabre and Silberman).
S (vapor) 1 kilogram = 0.03489 (constant pressure for I^2).
1 cubic meter = 0.4088
S (vapor) (above 1200°) = 0.225 per cubic meter for I.
= 0.0394 per kilogram.
Latent heat of change $I^2 = 2I$ = 7,240 Cal. per formula, I^2.
= 28.5 Cal. per kilogram of vapor.

BARIUM.

Sm (0° — 100°) = 0.05 (Mendeleeff).

TANTALUM.

Sm (0° — 100°) = 6.4 per Ta = 183 (rule).
= 0.035 Cal. per kilogram.

TUNGSTEN.

Sm (6° to 15°) = 0.035 Cal. per kilo. (De la Rive and Marcet).

IRIDIUM.

Sm (0 to t) = 0.0317 + 0.000006t (Violle).
Melting point = 1950° (Voille).
Latent heat of fusion = 2.1 T = 4668 Cal. for Ir = 193.
= 24.2 Cal. per kilogram.
= 28.0 Cal. per kilogram by rule 2.
S (liquid) = 0.055 (by rule, from solid at M. P.).

PLATINUM.

Sm (0 to t) = 0.0317 + 0.000006t (Violle).
Melting point = 1775° (Violle).
Heat in solid metal at M. P. = 75.2 Cal. per kilogram (Violle)
Heat in melted metal at M. P. = 102.4 Cal. per kilogram.
Latent heat of fusion = 27.2 Cal. per kilogram.
= 26.3 Cal. (calculated by rule 2)
S (liquid) = 0.053 Cal. (by rule, from solid at M. P.)

GOLD.

S (from 0° to 600°) = 0.0316 — constant (Violle).
[illegible] 18.96 Cal.
[illegible] 18.96

M. P.) $= 0.0289 + 0.0000045t + \frac{\quad}{t}$ (Violle)
Heat in solid metal at M. P. = 34.63 Cal. (by above formula).
Heat in melted metal at M. P. = 50.93 Cal. (Roberts-Austen).
Latent heat of fusion = 16.3 Cal. per kilogram.
= 15.5 Cal. (calculated by rule 2)
= 14.4 Cal. (calculated by rule 1)
S (liquid) = 0.0358 (from S of solid at M. P.).

MERCURY.

S (solid) (at — 59°) = 0.0319 (Regnault).
Sm (liquid) (— 36° to 0°) = 0.0333 (Pettersson).
Sm (0 to t) (0 to 250°) $= 0.03337 - 0.00000275t + 0.0000000067t^2$ (Naccari).
Melting point — 40° C.
Latent heat of fusion = 2.84 Cal. per kilo. at — 40° (Person).
Boiling point = 350°.
Latent heat of vaporization = 77.5 Cal. per kilogram (Fabre and Silberman).
= 72.5 Cal. (calculation by Trouton's rule).
S (vapor) per cubic meter = 0.225 Cal. constant pressure.
per cubic foot = 0.014 pound Cal.
per cubic foot = 0.025 B. T. U.
per kilogram = 0.025 Cal.
per pound = 0.045 B. T. U.

THALLIUM.

Sm (17° to 100°) = 0.03355 (Regnault).
Melting point = 290°.
Latent heat of fusion = 2.1 T = 1182 Cal. for Tl = 204.
= 5.8 Cal. per kilogram.
= 5.8 Cal. per kilo. (by rule 2).
Boiling point = 1700° (approximately) (Biltz and V. Meyer).
Latent heat of vaporization = 45,380 Cal. by Trouton's rule, Tl = 204.
= 224.5 Cal. per kilogram.
S (vapor) per cubic meter = 0.225 Cal. (if monatomic).
per cubic foot = 0.014 pound Cal.
per cubic foot = 0.025 B. T. U.
per kilogram = 0.024 Cal.
per pound = 0.043 B. T. U.

LEAD.

Sm (0 to t) = 0.02925 + 0.000019t (Bede and Regnault).
Melting point = 326°
Heat in solid metal at M. P. = 11.6 Cal. (from above formula)
= 11.6 Cal. (observed, Le Verrier)
Heat in liquid metal at M. P. = 15.6 Cal. (Person).
Latent heat of fusion = 4.0 Cal.
S (liquid) (335° to 430°) = 0.0402 Cal. (Person).
= 0.0418 Cal. (calculated from solid metal at M. P.)

BISMUTH.

Sm (0 to t) = 0.0285 + 0.00002t (Regnault and Bede).
Melting point = 267°
Heat in solid metal at M. P. = 9.0 Cal. (from above formula)
Heat in melted metal at M. P. = 21.0 Cal. (Person).
Latent heat of fusion = 12.0 Cal. (Person).
Sm (liquid) (280° to 360°) = 0.0363 (Person).

THORIUM.

Sm (0 to 100°) = 0.0276 (Nilson).

URANIUM.

Sm (0 to 100°) = 0.028 (Blümcke).

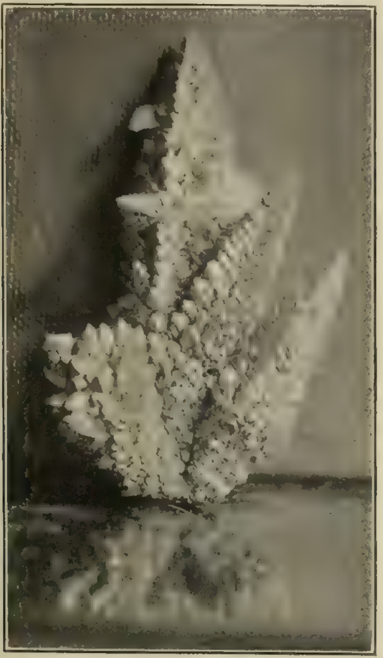
CORRESPONDENCE.

Electrolytic Copper.

To the Editor of Electrochemical and Metallurgical Industry:

Sir—I herewith send you a photograph of electrolytic copper which shows its crystalline structure very clearly.

The piece was broken from the lower corner of a full size cathode which had been operating under bad conditions in an insoluble anode tank. There was presumably a very high current density at this corner.



CRYSTALLINE STRUCTURE OF ELECTROLYTIC COPPER.

The photograph magnifies about one and a half diameter.

Lawrence Addicks.

Perth Amboy, N. J.

Notes.

American Electrochemical Society.—The next meeting of the American Electrochemical Society will be held on September 18, 19 and 20, at Bethlehem, Pa., in response to an invitation tendered by the president of Lehigh University. At the May meeting of the Board of Directors the following gentlemen were elected members: Hubert Buckley, Niagara Falls, N. Y.; H. F. Lewis, Boston, Mass.; Theo. W. Richards, Cambridge, Mass.; Edgar F. Smith, Philadelphia, Pa. At the June meeting the following gentlemen were elected members: Chas. Butters, Berkeley, Cal.; Clarence P. Hatter, Chicago, Ill.; Ralph D. Mershon, New York; Jas. K. Pumpelly, Indianapolis, Ind.; Harvey L. Williams, Bristol, Tenn.

Sulphate of Ammonia from Peat.—According to a recent statement in the *London Times,* the results obtained during the past two years at an experimental plant near London, in tests of the process of Eschweiler and Woltereck for utilizing peat for the production of sulphate of ammonia, have been so encouraging that $150,000 are now being spent on a complete recovery plant in the midst of the peat lands of Carnlough, Antrim County, Ireland. While the drawback to the use of peat as an industrial material has hitherto been the difficulty and expense of freeing it from its inherent moisture, it is claimed that the presence of up to 50 per cent of moisture is a positive advantage in the new process, and that there is a recovery of 68 per cent of ammonia in the form of the sulphate, which is a valuable fertilizing agent.

The Canadian Engineer.—It is announced that after June 1 *The Canadian Engineer* will be issued and conducted under new auspices. For twelve years this journal has been published by Biggar-Samuel, Ltd., with business offices at Montreal and editorial headquarters in Toronto. It has now been bought by the Monetary Times Printing Co., Ltd. The new editor is Mr. Samuel Groves, an English engineer, who was in 1904 lecturer on mines, furnace and foundry to the Carnegie Technical Schools in Pittsburg. Special attention is to be paid by the journal in future to metallurgical developments in Canada, "in order to meet the demands of the new era of prosperity in iron and steel, which one can hear coming on every passing breeze."

Indiana University.—We have received the 1905 catalogue of Indiana University, Bloomington, Ind. It contains in the first part a list of the board of visitors, trustees, faculty and officers, and historical notes on the university. Then follow the university regulations and an announcement of the courses for 1905-6. Among them are courses in elementary metallurgy and assaying, quantitative analysis, electrochemistry, lectures and laboratory work, special lectures and laboratory work on the deposition of metals, advanced electrochemistry and electrometallurgy, chemical engineering (Prof. Brown); physical chemistry, lectures, laboratory work and research work (Prof. Shinn), and a seminary in physical chemistry and electrochemistry. The register of students for 1904-5 shows a total of 1538 students (86 resident graduates and 1452 undergraduates).

Notes on Electrochemistry and Metallurgy in Great Britain.

(From Our Special Correspondent.)

The Meeting of the Iron and Steel Institute.

The members of the Iron and Steel Institute—who held their last session in New York in September, 1904—held their annual London meeting at the building of the Institution of Civil Engineers, on May 11 and 12 last. The lecture hall was packed, when, after such formal business as the reading of the reports of the council and the treasurer, Mr. Carnegie relinquished the presidential chair in favor of Mr. Hadfield. The retiring president, in his pleasing valedictory speech, compared his period of office to that of the position of a constitutional monarch. Here, some rather wondered that the author of "Triumphant Democracy" could compare himself to such a personage. There was not much time given for wondering, for the next sentence evoked applause by its reference to the force behind the throne—Mr. Bennett Brough, the secretary. The rest was a declamation of the true cosmopolitanism of science. He (Mr. Carnegie) had been the first president of the institution who was not a British subject. He had had an advantage over many Americans—he had been born in Scotland. In any case, with age, any sense of international distinctions passed into nothingness—he looked with equanimity on the prospect that succeeding presidents might be Americans (without the credential of Scotch parentage) or Germans, Frenchmen, Swedes or Belgians. His immediate successor was not absolutely free from foreign taint; even Mr. Hadfield was ruled from behind the scenes by foreign influences. Your correspondent failed for a moment to appreciate the ripple of laughter which followed, then he recollected that Mrs. Hadfield was born in the United States.

Mr. Hadfield then took the chair, his first official announcements being that of a further donation by his predecessor of £5000 to the Carnegie Research Fund, and the handing to Professor J. O. Arnold of the Bessemer Gold Medal. In making this presentation Mr. Hadfield recited Professor Arnold's services to the metallurgical knowledge of the world. The recipient's acknowledgement was tinged in its opening sentences by the spirit of badinage which had ruled a few moments before. This was well atoned for, that is, if atonement were needed, by Professor Arnold's handsome tribute to his collaborators at Sheffield. The meeting then settled down to enjoy Mr. Hadfield's selection of extracts from his presidential address. Mr. Hadfield is one of those rare men whose versatility is bewildering, being a combination of the successful manufacturer with the keen scientist, whose contributions to the meetings of the Institute are known throughout the scientific world. To these properties, the possession of which is everywhere acknowledged, Mr. Hadfield showed in his address high literary ability and a certain amount of statistical skill.

The address should be read carefully at one's leisure. Mr. Hadfield read extracts, from the 82 pages of which it is composed, for the space of 50 minutes, the reading suffering by reason of a desire to read as much as possible. Each section is in reality a monograph on a particular feature of a many-sided industry, whose sections of research and interest are perhaps too numerous to mention. Tributes to early metallurgists, many of whose names and pictures are rescued from obscurity. Perhaps—Mr. Hadfield does not say so—some of the opinions of these forgotten worthies are only worthy of note as curiosities. One of them is quoted as stating that steel "after being well cleaned, then heated, and quenched three or four times in equal parts of extract of radishes and water which contained earth-worms, cuts like lead." Another matter touched upon was the growth of the technical press, the contents of which were not ephemeral as was the case of daily papers written only for the moment. Metallurgy was now a distinct science, and a quotation was made of Borchers' asseveration that "the metallurgist has not only the task of supplying a product of a stated chemical composition, but he has also to supply the mechanical and constructive engineer with the same material in most diverse forms, with the most varied degree of hardness, strength and other physical properties. It is not by any means chemical analysis alone which will show him the way of fulfilling his task, which, as regards meeting such requirements, far surpasses that of supplying a material of a stated chemical composition." But a bare recital of some of the subjects touched upon by Mr. Hadfield in his address had better be given than isolated extracts. These include: alloys of iron with other elements, heat treatment, metallography, low temperature experiments, steel castings, war material, testing. The closing note foretelling an era of prosperity was gladly received. The prophet of a coming good time is always a more popular personality than one uttering jeremiads of doleful misgiving. Mr. Hadfield is, however, an authority of such weight that the prophesying of good was due to no desire to say nice things. The new president of the Iron and Steel Institute can well afford to despise either the popularity of one course, or the notoriety of the other.

At the close of the address the new president was warmly congratulated by Sir Edward Carbutt and Sir William White, who respectively proposed and seconded the vote of thanks.

Discussion of Papers at the Iron and Steel Institute.

The first papers read were those by Mr. S. Surzycki, of Czenstochowa, Poland, on "The Continuous Steel Process in Fixed Furnaces," and "Recent Developments of the Bertrand-Thiel Process in the Manufacture of Steel," by J. H. Darby, of Brymbo, and Mr. G. Hatton, of Round Oak. These were discussed simultaneously, a lengthy discussion being opened by Mr. Talbot, who criticized the authors of the second paper for giving details only from the Hoesch works in Germany, and then said that a basic lined tipping regenerative furnace which was used to refine molten pig-iron and take out more than 50 per cent of its impurities when working continuously with basic additions was not part of the Bertrand-Thiel process either, but was the continuous method of refining with which his name was connected. He (the speaker) considered it both from a practical and theoretical standpoint more advantageous to finish the steel in the one vessel rather than transfer it to another at practically the same temperature. As regards the composition of the slag, the figures given in two cases were only partial, and only 19 per cent of lime was shown. It was obviously necessary to know what the other bases were, especially the magnesia. In estimating the value of the slag the average weight per ton of steel obtained was also of importance. It was noticed that when the greater part of the refining was done in the continuous furnace, the bulk of the phosphorous was eliminated, but it still left .90 to be removed in the second furnace. As that was mixed with 25 per cent of scrap it would be interesting to know the percentage of phosphoric acid in the slag, and its weight per ton of steel.

Mr. Springorum, speaking as the manager of one of the first German works to adopt the process, replied at once to Mr. Talbot's criticisms. The slag contained about .8 per cent of phosphorus, .3 to .5 per cent of silicon, and about 1.5 per cent of manganese. The life of the furnace, without repair, was about 240 charges, and the phosphorus was from .8 per cent to .25 per cent, or .28 per cent. They started the process about a year ago, and he might say that they were now satisfied with it in every respect. It cost much less than the usual process, and the quality of the steel was as good; it was the best he knew. There was no difficulty in getting regularly .15 of phosphorus for soft steel, and .20 to .25 for a hard steel. The production was about 8 to 9 charges, and he was sure that they could have 10 charges without any difficulty.

Mr. F. W. Paul complimented Mr. Surzycki on the paper on the continuous process, but did not think that it was necessarily restricted to furnaces of 25 tons and over. Why could it not be applied to 10-ton furnaces? As regards Messrs. Darby and Hatton's paper, he asked for more information as to the composition of the slags and the necessity for using two furnaces. Comparing the sizes of the different Talbot furnaces in this country and America with those of Messrs. Darby and Hatton and Mr. Surzycki, as the latter produced 500 tons of ingots per week for 208 square feet of surface, the combined gas-heated mixture and two secondary furnaces should, on the same basis, produce 2016 instead of 840 tons per week.

Several other speakers followed, and the authors of the second paper replied to the discussion, stating that the German figures were quoted, as it was at the Hoesch works, that the best results were obtained. The transfer process was essential in the production of steel as low as 0.03 in sulphur, and 0.1 to 0.2 phosphorus, which could not be made without the secondary furnace. The high quality slag in the primary furnace was another advantage, saving the bulk of the phosphorus, the amount produced in the secondary furnace was negligible. The comparison of output with area of furnaces overlooked the circumstances that in America the metal contained 0.4 phosphorus, while at Round Oak and Brymbo pig-iron with 2 to 3 per cent phosphorus was used. The president then allowed Mr. Talbot to make a rejoinder to the effect that at Frodingham they worked with 2¼ per cent of phosphorus acid. There was no difficulty in working; it was merely a question of keeping the slag right, the process being continuous both in regard to metal and slag. The tilting furnace allows the slag to be taken off in any quantity, and whenever it is desired to do so.

The work of the second day began with Mr. Hadfield's temporary relinquishment of the chair to Mr. Windsor Richards, in order that the president might read an abstract of his paper on "Experiments Relating to the Effect on Mechanical and Other Properties of Iron and its Alloys Produced by Liquid Air Temperature." The discussion was opened by Mr. J. E.

Stead, who said that it was only natural to expect a metal to be more brittle at low temperatures, and that the chief interest in Mr. Hadfield's work lay in his discovery that alloys of iron with nickel manganese and carbon were quite ductile at such low temperature as that of liquid air. Professor Barrett laid emphasis upon the close alliance of certain different physical properties. In taking the electrical conductivity of the alloys, one could almost determine the percentage of carbon present. That was an exceedingly quick and easy thing to do, and if one had the electrical conductivity one practically had the thermal conductivity. Where there was a drop in the electrical conductivity there was a corresponding drop in the thermal conductivity. The 1414 B. iron manganese nickel alloy was one of the most remarkable that had ever been given to the world. Not only did it show the extraordinary change in properties when cooled to a low temperature, but its properties from an electrical and magnetic point of view were equally wonderful. It had the highest electrical resistance and the highest thermal resistance of any alloy that he had examined. Another remarkable feature was its extraordinary thermoelectrical property. Connecting the alloy with another piece of iron, and warming the ends, making a thermoelectrical junction of it, when its temperature was raised to about 400° C., or 500° C., the e.m.f. increased until suddenly it reached the maximum, and then, although one might go on raising the temperature from a black heat up to an intense white heat, there was absolutely no change in the e.m.f., which was constant through an enormous range of temperature. That suggested an exceedingly simple method of obtaining a constant e.m.f., which physicists very much wanted.

Mr. Gledhill, who spoke later on, brought the discussion from its physical and laudatory phases into a more commercial atmosphere, by pointing out that if anything could be done to carry out in a practical form the liquid air treatment of large masses of steel, great value would result. Of the remaining speeches little need be said.

Mr. Gayley's supplementary paper on "Dry Air Blast" evoked a spirited discussion. The general concensus of opinion was well expressed by Mr. Harbord, who said that, while a considerable amount of heat would be carried away in the gases when using moist air, the real saving was due to the regularity of work in the furnace. One remark, and that a closing one by Mr. Windsor Richards, foretelling a trial in the United Kingdom of the dry air blast, aroused much speculation as to when a paper on English experiences would be ready.

Only one speech was made on Mr. Axel Sahlin's paper on cleaning blast furnace gas, the speaker taking the attitude that so far as gas engines were concerned there was no need to demonstrate at this late period the necessity of cleansing gases.

Market Quotations.

Almost without exception the nominal prices of last month are still current. Copper sulphate has receded to £21 per ton, but the various soda compounds are unchanged. Shellac is again cheaper, being quoted at 145s. to 155s. per cwt. As for Para rubber, the demand is so great that fancy prices varying between 6s. and 7s. per pound are being paid, the wide range being due to the differences of quality coming from the various sources of supply. A rubber boom is now in progress in Ceylon, and extensive plantations are being laid out. The incentives to the chemist to produce a substitute synthetically are greater than ever. The incubus of past failures is in this case a heavy one.

Cleveland pig-iron continued to rise in price until 55s. per ton was reached on May 17. Speculation being the cause of the rise; an abrupt fall to 46s. 6d. took place on the 18th, and the closing price on May 31 was 45s. 6d. Copper has varied slightly from day to day, the tendency being generally downward, until the 24th, when a bottom price of £63 5s. was reached. Prices have since hardened to £65. Tin, which fetched £138 10s. on May 1, fell to £135 5s. by the middle of the month, closing at £136 5s. Lead has improved slightly, closing at £13 5s. per ton for ingot lead (English), and £14 5s. for sheet lead. Zinc is 10s. cheaper at £26 7s. 6d. per ton, while quicksilver is fetching from 145s. to 147s. 6d. per 75-pound flask.

London, June 3.

ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS.

By George P. Scholl, Ph. D.

Electric Furnaces and Furnace Products.

Method of Heating Carbon Articles.—F. J. Tone, Niagara Falls. Patent 786,859, April 11, 1905. Application filed April 2, 1904.

The method provides for the treatment of carbon articles, such as carbon brushes, battery plates and furnace electrodes by heating them in an electric furnace. The latter is provided with a resistance conductor, which, however, is not in contact with the materials to be heated, so that the current is practically confined to the conductor. It is claimed that this arrangement allows the carbon articles to be heated by convection and radiation, and results in a more uniform distribution of the heat. As shown in Fig. 1, the furnace consists of a resistance conductor 1, built up of plates or blocks of carbon. It rests upon a refractory base 3, and the articles to be heated are separated from it by a space 5, good results having been obtained with a space of ½ to ¾ in. between the core and the articles. The articles may be covered on the outside by a thin layer of powdered carbon 6 and a suitable refractory material. The sides and ends of the furnace are built up of brick walls 8. An alternative form of furnace, shown in the patent, contains several conductors, arranged parallel to each other, the groups of articles to be heated being interposed between them. This form of furnace is stated to be particularly useful for treating carbons of large size or considerable length, and may be operated in a continuous manner by connecting the cores successively with the source of current.

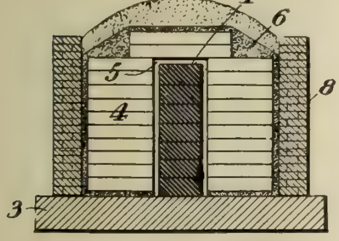


FIG. 1.—TONE FURNACE.

Arc Electrode.—D. A. Holmes, S. A. Tucker and E. van Wagenen, New York. Patent 789,609, May 9, 1905. Application filed Aug. 6, 1904.

The invention relates to an electrode to be used for arc lighting, which is formed of zirconium carbide, mixed with any desired percentage of a binding material, such as lampblack, coal tar, molasses, etc. The zirconium carbide is stated to be usually manufactured in an electric furnace, and is prepared from the mineral zircon mixed with an excess of carbon. As it is made at a very high temperature, the silicon from the zircon is distilled off, and the carbide is formed by the combination of the chromium and the carbon. The compound thus formed is stated to be of great stability, and various advantages are claimed for the electrodes, of which it forms the essential part.

Electric Furnace.—T. S. Prenner, Scranton, Pa. Patent 790,220, May 16, 1905. Application filed March 21, 1903.

The furnace consists essentially of a heating chamber, closed at the top, into which two electrodes enter through passages in the side walls, being arranged at angles of 120° from each other. A third electrode is arranged below these two, preferably in a vertical position. The electrodes are maintained at a proper distance from each other by electrical means, which are described in detail in the lengthy specification. In the same manner the action of the feeding mechanism is controlled, so that, if the arcs have not reached their normal condition, the feed of material will be slackened. If the arcs are broken, or there is no, or an excessive, current passing, the feed of the material will cease altogether.

Process of Reducing Metallic Compounds. E. F. Price, Niagara Falls. Patent 790,389, May 23, 1905. Application filed Aug. 31, 1904.

The essential point in this process consists in heating a charge of a refractory metallic compound, such as silica and alumina, mixed with a reducing agent, to the reduction temperature for a relatively short period, while the maximum temperature is kept below the volatilization point of the reduced metals, and the metal is at once withdrawn from the region of maximum temperature. The process may be carried out either in the incandescent or in the arc furnace, constructions for both types being shown in the specification. The incandescent furnace (Fig. 2) consists essentially of a vertical stack 1 of firebrick or masonry, provided with a refractory non-conducting lining 2 of magnesite or siloxicon and a water jacket 3. The electrodes 5 are introduced through the water-cooled top 4, and constitute one terminal of the current. The other terminal is formed by the carbon hearth 8, resting upon the metal plate 9, and connected to the course of current by connection 10. Tap hole 13 is intended for slag and tap hole 14 for metal. A gas outlet 15 is also provided. A charge 16, *f. i.*, silica and carbon, is fed into the furnace until it rises to and surrounds the lower ends of the depending electrodes 5. Current is then passed between these electrodes and the carbon hearth, and the charge, as soon as it is sufficiently heated, begins to act as a resistance conductor. As the cross-section of the furnace gradually decreases from the top of the furnace to the neck 11, the charge is gradually heated up higher until it reaches the necessary reduction temperature. The reduced metal then melts and at once drops out of the zone of maximum temperature into the cooler receptacle 12; it may be allowed to accumulate there in a layer 17, and withdrawn from time to time through the tap hole, while the layer of slag 18 accumulating on top of the metal is withdrawn as desired. Molten metal, or pieces of metal, either of that to be reduced or of an alloying metal, may be introduced into the furnace with the charge and allowed to percolate down through it for the purpose of collecting any scattered particles of reduced metal and carrying them into the body 17. The process and the several furnaces shown are stated to be suitable for reduction of refractory compounds or mixtures containing two or more metals, for example, manganiferous iron ores for the production of ferro-manganese.

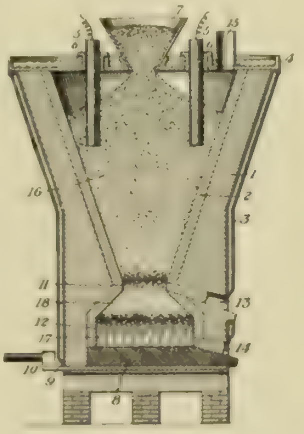


FIG. 2.—PRICE FURNACE.

Process of Reducing Metallic Compounds. E. F. Price, Niagara Falls. Patent 790,390, May 23, 1905. Application filed Aug. 31, 1904.

The process is based on the same principle as the preceding one, namely, to heat the charge for a relatively short period to its reduction temperature and allowing the metal to be immediately withdrawn from the region of the maximum. The furnaces which are stated to be used are, however, not incandescent, but arc furnaces; they are provided with a carbon lining, serving as one electrode, and have a central electrode, which depends vertically into the furnace to a point quite close to the bottom. The arc is sprung at this point, the rest of the furnace being filled with the charge. The reduced metal falls directly into a separate vessel, or an annular metal receptacle in the bottom of the furnace, whence it is tapped off.

Process of Smelting Metallic Compounds.—E. F. Price, Niagara Falls. Patent 790,391, May 23, 1905. Application filed Aug. 31, 1904.

The process relates to the production of ferrochromium from chromite, in order to produce metal low in carbon. The furnace shown in the specification consists of a body of firebrick masonry, with refractory lining of magnesite, siloxicon or carbon. Electrodes of opposite polarity depend into it. An arc is established between the electrodes or the electrodes and the carbon lining, when the latter is used, and a small amount of the charge, *f. i.*, a mixture of chromite, coke, lime and silica, is fed into the bottom of the furnace. It is then gradually filled, until in its normal working condition the electrodes are deeply embedded in the charge. This deep body of the charge, surrounding the electrodes, is claimed to effectively retain the heat within the furnace, both increasing the production of metal and maintaining the slag in a molten condition without special attention. It is stated that the use of this deep body is made possible by, and necessitates the use of, a minimum electromotive force, since the voltage usually employed in furnaces of this character would cause excessive waste of current by shunting or short circuiting through the charge if no current regulator were employed, which is a commercially impracticable method of procedure. Heat losses and oxidation of the hot exposed surfaces of the electrodes are also claimed to be avoided. The furnace is operated continuously, the metal and slag being drawn off and fresh materials added as required. The process is stated to be also applicable to the production of ferrosilicon from a mixture of silica and carbon.

Process of Smelting Iron Ores and Producing Ferrochromium.—E. F. Price, Niagara Falls. Patent 790,393, May 30, 1905. Application filed Oct. 19, 1904.

The reduction of chromite, magnetite and other ores of iron, as described in this specification, is carried out in an incandescent furnace, the charge and its products being employed as a resistance conductor. The furnace is of inverted cone-shaped form, decreasing in diameter from the top to the bottom. A water-cooled iron plate on the bottom contributes one electrode, while a water-cooled ring around the top serves as the other one. A body of refractory material, surrounded by a water jacket, operates the two. A body of the iron or alloy produced by the reaction in the charge is allowed to accumulate on the iron-plate bottom, and is liquid except at its lower portion, which is kept in a pasty or solid condition, owing to the action of the cooling water which circulates through the plate. Thus, during the process the bottom plate constitutes merely a terminal of the current, while the accumulated body of metal really serves as the lower electrode. By employing a charge containing a predetermined amount of carbon, it is claimed to be possible to produce an alloy with a minimum percentage of this element.

Process of Smelting Refractory Ores.—E. F. Price, Niagara Falls. Patent 790,394, May 23, 1905. Application filed Nov. 7, 1904.

The process consists in smelting an electrically conductive charge of a refractory ore and a reducing agent in a downwardly converging furnace between water-cooled metal electrodes. The latter distinguish it from the Cowles furnace, which is of a similar type, but in which the electrodes are of carbon. This construction was adopted on account of the expense and the difficulty of obtaining large blocks of carbon and of their rapid wearing, due to the attrition of the charge. One electrode is a water-cooled metal plate in the bottom of the furnace, while the other is constituted by a water-cooled metal ring around the top. It is stated that the process is applicable to the reduction of various refractory ores, such as those of chromium, titanium, vanadium, aluminium, calcium, boron, etc.

Process of Producing Low-Carbon Metals or Alloys.—E. F. Price, Niagara Falls. Patent 790,395, May 23, 1905. Application filed Nov. 22, 1904.

The distinguishing feature of this process is the fact that the production of low-carbon ferrochromium and other metals and alloys is effected by a non-continuous process in two steps. In the first stage an alloy with a relatively high percentage of carbon is produced by smelting a charge containing sufficient carbon to protect the carbon electrodes. The alloy thus produced is tapped from the melting furnace, and allowed to solidify. The ingot is then broken into fragments, which are mixed with a decarburizing agent, such as lime, and the mixture is smelted preferably in a furnace in which the charge serves as a resistance conductor, the carbon reacting on the lime to produce calcium carbide, which is separated from the purified alloy, the two collecting in the bottom of the furnace in superimposed layers.

Process of Producing Low-Carbon Metals or Alloys.—E. F. Price, Niagara Falls. Patent 790,396, May 23, 1905. Application filed Nov. 22, 1904.

The present process, claimed to be applicable to the production of low-carbon ferrochromium and other metals and alloys, differs from the preceding one by being conducted in a continuous operation in two stages. In the first stage an alloy with a relatively high percentage of carbon is produced, which process is carried out in the same manner and in a furnace of the same construction as in the preceding patent. Instead of cooling and breaking up the ingots, before treating it in the second furnace, the metal produced in the first one is run into a second furnace, or into another compartment in the same furnace, and is subjected to the action of a decarburizing agent. As such lime is used, a layer of molten lime being preferably maintained on the surface of the molten alloy, until the desired percentage of carbon has been withdrawn. The carbon reacts on the lime to produce calcium carbide, which separates as above from the alloy.

Process of Producing Low-Carbon Metals or Alloys.—E. F. Price, Niagara Falls. Patent 790,397, May 23, 1905. Application filed Dec. 10, 1904.

This process presents still another modification of the methods used in the two preceding ones. The high carbon alloy, *f. i.,* ferrochromium, if a mixture of chromite, coke, lime and silica is used, is produced by heating this charge in a furnace, the carbon lining of which constitutes one electrode, while a carbon rod depending vertically into the furnace constitutes the other one. The furnace is an arc furnace. The high carbon alloy thus produced is percolated through a vertical chamber, which is filled with a granular body of a decarburizing agent, *f. i.,* broken lime. The latter forms calcium carbide as above, withdrawing the required amount of carbon. An electric current is passed through this vertical chamber by means of electrodes suitably arranged at the top and the bottom, in order to maintain the granular body of lime at a high temperature.

Process of Producing Ferrochromium.—E. F. Price, Niagara Falls. Patent 790,392, May 23, 1905. Application filed Aug. 31, 1904.

The furnace employed for carrying out this process tapers again in cross-section from the top to the bottom. The electrodes which depend from the top are carbon rods, while a carbon plate constitutes the hearth of the furnace, and at the same time serves as the other electrode. The reduced iron and chromium form a molten alloy which drops below the charge sustained by the converging walls into the crucible, which is constituted by the lower end of the furnace. A layer of molten slag is preferably maintained on the molten alloy beneath the charge.

Electrolytic Production of Metals and Alloys.

Utilizing Spent Pickle Liquors.—A. S. Ramage, Detroit, Mich. Patent 788,064, April 25, 1905. Application filed Jan. 31, 1905.

The process is claimed to be applicable to the utilization of ferrous sulphate liquors from the pickling of iron or copper, etc. In carrying out the process the pickle liquor containing ferrous sulphate and free sulphuric acid, the latter in too small amounts for efficient pickling, is saturated with sulphurous acid. This operation is performed in a tower filled with coke, over which the liquor is distributed and flows down. A current of sulphurous gas is introduced at the bottom of the tower, and, ascending, meets the stream of liquor. The acid liquor is then allowed to run into covered electrolytic vats, preferably so constructed as to force the liquor to flow in a tortuous fashion past the anodes and cathodes. The vats are lined and in electrical connection with the anodes, while the cathodes extend transversely across the vat. The vats are arranged on an inclined supporting structure. The pickle thus regenerated is drawn into a collecting tank and used over again.

Apparatus for Agitating the Contents of Electrolytic Cells.—E. A. Ashcroft, Weston, via Runcorn, England. Patent 788,506, May 2, 1905. Application filed Nov. 16, 1903.

The invention relates especially to electrolytic cells in which a fused electrolyte is employed at a comparatively high-current density, and requires agitation or circulation for moving and circulating an intermediate electrode of liquid metal, such as fused lead, in the production of caustic soda or metallic sodium. The inventor states that if a circular vessel containing fused chloride of sodium be carrying a current which enters the bath more or less centrally, and leaves its more or less peripherally, as when a fused lead cathode and a carbon anode are employed in the production of a sodium lead alloy or sodium amalgam from chloride of sodium, a common pot-electromagnet, having concentric poles, when placed concentrically beneath the bath, will set up a violent swirl of the whole contents of the bath. The necessary condition is, that the lines of magnetic force shall cut the lines of current—flow more or less at right angles to the direction of the required motion. The direction of rotation of the fluid armature will be according to the relative directions of the magnetism and the current flow. The apparatus, as shown in cross-section in

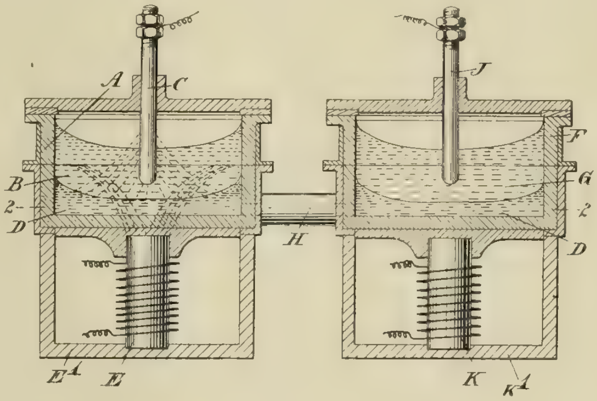


FIG. 3.—ASHCROFT'S APPARATUS.

Fig. 1 consists of an electrolytic vessel A, containing an electrolyte B, such as sodium chloride. It has a central anode C of carbon, and a cathode D, which may be a mass of fused lead at the bottom of the cell. An electromagnet E with an iron pot E′ is placed concentrically beneath the cell A. A second electrolytic vessel F, containing an electrolyte G of fused caustic soda, is connected with the first vessel by tangential passages H at the bottom of the vessels. The second cell is provided with a cathode J of nickel, and the anode consists of the intermediate electrode D, such as sodium-lead alloy, and the current entering the first cell at C is thus conducted through the alloy D to the second cell, where it leaves by the cathode J. An electromagnet K in an iron pot K′ is placed concentrically beneath the vessel, as with the first one. If the swirls in the two vessels A and F be caused to take place in the same direction, a flow of the intermediate electrode will take place along the channels H.

Electrolytic Apparatus.—W. McA. Johnson, Iola, Kan. Patent 789,740, May 16, 1905. Application filed Aug. 14, 1903.

The apparatus is stated to be more especially intended for the separation of copper from sulphate solutions obtained by leaching soluble surface ores or roasted sulphide ores. The apparatus comprises a tank, lined with a material unaffected by the electrolyte and furnished with stationary cathodes, which consist preferably of the metal to be deposited, in the above instance of copper. Anodes, consisting of cast discs of hard lead, are mounted upon a rotatable shaft supported upon the tank. Each anode consists of a disc of hard lead, which carries a series of lead vanes. The purpose of the latter consists in directing a stream of the electrolyte against the faces of the adjoining cathodes, during electrolysis, in order to facilitate the deposition of the copper in commercially available form from weak or highly acid solutions.

SYNOPSIS OF PERIODICAL LITERATURE.

A Summary of Articles Appearing in American and Foreign Periodicals.

Industrial Applications.

Electrolytic Refining of Lead.—Hans Senn, in a dissertation presented to the University of Basle, and published in the *Zeit. f. Electrochemie,* April 14, gives results of his experiments on Betts' electrolytic lead refining process; he tested in addition the electrolytic refining of cadmium with the fluosilicate solution. The author experimented on the addition of various amounts of gelatine to lead nitrate and acetate solutions, without getting the coherent dense lead deposit obtained with the fluosilicate solutions, and says that in consequence the fluosilicic acid has some specific property not possessed by the nitrate and acetate solutions, and that the solid deposit is not the result of the action of the gelatine alone. In electrolytically refining cadmium the author used a solution containing about 20 per cent SiF_6 and 1.23 per cent of cadmium, which increased by solution of hydrogen on the electrodes during the experiment to 2.59 per cent cadmium. He found 0.3 gram of gelatine per liter to be a sufficient amount for the production of a solid, smooth deposit. The deposits were about 1 mm thick, and of very good quality. He found 0.1 gram gelatine per liter with the lead solution to be enough to produce a solid deposit. The author electrolytically refined a series of alloys of lead with copper up to 17, with bismuth up to 26.67 per cent, and with antimony up to 10 per cent. If a current density of not more than 1 amp. per square decimeter was used, in no case did the cathode deposits of lead show a detectable quantity of the impurity present in the anode.

He found in his experiments a slow decomposition of the electrolyte with separation of silica. In one experiment he electrolyzed a solution of fluosilicic acid with platinum electrodes, and obtained a gelatinous precipitate of silica. As the experiments were carried out in glass, the silica may have come from the glass. In one series of experiments the proportion of free acid in the electrolyte diminished from 11 to 8.56 per cent, and in another increased slightly, while the lead percentage went down from 9 per cent to 4.83 per cent and 2.48 per cent respectively, indicating absorption of solution of neutral salt by the anode slime.

The author investigated the electrolytic refining of lead-platinum alloys, in the hope of finding a cheaper separation of the lead from lead-platinum alloys obtained in refining platinum. The experiments were only partly successful, for the anode slime contained lead and platinum in about the proportion required by the formula $PtPb_2$. By greatly increasing the voltage only a little more lead may be removed from this compound. The author found in his experiments that the loss of lead at the anode was over 100 per cent of that required by theory from the current passed, while the deposit on the cathodes averaged 98 per cent. With deeper electrodes and solution, so that the action of the atmospheric oxygen would have been less in proportion, the cathode-deposited lead would probably be much nearer the theoretical. A summary of the results is given in the table below.

Anode.	Current Density Amp per Sq. Dm.	Amperes.	Hours Run	Cathode Lead, Grams	Slime, Grams	In 25 Grams. Cathode Lead.	Composition Anode Slime.		
							Pb		SiO_2
Lead + .92 per cent. Cu . .	0.59	.5	29	55.9	1.64	no Cu	. . . .	23.41 Cu	. . . .
Lead + .92 per cent. Cu	1.07	.9	24	83.3	0.87	no Cu	. . . .	36.31 Cu	24.87
Lead + 1.006 per cent. Cu. . . .	1.55	1.3	18	90.2	3.11	no Cu	. . . .	19.47 Cu	. . . .
Lead + 1.006 per cent. Cu. . . .	2.30	2	9	69.4	1.10	no Cu	10.03	57.96 Cu	25.12
Lead + 12 per cent. Bi.	.59	.5	24	48.3	. . .	no Bi	. . . .	70.49 Bi	. . . .
Lead + 12 per cent. Bi	1.07	.9	7.25	25	12.72	no Bi	. . .	42.76 Bi	. . . .
Lead + 12 per cent. Bi	1.55	1.3	11	55.1	12.60	no Bi	. . . .	35.44 Bi	. . . .
Lead + 26.67 per cent. Bi.	1.07	.9	17	59.	13.25	no Bi	12.4	83.97 Bi	1.82
Lead + 26.67 per cent. Bi	1.55	1.3	16.5	82.6	36.38	.94 per cent. Bi	34.83	60.15 Bi	1.26
Lead + 10.03 per cent. Sb	0.59	.5	30	57.9	8.58	no Sb	(9 p.c.F.)	67.71 Sb	. . . .
Lead + 10.03 per cent. Sb	1.07	.9	8.5	29.5	5.19	no Sb	. . . .	47.52 Sb	1.04
Lead + 10.03 per cent. Sb . . .	1.55	1.3	22	110.3	18.52	.13 per cent. Sb	. . . .	53.47 Sb	. . . .
Lead + 9.81 per cent. Sb	1.55	1.3	18	90.3	10.43	.05 per cent. Sb	. . .	45.00 Sb	. . . .

Electrometallurgy of Iron and Steel.—*Stahl und Eisen* of

June 1 contains an address by Prof. W. Borchers on the present situation of the electrometallurgy of iron and steel. The author gave a review of the more important electric furnace processes which have been devised or introduced into practice. This review does not seem to contain anything new to our readers. However, the concluding words of the address are interesting. Prof. Borchers thinks that what has really been accomplished so far was the work of experimenters who were able electrical engineers, electrochemists and furnace designers, but were more or less unacquainted with the iron and steel industries, at least when they began their experiments. It is thus still too early to state definitely what may be accomplished by the electric furnace in the iron and steel industries. But the time is now ripe for experienced iron and steel men to take up the problem vigorously.

Magnetic Separation.—Iron Age, of June 22, contains an illustrated article on the Buchanan magnetic separator.

Theoretical and Experimental.

Electric Laboratory Furnace.—A new type of electric laboratory furnace was described by J. A. Harker, in a paper read before the Royal Society on April 13, an abstract appearing in the London *Electrician* of May 19. The same furnace was also exhibited before the (British) Physical Society; an abstract of the latter paper may be found in the London *Electrician* of June 2. The furnace is designed for the attainment, in absence of noxious gases, of temperatures between 800° C. and 2200° C. The conductor conveying the electric current is a tube of solid electrolytes similar in composition to the filament of a Nernst lamp. An essential feature is that for many purposes the usefulness and life of a furnace constructed in this way may be much increased by adopting a "cascade" system of heating; that is, the energy supplied may be divided, so that only sufficient is put through the tubular conductor to raise its temperature, say, 1000° C. above its surrounding, the surrounding itself being maintained at 1000° C., thus enabling a temperature of 2000° C. to be attained in the tube without straining it unduly. The regulation of temperature in small furnaces of this type is so perfectly under control that very well-defined melting points may be taken with very small quantities of substance. The thermoelectric method has been used in these furnaces for determining the melting point of platinum, the mean result of the experiments giving 1710° C. within 5°. The furnace is shown in Fig. 1. The arrangement consists of an inner tube AB, formed of Nernst filament earths and heated by a current at 200 to 500 volts, supplied by flexible platinum terminals. This tube is surrounded by a jacket of pure zirconia, and that again by a fire-clay tube, kept at 1000° by means of a nickel coil CD. This ensures a uniform gradient of temperature.

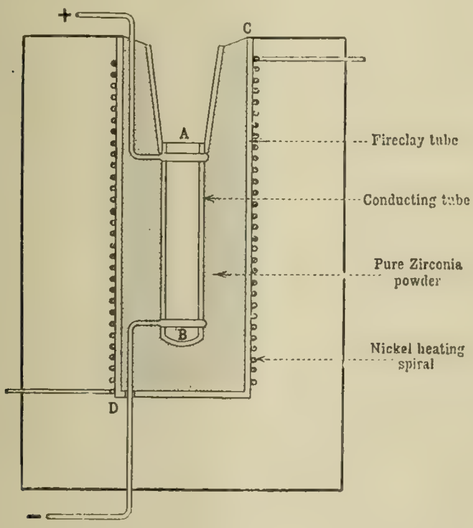


FIG. 1.—ELECTRIC FURNACE.

Chemistry of Electroplating.—In our Vol. II., page 389, we published in full W. D. Bancroft's paper read before the International Electrical Congress in St. Louis, and developing a tentative chemical theory of electroplating. This paper has now been enlarged and is published, together with some experimental work in support of his theory, in the April issue of the *Journal of Physical Chemistry*. The author has not changed his theory in any essential point. Some minor changes are as follows. One of his former general conclusions read: A fine-grained deposit is favored by high-current density and potential difference, by acidity and alkalinity, by low temperature and by presence of colloid. In his new statement he leaves out "by acidity and alkalinity." Another general conclusion of his former paper was that "solutions containing oxidizing agents appear to yield small crystals, while larger crystals are obtained from solutions containing reducing agents;" this statement is omitted under the general conclusions of the new paper. The following two general results are new: "Treeing is favored by anything which increases the fall of potential per unit length of electrolyte, or which increases the coarseness of crystallization; surface tension phenomena may also be a factor, though this has not been proved. The theory presented has been able to account for the phenomena, due to composition and concentration of the electrolyte to current density, temperature, solvent, colloids, other metals and cathodes. It has not accounted for the effect of acids and alkalies, nor for the effect of oxidizing and reducing agents; but this seems to be due to our ignorance of the chemistry of these solutions rather than to a defect in the theory."

Polarization of Metallic Anode.—In a paper published in the May issue of *Phil. Mag.*, S. R. Milner applies Nernst's theory of concentration cells to the calculation of the polarization of a metallic anode by a given constant or variable current. The anode chosen was in some of the experiments silver in a solution of nitric acid containing only a small amount of silver nitrate. The cathode was arranged in such a way that it was not polarized during the passage of the current. The author develops the theory for a certain construction of the cell on the basis of Nernst's and Planck's formulas. The results of his measurements agree with the theory even better than should be expected.

Electrolytic Chromium.—Chromium begins to become of considerable importance in the steel industries, especially in the manufacture of shells and tempered products, tool steels and the like, in general whenever properties of particular hardness are desired. It is introduced into the steel either in form of pure metallic chromium, as made by Goldschmidt's aluminothermic process, or in form of ferro-alloys, which are now made in considerable quantities, especially in the electric furnace. The question whether pure chromium could be produced commercially by electrolysis is still open. An advantage of such chromium would be that it is obtained in a very pure condition, but various authors have obtained such very different results that the whole subject is somewhat enveloped in mystery. H. R. Carveth and W. R. Mott publish in the March issue of the *Jour. of Phys. Chemistry* an interesting paper, in which they endeavor to determine the various factors which are of importance for the electrolytic production of chromium. In the present paper they only consider those cases where chromium acts as the base in trivalent compounds. The main results of their investigation are as follows: The presence of chromous salt appears to be essential to the successful deposition of the metal from its chloride or sulphate solutions. This has been proved by showing (*a*) that the current does work in reducing the chromous to the chromic salts; (*b*) that it is necessary to electrolyze the chromic salts for some time before the metal can be deposited with any considerable efficiency; and (*c*) that the efficiency is destroyed by oxidation of the chromous salt. The temperature has a very decided influence upon the efficiency of deposition; this result is in direct opposition to those recorded by Neumann. For each special set of conditions there seems to be a temperature at which the loss in efficiency caused by the decomposition of the chromous salt is balanced by increased efficiency caused by other factors. For example, it was found that after the sulphate had been electrolyzed for some time, rise of temperature increased the efficiency; the same result was observed when the current density was lowered. For a chloride solution between 0° and 16° the same effects were noticed. As might be predicted, the more concentrated the solution the higher the temperature which may be advantageously employed. The nature of the anode solution affects the yield very considerably; that this is probably due to the diffusion of the anolyte into the cathode chamber is indicated by the peculiar results obtained in the continued efficiency runs. On continued electrolysis of the solutions of chromic salts, the efficiency rises from zero to a value which is practically constant. This may be accounted for by assuming that the absolute and relative masses of the chromous and chromic salts have, under the conditions employed, become practically constant in the solution. Any factor tending to change these relations will affect the current efficiency—for example, change of temperature, increase of acidity, change of concentration and the presence of catalytic agents, such as toluene or petroleum ether. In the ordinary problems of electrochemistry it is conceivable that in reducing from the higher stages of oxidation, that solution from which metal can be deposited may not be obtained. For this reason solutions in which the metal shows its lowest valency should in this and in all analogous cases be used when it is desired to obtain good current efficiencies in the deposition of metal. (In connection with this paper the Electrochemical Society paper of H. R. Carveth and B. E. Curry should be consulted, which was published on page 176 of our May issue.)

Electrolysis of Fused Cuprous Sulphide.—Since all attempts which have so far been made to win copper directly from the ores by electrolysis of aqueous solutions have been failures, G. Bodlaender (who recently died in the prime of life) and K. S. Idaszewski have tried the electrolysis of fused Cu_2S. Their results are given in *Zeit. f. Elektrochemie*, March 24. While the results are completely negative from an industrial point of view, they are scientifically interesting. The melting point of Cu_2S was found to be 1091° C. Hittorf's result that solid Cu_2S is an electrolyte and a very poor conductor of electricity at ordinary temperature, was confirmed; the conditions are, however, completely changed, if in Cu_2S some CuS is dissolved, which acts practically as a metallic conductor. The authors, in trying to electrolyze fused Cu_2S could not find any decomposition point on account of the dissolved CuS. Fused Na_2S is decomposed at 1.62 volts, fused K_2S at 1.65 volts; fused Na_2S and Cu_2S are to a limited extent miscible, since up to a certain concentration Cu_2S is dissolved in Na_2S, and reversely Na_2S in Cu_2S. The authors found a new compound, NaCus. If a solution of fused Cu_2S in fused Na_2S is electrolyzed, copper migrates in form of a complex anion to the anode; this is the first example of a complex anion in a fused salt. At the cathode copper is deposited by a secondary reaction. There are no prospects whatever of winning copper on an industrial scale by electrolysis of pure fused Cu_2S or of fused Na_2S + Cu_2S.

Alternating-Current Electro-Analysis.—Brochet and Petit, who have carried out several investigations on this subject, have published another paper in *Comptes Rendus*, Feb. 13, which is abstracted in Lond. *Elec.* of March 3. The authors analyzed certain organic acids by means of an alternating current and obtained a high efficiency. They used a test tube containing about 150 cc. of solution and ribbons of platinum as electrodes, the mean current density being 1 amp. per sq. cm. The solutions used included solutions of the formates of potassium and barium, and acetic and oxalic acids. With the potassium formates the results are extremely variable, and the electrodes are rapidly covered with a light deposit of carbon. The efficiency rapidly diminishes, and it is necessary to make the measurements quickly, and to calcine the electrodes after each experiment. The carbon obtained does not adhere to the electrodes, but spreads through the liquid and gradually falls to the bottom. The gas developed is nearly pure hydrogen, with a very slight admixture of oxygen and carbonic acid. The efficiency is 80 to 90 per cent. The barium formate gives about 65 cc. per minute of a mixture of hydrogen and carbonic acid, the efficiency being about 75 per cent. There is practically no oxide of carbon disengaged. The analysis is in every way similar to direct-current analysis.

Iron and Steel.

An Iron-Ore Cable Carrier.—*Stahl und Eisen*, of March 1, contains an illustrated description of the most active cable road in the world. It is a cable 10.75 km long, with a fall of 145 meters, uniting the blast furnaces in Kneutinger, Lorraine, with the mines, and built by the firm J. Pohlig, in Cologne. The buckets hold on an average 650 kg of ore; they follow each other at a distance of about 65 meters upon the cable, and run at a velocity of 2.5 meters (8 feet) per second. The grade is 1.70, and the friction is so small that the cable practically runs itself. Working two ten-hour shifts a day, the roadway furnishes about 1700 tons of ore per day, which runs four blast furnaces. In 300 working days the road furnishes 500,000 metric tons of ore, representing a yearly tonnage of 5.75 million ton-kilometers (3.5 million ton-miles). The loaded buckets pass over ore bins, into which they dump their load without being detached from the cable. The railway cost for this transport was formerly 1.20 marks (30 cents) per ton, while the total cost by cable is but 0.25 mark (6 cents). Further comment on the enterprise of our German ironmasters is superfluous; the iron industry is moving with gigantic strides all over the world.

The Fine-Ore Problem.—The difficulty of smelting powdered ore is not the exclusive property of American ironmasters; but foreigners are meeting and overcoming the same difficulty, and their experience cannot but be suggestive and, perhaps, useful to us. Director Zeidler, of St. Petersburg, describes in *Stahl und Eisen*, of March 15, a briquetting process which has been operated at Kertsch, in the Crimea, for several months, at a phenomenally low cost. The ore is a powdery limonite, carrying 42.5 per cent of metallic iron, 2.00 manganese, 1.25 phosphorus, 1.50 lime, 0.25 magnesia, 4.50 alumina, 15.00 silica, 0.07 sulphur and 11.00 water chemically combined, while the freshly mixed ore contains besides this 15 to 18 per cent of moisture. Such ore must be dried before briquetting, and for this purpose drying ovens were constructed similar to the Scott mercury roasting furnaces, or more like the Cermak-Spirek roasters, in which the ore falls over inclined shelves, while the heating gases pass transversely across the furnace under the shelves. These are run by waste gases from a group of 200 Coppee coke ovens, situated at a distance of 150 meters. In order to bring the gases that distance a novel arrangement was devised, well worth describing. The waste gases left the ovens at 1100° to 1200° C., and were sucked into a Körting air-suction apparatus, using cold compressed air at 0.25 atmosphere pressure. It was not desired to have the products of combustion coming to the drying ovens at over 500° or 600° C., and the suction apparatus was so proportioned that 150 cubic meters of compressed air drew away 500 cubic meters of hot waste gases, forwarding them with a pressure equal to 100 mm. of water column (2.5 ounces per square inch). Whatever unburnt constituents were in the gas were completely burned in the injector, which was lined with fire-brick, and the gases coming from it were at a temperature of 550° to 650° The apparatus furnished a total of 274 cubic meters of mixed gases per minute, measured cold; the rest of the waste gases went, as before, to a boiler plant. The hot-gas flue, 150 meters long, was placed underground, with dimensions of 1.5 x 1 meter, and the fall of temperature to the drying ovens averaged only 50°. The gases left the ovens, with the 8 per cent of moisture extracted from the ore, at 60° to 100°, therefore giving a usefully applied heat of 78 per cent of their original heat. Each oven had a content of 10 cubic meters, and dried 80 to 120 tons of ore per day, down to 10 per cent of remaining moisture. The cost of this partial drying amounting to 0.46 mark (11 cents) per ton.

By drying the ore until it retained 10 per cent of moisture (aside from its chemically-combined water), it was found that it could be briquetted finely by simply powdering it over with fine lime (1.5 per cent of its weight) and putting through the press. Briquettes were made 100 mm. (4 inches) in diameter by 100 mm. high, using a pressure of 700 atmospheres tons, both above and below, the press furnishing 100 to 120 tons in twenty-four hours, using a 45-hp. motor. Experiments with ore of varying degrees of dryness showed that, using 500 atmospheres pressure, the strength of the briquette increased slowly, as the moisture in the ore decreased to 12 per cent; between 12 and 10 per cent of moisture the strength increased surprisingly fast, to a maximum, and then fell off just as quickly when less than 10 per cent of moisture was present. It is evident that with such ores, the moisture should be just sufficient to fill the pores between the compressed ore particles, and neither more nor less, to attain maximum strength by simple pressure. By acting on this principle, all burning was abandoned, the pressed briquettes being capable of standing a fall of 5 meters without breaking; they are carried direct from the press to the blast furnace. The total cost of sieving, drying and briquetting amounted in two months' running to 1.50 marks (37 cents) per ton.

Against this cost, the figures showing the running of the furnaces with raw ore and 80 per cent of briquettes in the charge are most striking. They are best shown by the deadly parallel column, in which I. is the original running, and II. the running with 80 per cent briquettes:

	I.	II.
Extraction of iron from ore.......	32.61%	39.68%
Output of pig iron daily...........	110.00 tons	185.00 tons
Tons of coke per ton of iron.......	1.48 tons	1.24 tons
Cost of charges per ton of iron.....	43.65 marks	40.03 tons
Manufacturing costs per ton of iron.	12.44 marks	6.46 tons
Total costs per ton of iron.........	56.08 marks	46.49 tons
Blast pressure	1.00 atmos.	0.50 atmos.

Besides these advantages there is not reckoned in, in marks and pfennigs, the sparing of brain fag to the management, caused by the avoidance of slips, explosions, flue dust and all irregularities to such a degree as to inspire Director Zeidler to say: "Der Betrieb was so ideal regelmässig, wie man ihn sich nicht besser vorstellen kann."

Mild Steel Castings.—The veteran iron metallurgist, Dr. Wedding, of Berlin, contributes to the March number of our esteemed contemporary, *The Iron and Steel Magazine,* a very clear presentation of "The defects in ingot-iron castings." Dr. Wedding's semi-official position in Berlin accounts for his using the term "ingot iron" to designate what not only we, but his own countrymen, call "mild" or "soft" steel, that is, cast steel which does not contain enough carbon to harden. The explanation is that the German government officially recognizes as "steel" only such varieties as can be hardened and tempered, and therefore it would be *lese majeste* towards the king of metals for a "Geheimer Bergrath" to use the term "soft steel." We think, however, that since *The Iron and Steel Magazine* is published in the English language, that "ingot iron" could have been more properly called "mild steel." Discussing the properties of such castings, their tensile strength is less than 50 kilograms per square millimeter (70,000 pounds per square inch), yet they possess greater strength than cast-iron castings, and are superior to the latter in being forgeable. Their chief defects are the enclosing of cavities, due to shrinkage or blow-holes from gases. Three kinds of furnaces furnish the melted material, viz.: open-hearth furnaces, converters and crucibles, the latter giving the best metal because its composition can be more exactly regulated, and it is capable of being made comparatively free from enclosed gases. To these Dr. Wedding will, in the next article he writes on this subject, be able to add the electric furnace, which combines the advantages of the crucible furnace with a cheapness which begins to rival that of the open-hearth furnace at certain localities. The defects in such castings are those due to (1) blow-holes caused by evolution of dissolved gases; (2) shrinkage cavities, caused by contraction of the melted interior after the outside has solidified; (3) air bubbles, caused by air mechanically carried into the melted metal during pouring; (4) steam cavities, produced by development of steam or other gas from the materials of the mould; (5) surface cracks, due to gas pressure arising from the steam or gas evolved by the moulding material; (6) cracks, caused either by contraction of the casting in the mould or by exposure to cold air when stripped too hot. The methods in use for avoiding these defects are discussed at length; it is to be regretted, however, that in describing the action of silicon and aluminium in preventing blow-holes, the distinguished author should still adhere to the discredited occlusion theory, viz.: that these elements in some mysterious way increase the capability of iron for holding gases in solution. Ledebur's de-oxidation theory, supplemented by Le Chatelier's beautiful explanation, has rendered the "occult" occlusion theory quite superfluous.

The article concludes with the methods of cure of such defects, as distinguished from the methods of prevention. They are: (1) Filling the cavities by electric welding; (2) filling by patching, *i. e.,* by welding in very hot pure iron; (3) filling by molten iron; (4) filling by "Thermit" treatment. The methods of cure are, in general, so unsatisfactory, that the metallurgist is strongly recommended to put his best efforts in the direction of prevention.

Case-Hardening.—Those particularly interested in this treatment of steel articles will find a good *résumé* of present information upon the subject in an article by David Flather in the April issue of *The Iron and Steel Magazine,* being a reprint from an English publication of 1903.

Iron Foundry Practice.—The same number of the journal just referred to contains a reprint of a very instructive paper by Mr. J. E. Stead, of Middlesborough, England, read before the Cleveland Institution of Engineers, Feb. 20 of this year. As the Transactions of that Institution are difficultly accessible, our readers interested in this subject will welcome the opportunity of reading this paper in full in our American contemporary. The following practical suggestions, however, are so entirely pertinent to every sort of metallurgical practice that we quote them here for the benefit of all of our readers: "(1) An analyst who has no metallurgical knowledge is only useful for supplying true analysis, but in that respect he is indispensable. (2) Foundries rarely have fully trained metallurgists to control the mixing and casting of metals. (3) If there is a metallurgist in the foundry he will be able to usefully apply the analyses provided by the analyst. (4) The primary essential is to have accurate analyses, and in making a selection a chemist should be chosen who has had a metallurgical training in a technical college. (5) It is a rare thing to find a chemist who has had practical experience in a foundry, and the founder must take the material available. (6) Having selected a man, his real training must begin on entering a foundry laboratory, and whether or not he develops into a useful metallurgist must depend largely upon his ability and capacity for gaining knowledge. (7) To facilitate this his employer should furnish every standard metallurgical treatise containing useful data on foundry practice, a complete set of metallurgical journals and technical papers, and the analyst should become a member of the Foundrymen's Association and attend its meetings. (8) One of his duties should be to read everything published on foundry work, and to ensure that he does this he should supply periodically to the head or principal a review of the publications accessible. * * * (12) An analytical metallurgist in a works will be found as valuable in the conduct of the establishment as the secretary, accountant or the timekeeper."

RECENT METALLURGICAL PATENTS.

Copper.

G. Mitchell (788,589, May 2) patents the following process of treating silicious copper ore: He feeds into the smelting furnace any kind of copper sulphide, pyrites, or silicious ores in particular, after having finely crushed them and without roasting. He then subjects the ore, without the admixture of any carbonaceous or other fuel not already contained in the ore, to a sufficient degree of heat in a reducing atmosphere within the smelting furnace to effect fusing, admitting only sufficient air to support combustion and avoiding the presence of sufficient oxygen to cause a chemical union of the metallic portion with silica. Thus the metals of the ore will be fused, leaving the silica behind and unattacked by the metals. The advantages aimed at by the inventor are to avoid roasting and to save fuel, on account of no silicates being formed in the smelting process. The fused matte, minus the silica, is drawn off from time to time, and charged into a Bessemer converter.

Vanadium, Molybdenum, Titanium, Tungsten.

H. L. Herrenschmidt (787,758, April 18) patents the following process for treating vanadium ores, which is stated to be equally applicable for treating molybdenum, titanium and tungsten ores. The process consists essentially of three steps. First, the refining of a vanadate-of-soda liquor, obtained by treating the vanadium ore, or product, with a soda salt, either by evaporation and crystallization or by the addition of vanadic acid (which latter may result from a fractional precipitation obtained by the addition of vanadium sulphate, which may be impure), or of an acid or of a metallic salt.

Second, the precipitation of the vanadium contained in this purified liquor, either by means of the sulphate, chloride, or other soluble salt of a metal—such as iron, nickel, copper, etc., with the object of obtaining the vanadate of iron, nickel, copper, etc., respectively; or by the action of concentrated sulphuric acid upon the liquor previously concentrated to a syrupy condition, with the object of obtaining vanadic acid.

Third, the reduction of the oxygenated compounds of vanadium; the vanadates of nickel, iron, copper, etc., being reduced by carbon, tar or other reducing agent, to produce the desired alloy.

Nickel.

The important industrial problem of extracting the nickel from copper nickel matte is the subject of a patent of Hans A Frasch (791,090, May 30). If copper-nickel matte, and particularly matte which has been concentrated by the Bessemer process, be subjected to the action of acids, like sulphuric acid or hydrochloric acid, the nickel sulphide in the matte is readily dissolved; but the nickel which is not combined with sulphur, dissolves very slowly if at all. To dissolve this insoluble, or sparingly-soluble nickel, it is proposed to finely pulverize it and mix it with sulphur and a solvent acid and apply heat.

Zinc.

The very low efficiency of our present zinc furnaces is well known. If the main features of our ordinary method of zinc reduction and distillation are to be maintained it is evident that, besides the important problem of retort construction, the [illegible] possibility of improving fuel economy would be by means of a very uniform temperature in the furnace, just sufficient to reduce the ore in each retort in the allotted time. At present the Siemens regenerative retort furnace is limited in height to five tiers of retorts, and it is impossible to heat uniformly the different tiers of retorts. The lowest tiers are subjected to an intense heat, with the result that the ore in them is reduced more quickly than in the upper tiers. For this reason it is preferred to charge the upper tiers with the less refractory oxides or blue powder.

Herman Hegeler and Julius W. Hegeler (792,773, June 20) propose the construction of a regenerative retort furnace, shown in Fig. 1, in order to control completely the temperature throughout the full height of the furnace, so that the number of tiers of retorts is limited only by the draft and the practical height of furnace construction. For this purpose the inventors introduce the entire amount of the air at the bottom of the retort chamber under the lower tier of retorts, and provide an excess of air up to the top of the combustion-chamber, the gases being introduced separately from the air at different heights above the air-ports, to produce the combustion progressively as required, in order to heat all the retorts simultaneously and uniformly to the temperature required and reduce the ore in all of them at the same rate.

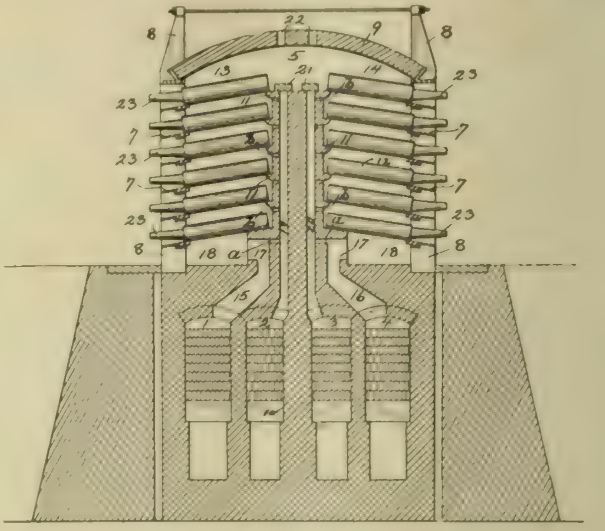


FIG. 1.—ZINC FURNACE.

The diagram is almost self-explanatory, when it is understood that 1 and 4 are air-regenerators, and 2 and 3 gas-regenerators. The flues 15 and 16, with their ports 17, are so constructed as to throw the entire volume of air into the lower part of the combustion chamber beneath the lowermost tier of retorts. Flues 19 and 20 are connected with the gas-regenerators 2 and 3, and extend straight up on both sides of the central division wall 10.

Iron and Steel.

N. McConnell (792,914, June 20) patents the following method of making open-hearth steel, the principal advantage claimed for the same being a reduction of the time of treatment. The inventor introduces molten pig metal into an acid-lined converter and blows air through it in the manner common in Bessemer practice, until the silicon is removed, the carbon considerably reduced (to about 1 per cent), and the temperature of the metal raised by the reaction to a point corresponding with that required for keeping it melted, notwithstanding its loss of carbon. The metal is now transferred from the converter into an intermediate regenerative furnace, in which a large body of such metal is held under the influence of heat. The final treatment of the metal is effected in basic lined open-hearth furnaces, into which charges of the molten metal are transferred from the intermediate furnace as they are required. For illustration, the intermediate furnace may hold from 200 to 300 tons of the desiliconized and partially-decarburized metal, and the basic open-hearth furnaces in which it is finally treated may have a capacity of 50 tons. In the open-hearth furnaces the metal is treated with basic additions in the usual manner, the phosphorus is eliminated, the carbon reduced to the desired point, and the temperature of the metal is raised to the degree required for casting. The process enables the inventor to dispense with scrap, for the metal is rapidly desiliconized and partially decarburized before it is delivered to the open-hearth furnace, and the final operation in the latter is short.

J. Vernon (791,170, May 30) patents a "physic" to be introduced into the ladle before the molten steel in order to prevent blow-holes and piping. This physic contains in the condition of fine powder, ferromanganese and aluminium and refined silicious cullet, obtained either from native rock-crystal or from lead glass, which is comparatively free from lead and other metallic impurities. The cullet will yield silicon, and in melting will act mechanically upon the molten mass, besides increasing somewhat the temperature thereof. Other ingredients of the mixture are porous earth of a clayey or loamy nature, and what is known as "kelp," viz.: the calcined ashes of seaweed. The former, from its comparative lightness, will have a tendency to rise through the molten mass, thereby providing an escape for the occluded gases, which, if they remain in the metal, are the cause of the blow-holes. The kelp, properly dried and ground, contains sodium, potassium, carbon and iodin, and has the effect of decomposing the phosphorus and sulphur, which can then escape in the form of vapor.

BRIQUETTING.

E. Pohl (792,449, June 13) produces solid lumps from granular, or pulverulent, iron ores in the following way: The pulverulent ore is agglomerated by means of liquid slag, which falls down as rain or spray onto the ore and is intimately mixed with it. The process is carried out in a revolving retort. Finely divided, white-flowing liquid blast-furnace slag is used as binder where available; Thomas slag may also be used or molten iron.

W. A. Köneman (791,799, June 6) makes ore-briquettes for blast-furnace smelting by mixing together ore-fines and finely divided uncoked bituminous coals and anthracite coals, and adding to the mixture a binding agent (like lime or a lime mixture or gelatinous matter or slag), with or without a fluxing material. The mass is then formed into briquettes and dried without coking, so as to provide briquettes of mixed are and fuel for subsequent coking by the direct heat of the blast furnace.

J. Furnkawa (784,850, March 14) casts a shell of matte around a core or ball of fine ore, flue-dust and smalls of coke.

GOLD AND SILVER.

John A. Just (788,912, May 2) patents the following process of extracting gold and silver from ores: The ore is first subjected to a chloridizing roasting, and is then washed and introduced into the extraction vessel, which contains a calcium chloride solution, and into which chlorine is introduced at the bottom. The chlorine gas diffuses readily and auric chloride is formed, while the calcium chloride solution simultaneously dissolves the silver chloride. The solution is maintained in an acid state by a limited amount of hydrochloric acid. The solution goes into the precipitating tank, in which air is used for stirring, while the gold and silver are precipitated by means of metallic zinc. The solution is separated from the precipitate by a filter press.

MISCELLANEOUS.

S. Groh (783,429, February 28) patents details of construction of a regenerative furnace, the object being to bring both the gas and air at proper and equal temperatures to the point of mixture and combustion. For this reason the inventor controls the passage of the products of combustion through the conduits, so as to subject the checkerwork, through which the air subsequently passes, to the action of the major portion of the heat units of the products of combustion. Thus the cold air assumes in its passage a temperature approximating that of the gas at the point of mixture.

T. C. King (781,887, February 7) patents a system for the removal of slag from smelting furnaces. The slag is delivered to a certain kind of pit, where it is granulated by a stream of water The water is heated to the boiling point, and nearly half of the water is converted into steam. This heat is utilized for heating or power purposes, while the granulated slag is delivered to an endless conveyor.

M. Murphy (783,535, February 28) patents details of construction of an elliptical smelting furnace. Its roof is arched and its ore-bed is an inverted arch, having its ends extended into the side walls of the body of the furnace. Stay-bars extend longitudinally through the body of the furnace below the ore-bed and support the same, while other stay-bars are arranged against the outer sides of all the walls of the furnace; extensible connecting rods securing the stay-bars.

C. M. Gunn and W. D. Mulloy (782,334, February 14) patent details of construction of a hot-air blast stove for supplying heated air to smelting furnaces. The air tubes are so connected as to distribute the strains of contraction and expansion uniformly throughout the series. The end tubes of each series are connected with end boxes for receiving the cool air and delivering the same to the feed-tube of each series, and the hot air from the discharge tubes and delivering the same to the supply pipe for the heated air.

F. L. White (783,234, February 21) patents details of construction of hot-blast stoves. They consist essentially in an improved arrangement of supporting arches in the lower regenerative chamber, while the regenerative passages between the upper and lower regenerative chambers are formed in a body of novel checker-brick laid so as to break joints and present a bonded structure.

L. de Rome (782,438, February 14) patents details of construction of foundry furnaces, the principal object being the utilization of fuel-oils in the melting with its consequent saving of time and fuel. A depressed inclined hearth is surmounted by a dome, and with an outlet for the products of combustion. An oil-burner has its flame directed against the metal piled upon the hearth, while a well contains a crucible beneath the drip from the hearth. A second oil-burner has its flame directed around and about the crucible.

L. O. Brightbill (783,413, February 28) cools the walls of a furnace by spreading cooling water from a trough mounted on the wall, in a continuous film over the surface.

G. L. Davison and D. R. Mathias (783,778, February 28) patent details of construction of a water-cooled arch particularly adapted for use, in conjunction with open-hearth furnaces.

J. Reuleaux (782,697, February 14) patents details of construction of a continuous heating furnace for heating billets; J. H. Vinton (782,888, February 21) details of construction of a furnace for heating toe-calk blanks.

J. T. White (782,298, February 14) patents details of construction of a spout for blast furnaces. The spout comprises a hollow shell provided in its outer side near one end with a plurality of water-inlet supply openings arranged in alinement and longitudinally with the spout, and near the other end with a water-outlet opening; heads are detachably connected with the shell. The object is to increase the life of the spout and facilitate repairs.

A. R. Brink (780,592, January 24) patents details of construction of a spout, comprising a number of water-jacketed, self-contained detachable sections joined together; each section is supplied with water independently of the others.

S. Peacock (781,546, January 31) patents details of construction of a gas-seal for rotary roasting furnaces; W. Stubblebine (782,082, February 7) details of construction of a rotary puddling or busheling furnace or a furnace for treating scrap metal.

BOOK REVIEWS.

The Theory of the Lead Accumulator. By Dr. F. Dolezalek. English translation by Dr. Franz L. von Ende. New York: John Wiley & Sons. 241 pages. Price, $2.50.

Prof. Dolezalek's German monograph on the theory of the lead accumulator is undoubtedly the best and most up-to-date book on the subject. While essentially a concise theoretical treatise, we believe it has, nevertheless, exerted a healthful influence on storage battery engineering. It has brought home to the practical man forcefully the enormous importance of perfect diffusion of the acid into the pores of the active mass, although this point of view has apparently been pushed a little too far in a few instances in the book. The main reason, however, why this book has been received so very friendly, and even enthusiastically, by the electrochemical public at large, is that it is the only work in which the modern theories of solution and the dissociation theory have been applied with strict consequence to a single practical engineering problem. While not pretending to teach theoretical electrochemical principles, yet it is in effect the best book which electrochemical engineers may take up to become acquainted with the theory. In this respect it is a little masterpiece.

It is gratifying to have the book now issued in an American translation. The translator has accomplished his task carefully and faithfully. But there can be no doubt that in some small details his faithfulness towards the original German manuscript goes too far. One understands why in the original German book the reader is referred to a German translation of Gladstone and Tribe's English paper. But it would seem almost ridiculous that in the book before us (page 2), which is intended for American readers, again the German translation is referred to, and the original English paper is not mentioned. In the enumeration of books on storage battery engineering in the preface, Mr. Lamar Lyndon's well-known book—certainly the best book in English language on this subject—should not have been omitted. In a few instances the translation of special technical terms is too literal. "Degree of efficiency" (page 174) and "working efficiency" (page 175) are certainly unfortunate. In the whole, however, the translation is careful and reliable.

Lifting Electromagnets in the Iron and Steel and in the Tin Industries.

Electromagnets have in the past been mainly used for creating a magnetic field, as in dynamos and motors, while the simplest application of an electromagnet, namely, for lifting purposes, was usually limited to small scale operations, to instruments and to work in the laboratory. However, this situation has been completely changed in recent years, and the lifting electromagnet is now an important piece of machinery in modern iron and steel works.

This development is essentially due to two leading tendencies in modern manufacturing methods: to push the output of a plant to the utmost, and to reduce as much as possible the number of workingmen. The first requirement is fulfilled by a lifting electromagnet on account of the simplicity of its action: it is lowered on the piece or pieces to be lifted, the switch is closed and the magnet is thus instantaneously energized, the load is hoisted and transported; the switch is opened and the load thereby instantaneously released. This means that all the time is actually spent in hoisting and transporting; no time is lost as with chains where the piece must be barred up and the chains attached and adjusted, and later on must be released. With the lifting magnet the attaching of the iron masses to the magnet, and afterwards their release are essentially automatic and instantaneous. All the heavy work is done by the electromagnet and crane. This explains why only a single workingman is required for most cases. He has nothing to do but to close and open the switch of the electromagnet and attend to the crane. Only in such cases where several pieces as billets are to be carried at each lift, a ground helper is required to adjust the several pieces.

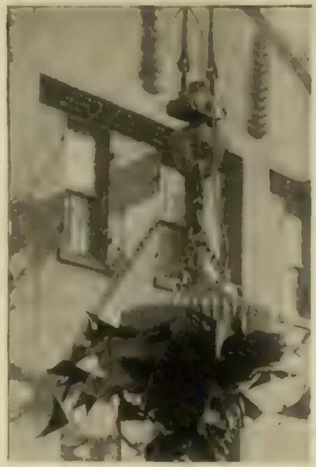


FIG. 1.—LIFTING MAGNET FOR TIN SCRAP.

The Electric Controller & Supply Co., of Cleveland, Ohio, has made a specialty of the design and construction of such lifting electromagnets, and their magnets are the result of years of experience in this field. The case is a heavy casting made from such a grade of steel that when the circuit of the energizing coil is opened the magnet drops its load instantly, so that no time is lost. The energizing coil is form wound, heavily insulated and treated by a vacuum impregnating process which insures it against grounding, or short-circuiting. The coil is completely enclosed and protected by the case, which is ribbed to provide for rapid radiation of the heat of the coil. The current is led to the coil through a heavily guarded plug connection. This plug is simply pulled out when it is desired to detach the magnet from the crane hook.

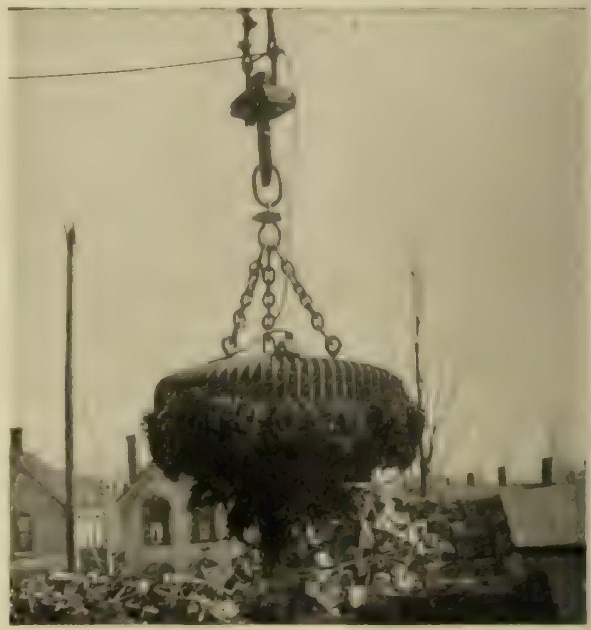


FIG. 2.—HANDLING IRON AND STEEL SCRAP.

These lifting magnets are suitable for handling pig-iron, steel or iron scrap, bolts, billets, blooms, slabs, or cold ingots in quantities, rails, pipes, sheets or other special shapes in quantities or under special conditions. The "pig magnet" of the Electric Controller & Supply Co. is made in two sizes; the smaller size handles an average of 500 pounds pig iron per lift; the larger size from 1100 to 1400 pounds. These magnets are particularly efficient for such material which can be handled direct from the car to the regular open-hearth charging-boxes, or to the stock pile. They are also very suitable for handling light melting stock, shearing scrap, rod scrap, bolts, nuts, punchings, rivets, spikes, etc. This class of material is very difficult to handle by hand. It cannot be shoveled and is very slow and costly work where a fork is used. It is light and easily and quickly magnetized. The larger type of magnet is especially useful for unloading or loading cars with pig-iron billets, scrap, etc. The time required to load or unload with lifting magnets in such cases will vary from one to two minutes per ton, depending almost entirely on the skill of the crane man and the amount of crane travel required, as practically no time is required to attach or release the load. Iron or steel turnings or borings, scrap, bolts, rivets, punchings, or similar small scrap, so difficult to load or unload by hand, are handled at a rate of 600 to 1200 pounds per lift by the larger type of "pig magnets," as is shown in Fig. 2, The handling of bundles of iron wire by lifting magnets is shown in Fig. 3.

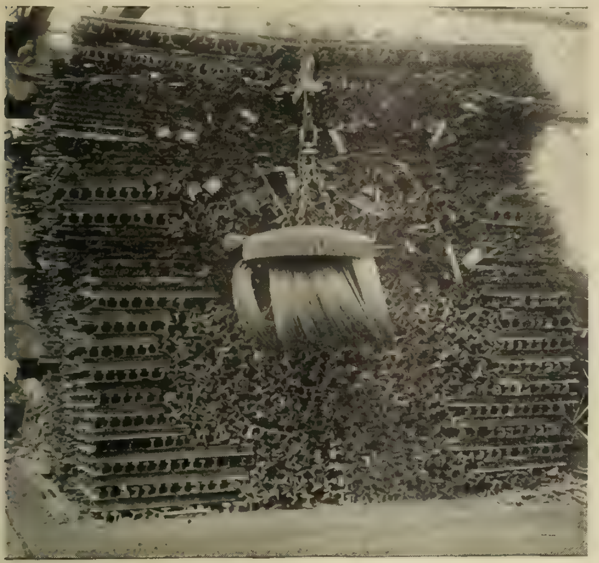


FIG. 3.—HANDLING BUNDLES OF IRON WIRE.

The lifting magnets are now also playing an important rôle in the tin industry for the reason that on account of the limited supply of tin ore the electrolytic detinning of tinned iron scrap ("tin cans" for preservers, etc.) has become a profitable industry. The handling of tin scrap by the lifting magnet is shown in Fig. 1.

Producer Gas Plants in Scotland.

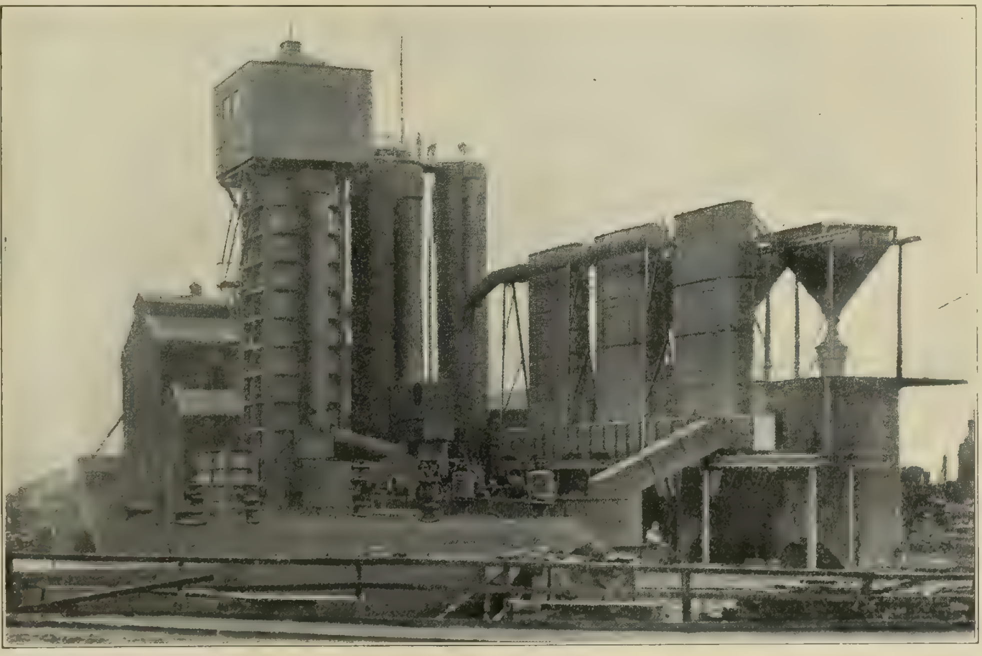
With reference to an article of Mr. F. C. Perkins in our February issue (page 85), we have received some notes by Mr. Edward J. Duff, of Duff Brothers & Co., Liverpool, referring especially to the Parkhead Steel Works, of William Beardmore & Co., Glasgow. Mr. Duff states that this producer gas plant was erected to his patents, designed and started into operation by himself, and is entirely a Duff gas installation. Some further notes concerning this plant may be of interest.

This plant was the first steel plant in Scotland in which the recovery of ammonia from the fuel has been undertaken. Initially, only five producers were installed, but they proved so successful that there are now 10 producers in operation at the Parkhead Steel Works. The gas is used for steel melting, for making large forgings for guns, shafting, etc., up to 60 tons weight, for the manufacture and treatment of armour plate, and in the running of a number of large gas engines, some of them 1000 horse-power each. Another installation, first with 3, and later with 5 producers, is now successfully at work at William Beardmore & Co's large, new shipyard at Dalmuir on the Clyde, which was also installed by Mr. Duff. The first installation, containing 3 producers, is shown in the adjoining illustration. The bulk of the gas is used in gas engines of the Oechelhaeuser type, built by Messrs. Beardmore & Co. themselves. The gas engines drive dynamos for the supply of electric power. Both plants burn Scotch bituminous slag, a very cheap fuel.

A brief description of the operation of the plant should be of interest.

The coal is brought into the works in wagons, from which it is tipped into a large boot receiver below rail level. By means of a chain-and-bucket continuous elevator, it is raised to an upper platform above the producers. Along this platform the coal is conveyed and dropped at will into any one of the storage hoppers provided for each producer feed. The coal is shot as required into the producers through gas-tight trap-boxes.

The gas issuing from the producers is partly cooled by passing through a regenerator by which a portion of the heat is transmitted through tube surfaces to the incoming air and steam. The incoming air and steam are fed to the superheaters through the pipe shown near the top of the oval-shaped chambers above the producers; they leave them near the base and pass through a pipe to the blowing chamber of the producer below the fuel bed. The superheaters are provided with dust separating boxes, and the same is the case with the rectangular gas main, which is shown in front of the base of the superheaters, and into which the gas passes from the superheaters. The gas then passes down the connecting pipe into a scrubber on the ground level, where it meets with



GAS PRODUCER PLANT.

a shower of water whereby any soot or remaining dust are removed from the gas and also a portion of the tar.

The gas then passes into the ammonia tower which is the square tower shown at the left of the illustration. Here the gas passes upward through a shower of sulphuric acid liquor, and ammonium sulphate is formed, from which the commercial crystal sulphate of ammonia is derived by subsequent treatment. After leaving this tower the gas is cooled with water in the "gas tower;" it is then put through a centrifugal washer and a saw-dust scrubber, and is then ready for use in the gas engines.

The third tower in the row is the air-heating tower in which the water heated by cooling the gas in the "gas tower" is again cooled by incoming air and made ready again for the purpose of cooling gas. The plant is thus recuperative for both heat and water. The air then passes from this tower to the superheaters and the gas producers as already described above.

The American representative of Messrs. Duff Brothers & Co., is Mr. A. B. Duff, 4716 Maripoe Avenue, Pittsburg, Pa.

Metallographic Equipment of Laboratories.

The enormous industrial importance of high-speed tool steels has been the reason for attracting in recent years the attention of numerous investigators to a systematic research of ferro-alloys and of alloys in general. In this research the limitations of chemical analysis were soon recognized. It is now clearly understood that "chemical composition is not everything," *i. e.*, that the properties of a "solid solution" depend not solely on the chemical composition, but to an important degree on the previous history and treatment. Of course, the effect of annealing or tempering of steel has long been perfectly known. But the reason why the physical properties of metals, especially of alloys, depend on the way in which they were cooled from the state of fusion, has only been recognized in recent years, when the microstructure of alloys was systematically investigated and considered and viewed in the light of the phase-rule. The experimental research was made possible by the fact that if a hot metal is suddenly cooled by chilling it in cold water, its microstructure at the high temperature is fixed and preserved by the sudden chilling, so that it may be afterwards examined in the cold metal by polishing and etching it in the usual manner. Some general principles of metallography were given in an article by Dr. Wm. Campbell in our February issue, 1904.

In view of the importance of the subject, a brief description of the new metallographical equipment of the Institute of Technology, of Aachen, Germany, should be of interest. This institute has lately taken up metallography in its official plan of instruction.

The grinding of the specimens intended for examination of the microstructure is done, according to the most universal practice in Germany, on rotating triplex cross-glued wooden discs, pasted with the best emery paper. The following numbers are successively employed: 3, 2, 1G, 1M, 1F, 0, 00, 000, 0000. Then the specimens are polished with finest jewelers' rouge on a disc covered with cloth.

The entire microscopic and microphotographic equipment, except, perhaps, one microscope stand, came from the shops of the firm of Carl Zeiss, of Jena. This microscope stand serves in connection with the lower power lenses for the preliminary investigation of the ground specimen, which quickly and con[illegible]ently follows the course of the grinding, polishing and [illegible].

For close examination of the prepared specimens, as well as for photographing them, the microphotographic apparatus is used. It consists of a Martens microscope with illuminating arrangement and photographic camera. They are placed separately on iron brackets, bedded in the wall, so as to prevent vibrations, which is an absolute requisite to get sharp photographs with the stronger objectives.

The strongest magnification is, for direct observation, 1000, and for photographs with the sensitive plate 1 meter away from the eye-piece, 4000 diameters.

The construction of the Martens stand is shown in Fig. 1. Externally it differs from the ordinary form through the horizontal, instead of vertical, position of its optical axis. The tube, moved by wheels, carries an extension tube provided with a millimeter graduation for setting it at the length prescribed for the several objectives. The collar is slipped on the holder of the eye-pieces, and in photographing, forms the opaque coupling of microscope and camera; actual contact between them is, however, avoided. The shell on the camera, when brought up to the microscope, enters the collar.

If the stand is to be used for photographing without eye-piece the extension tube is entirely removed and replaced with a so-called light-excluding shell, which may likewise be given a light-proof connection with the camera without touching it. This light-proof shell is shown on the Martens stand in Fig. 1. The objective can be given a rapid movement by the wheel, or a fine one by the micrometer screw. On the latter a travel of the table as small as 0.005 mm. (0.0002 inch) may be read directly. The stage for mounting the object is revolvable, and can, by co-ordinate drives, be moved in two mutually perpendicular directions, thus permitting a systematic examination of the specimen. Since it is generally necessary to mount the object exactly perpendicular to the axis of the microscope, there is placed on the stage proper a second, smaller, stage parallel with it and furnished with adjusting screws, by means of which an exact position may be quickly and positively given. The specimen, glued to a glass slide with a mixture of rosin and beeswax, is held against the smaller stage with screw clamps.

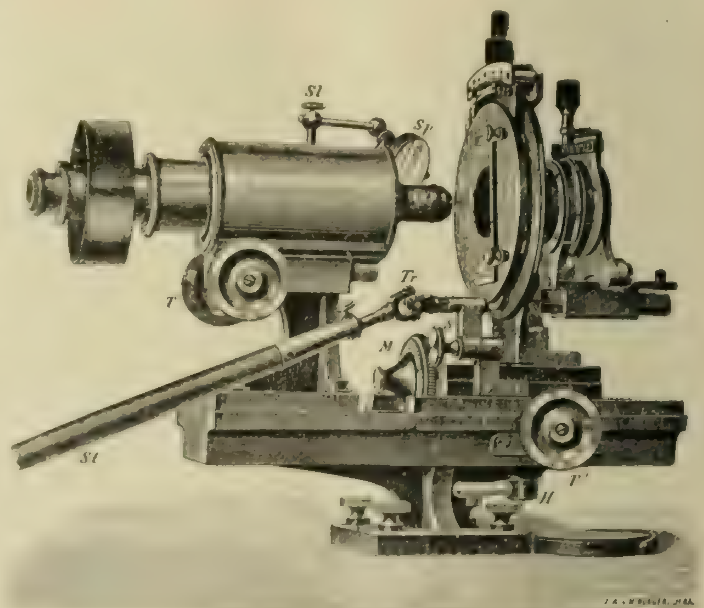


FIG. 1.—MARTENS STAND.

The illumination, which in the case of metal sections, as with all opaque objects is by superficial light, is secured by the movable mirror shown in the illustration, or a thin, flat glass, so arranged that the day or lamp light falling perpendicular to the optical axis is reflected upon the specimen. Since this method of lighting is applicable only with objectives whose distance from the specimen is sufficiently great, resort is had in case of the stronger dry systems, as well as of the homogeneous immersion lenses, to the so-called vertical illuminator, which is inserted between tube and objective. From a side window before which, by means of pins, an iris diaphragm may be put up, the light falls on a little prism, by whose hypothenuse face it is totally reflected through the objective to the specimen.

A scheme of the lighting arrangement used with the vertical illuminator is as follows: The rays coming from the source of light—in our case a self-regulating arc lamp—pass through a compound lense system, consisting of the double lense part and a single lense part, and fall on the window of the vertical illuminator. Between the double and single lenses is inserted a water chamber for the absorption of the co-called "heat rays" having great wave-lengths. For reducing the field of vision there is an iris diaphragm.

For examining and photographing larger objects, broken pieces, etc., where high magnification is not needed, but facility of adjustment is desirable, the binocular coarse-work stand of Braus-Druener is used, Fig. 2. This also is a Zeiss instrument.

The apparatus, mounted on a heavy rectangular bed, with three rack and four swiveling movements, is adapted to the systematic inspection of large irregular bodies, wherein the comparatively long distance from the object is of advantage. Further, by means of the stereoscopic camera, interchangeable with the double tube, this stand performs a very valuable service, and can best be appreciated in that connection. In direct examination with the lenses provided, a magnification of 10 to 65 diameters is attained, and in photographing with the stereoscopic camera 1.6 to 6.2 diameters. The depth of the sharp picture necessary to produce the idea of solidity varies, according to the degree of magnification, from 5 to 10 mm. (0.2 to 0.4 inch).

Finally, must be mentioned an apparatus employed for manifold metallographic services—the tube furnace. The Institute for Iron Metallurgy has for many years employed almost exclusively the Heraeus electric furnace, in which the heating resistor consists of platinum foil 0.007 mm. thick, which is wound in spiral form on a Marquardt tube. These furnaces are notable for swift, certain and regular service, and are to-day so improved with suitable heat-guarding arrangements that, for example, in a furnace 500 mm. (19.7 inches) long and 30 mm. (1.18 inches) broad a temperature of 1450° C. can be continuously produced without fear of the foil burning through. Naturally, for furnaces of large section the high temperature permissible for continuous experiments is somewhat less.

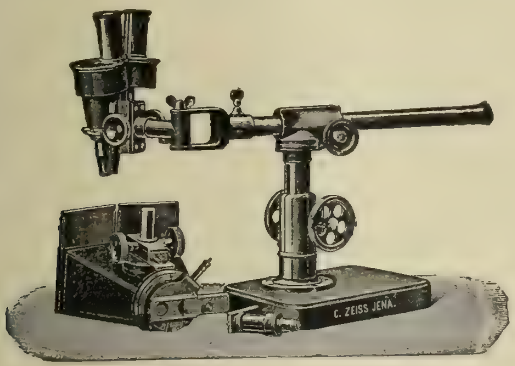


FIG. 2.—BRAUS-DRUENER STAND.

Water-Power Development in Wisconsin.

The Rhinelander Power Co., of Rhinelander, Wis., composed largely, if not entirely, of local parties, has completed arrangements to develop the water power of the Wisconsin River at a point about 6 miles below the city of Rhinelander. A dam will be built, giving a head of about 24 feet, and sufficient water is obtainable to develop an average of about 600 hp. In the rainy season more than 2000 hp., at a conservative estimate, is obtainable.

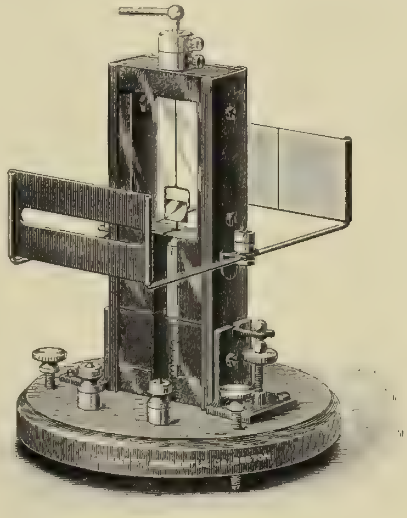
The Allis-Chalmers Co. will furnish the entire electrical equipment, which includes three 400-kw, 1100-volt, three-phase, 60-cycle generators, with three 70½-kw, 120-volt separately-driven exciters, six 150-kw, 10,000-volt transformers and complete switchboards at both the generating station and the sub-station at Rhinelander.

The current will be carried at 11,000 volts over a single three-phase circuit to the sub-station at Rhinelander, a distance of about 6½ miles, and from this sub-station will be distributed at 1000 volts for illuminating purposes, both commercial and street lighting, and also at 440 volts to the mill of the Rhinelander Paper Co. and other industries in the neighborhood of the sub-station.

This plant is one of the first long-distance transmission plants on the Upper Wisconsin River. It is expected that many more enterprises of a similar character will be undertaken in that region within the next two years.

Galvanometer.

The adjoining illustration shows the W-K, No. 2 d' Arsonval galvanometer, made by the L. E. Knott Apparatus Co., of Boston, for use in schools. The sensibility is 20 megohms. It is dead-beat, coming to rest after only one swing through the zero. Special points of the design are the scale attachment (reflecting non-parallax method), the strict proportionality of the scale, and the accessibility and visibility of the moving parts, since glass slides on front and back render all parts visible and accessible. On account of the open construction no special illumination is required.



GALVANOMETER.

Industrial Notes.

The J. T. Baker Chemical Co., of Easton, Pa., has issued its first price list of c. p. chemicals. An important innovation is to print an exact statement of the analysis on every bottle containing a "c. p." chemical. The company has a New York office at 164 Front Street, and will be represented by the Chas. E. Sholes Co., Manufacturers Agents.

Coal Handling Machinery.—The C. W. Hunt Co. has issued a profusely illustrated pamphlet on coal-handling machinery for power stations, boiler rooms, coaling stations, gas companies, coal yards, shipping docks, manufactories, etc.

Nickel Steel.—The International Nickel Co., 43 Exchange Place, New York City, has issued a pamphlet on "The Applications of Nickel Steel." It is an alphabetical index of and various applications which nickel steels, containing 3 to 45 per cent nickel, have found in this country, France, Germany and England. The addition of 3 per cent to 4 per cent of nickel to steel increases the proportional elastic limit, adds to the ductility of the steel, increases its resistance to compression, increases its resistance to abrasion, increases its resistance to shock, and increases its toughness. Nickel steel is, therefore, a safer material than carbon steel."

MANILA ROPE.—The **C. W. HUNT CO.**, of West New Brighton, S. I., N. Y., have issued a very neatly printed and illustrated pamphlet on manila rope transmission and hoisting, the author being Mr. C. W. Hunt, who has long been recognized as one of the highest authorities on the subject. The pamphlet is far more than a mere trade catalogue, being, in fact, a concise treatise on the subject, with much useful information for engineers. The pamphlet begins with a description of the construction of manila rope. This is followed by an account of the use of ropes for power transmission, rope-driving diagrams and a table and diagram for the horse-power of transmission ropes at various speeds being given. Then follow notes on splicing and coupling of transmission ropes, knots, hitches and bends. A short account is then given of the use of ropes for hoisting. The appendix contains reprints of two papers, presented by Mr. C. W. Hunt on rope driving and on working loads for manila rope, before the American Society of Mechanical Engineers, and some shorter notes, among them a United States Navy test, showing 14.6 per cent less friction for stevedore rope than for ordinary three-strand manila rope. The stevedore rope is a special manila rope, made by the C. W. Hunt Co., in two types, for power transmission and for hoisting respectively.

HARD RUBBER FOR CHEMICAL PURPOSES.—We have received from the **AMERICAN HARD RUBBER CO.**, 9 Mercer Street, New York City, their illustrated catalogue on hard rubber pumps, pipes, fittings and other utensils for chemical work. As is well known, hard rubber is used with acid and alkaline solutions, which either corrode metal or are contaminated by contact with it. In many cases the use of hard rubber has solved that ever-present and fundamental problem in wet processes—to keep the solutions clean, since, as was once said, everything may be accomplished with pure solutions, nothing with foul solutions. In hard rubber pumps all the parts which come in contact with the liquid are made of hard rubber; these are held by and mounted on iron parts, which take all strain other than internal pressure, due to head of discharge. The iron plate can be bolted to any suitable base. The company manufactures hard-rubber double-acting pumps, either with steam drive or with belt drive, or directly connected to electric motors, and single-acting hand pumps or power pumps. The variety of purposes to which hard rubber is applicable is shown in the catalogue by illustrations of hard rubber pipes (made in sizes from ½ to 4 inches), hexagon couplings, angle couplings or bends, elbows or quarter bends, tees, nut unions, caps, lock nuts, plug cocks with screwed socket ends (all in sizes from ½ to 4 inches), plug cocks with nipple ends for hose or threaded (½ to 2 inches), plug cocks, bibb ends, nipple end for hose or threaded (½ to 1 inch), air cocks (¼ inch opening, ⅛-inch iron pipe thread), buckets (1, 2, 3 gallons), mugs (1 pint), measures (1 pint, 1 quart, 2 quarts), ewers (2 quarts), funnels (⅛ pint to 1 gallon), floats (3¼ to 7 inches), dipping baskets and bottles for hydrofluoric acid (½ to 16 ounces). An interesting and apparently new application of hard rubber was recently mentioned in this journal (our June issue, p. 221), namely, for filter press linings and cocks.

We have received from the **INTERNATIONAL INSTRUMENT CO.**, of Cambridge, Mass., a set of illustrated pamphlets on laboratory instruments made by this company. Among them are [illegible] laboratory clock, galvanometer lamp, absorption [illegible] small high-speed engines and motors), absorption rheostat (liquid), Johnson cement testing machine (for determining the tensile strength of cement), distributing switchboards, adjustable laboratory stand (for general work, but especially useful where electrodes have to be immersed in and then withdrawn from solutions), Ferguson rain gage, several types of centrifuges, characterized by high-speed and large bucket capacities, primary batteries (closed circuit type: Excello battery; open circuit type: Samson, Nosmas and Columbia batteries), secondary batteries (chloride accumulator).

Messrs. **QUEEN & CO.**, of Philadelphia, have issued a new circular, No. 1204, on condensers and self-induction apparatus. This circular shows a wide variety of condensers, from the most elaborate standards down to the small tin-case condensers used extensively in telephone work. Standards of self-induction, both fixed and variable types, are illustrated and described. Curves are also given, showing the absorption and leakage in the standard condensers made by this company.

WATER-POWER EQUIPMENT.—The **WELLMAN-SEAVER-MORGAN CO.**, of Cleveland, Ohio, has issued an illustrated pamphlet on water-power equipment for low heads. The installations of this company for heads up to 100 feet embody the Jolly-McCormick turbines, which are furnished in a variety of settings and combinations, some of the most common of which are as follows: Single vertical setting in iron, single horizontal setting in iron, double horizontal setting in iron casing, with one or two inlets and one or two discharge openings; horizontal setting with any number of wheels on shaft, and variable arrangement for discharge; double vertical setting with turbines located above each other. The exact form of installation depends, of course, on local conditions, such as head, speed of flow, power required, contour of ground on which the plant is located, etc. In this respect a table at the end of the pamphlet should be very useful, summing up in eighteen questions the most important data which are required to decide on the best form, and give an estimate of the cost of the installation. The main part of the pamphlet gives illustrated descriptions of standard vertical cylinder-gate wheels; a form of balance gate with which the regulation is secured by adjusting the opening of the chutes or gates as required; construction of standard runners, and design of standard horizontal single turbines, as well as of various types of special horizontal turbines. On four pages complete power tables of the standard turbines of the Wellman-Seaver-Morgan Co. are given, based on tests of the whole series of nineteen patterns for fifty-two different heads between 4 and 100 feet; speed in revolutions per minute, water and horse-power are given for each of the nineteen patterns and each of the fifty-two heads. The efficiency is between 80 and 86 per cent.

The **LEEDS & NORTHRUP CO.**, of Philadelphia, has just issued the second edition of their profusely illustrated catalogue of their electrical measuring instruments, for which they received a grand prize at the St. Louis Exposition. The catalogue covers condensers, galvanometers, slide wire or meter bridges, Carey Foster bridge, resistance standards, apparatus for measuring low resistances, standard resistance boxes, rheostats, Wheatstone bridges, standard high resistances, electrodynamometers, self-induction apparatus, portable testing sets, cable testing apparatus, cable splicers and lineman's fault finders, potentiometers, photometers, curve tracers, high-frequency apparatus and induction coils.

We have received from Messrs. **WM. GAERTNER & CO.**, of Chicago, three illustrated catalogues. One deals with interferometers and accessories; the second with instruments of precision, laboratory apparatus and astronomical apparatus; the third with universal laboratory supports for manifold purposes.

Circular 1104 of the **WESTINGHOUSE ELECTRIC & MFG. CO.** deals with portable instruments, voltmeters, ammeters and wattmeters for direct and alternating current, power factor meters and portable testing transformers.

Single-Phase Vapor Converter for Charging Storage Batteries.—The Cooper Hewitt Electric Co. has issued bulletin No. 8, describing the mercury vapor converter for charging small storage batteries from a single-phase alternating-current supply. The action of the converter is well understood from the behavior of the mercury vapor lamp. The "type PA" converter provides, in its present standard design, for 30 amps. direct current, and a d. c. voltage of from 80 to 115. Special outfits adapted to give d. c. voltages lower than 80, or exceeding 115, and not over 240, can be supplied. The converter is supplied for any single-phase a. c. supply voltage between 25 and 150 cycles. In case the a. c. supply does not exceed 450 volts, an auto-transformer is used for adapting the a. c. voltage to the outfit. On higher voltages a step-down transformer with a special secondary winding is used.

Electric Drive of Machine Tools.—The Crocker-Wheeler Co., of Ampere, N. J., has published as bulletin No. 58 a well-illustrated, concise treatise on the individual motor drive of old machine tools, the author being Mr. R. V. Wright. After a brief discussion of the problem, the application of the electric drive to various machine tools is discussed in detail.

Agate-Shellac.—Under this name a new shellac has recently made its appearance in Europe, and is now being introduced in this country by Mr. Felix Hamburger, 90 William Street, New York City. While being used for many purposes it is stated to be specially applicable as an insulating lacquer for electrical engineering purposes. According to tests of the German Reichsanstalt, in which the agate-shellac was compared with ground Indian shellac, the insulation resistance with agate-shellac was found to be from six to nine times greater than with Indian shellac, while the perforating voltage is slightly lower. It is stated that the Siemens-Shuckert Co. has adopted the shellac for use in their works.

Big Crank Shafts.—The Bethlehem Steel Co. is at work on three crank shafts which will weigh 86,600 pounds each when finished. They are turned out of solid steel ingots, 25 x 4 x 4 feet, and are intended for three Snow gas engines, which are to drive 4000-kw Crocker-Wheeler alternators, the largest gas-engine-driven generators ever built, ordered by the California Gas & Electric Corporation.

Vanadium.—We have received from the Vanadium Alloys Co., of New York City, a pamphlet giving the results of extensive tests of vanadium steel, carried out by Messrs. Williams & Robinson, Queens Ferry, England.

The Frenzel Smelting & Refining Co., of Denver, Col., is shortly to be incorporated under the laws of the State of Colorado. The erection of the plant will be started in about four weeks, with a capacity for treating 50 tons of ore per day at the start, while an extension to 100 tons per day is planned for the future. The output will consist of alloys of vanadium, tungsten, uranium, molybdenum and titanium, for use in the steel, cast iron, aluminium and copper industries. Electric furnace processes will be largely employed. Mr. A. B. Frenzel, Equitable Building, Denver, Col., states that he will be glad to correspond with parties with respect to improvements in the reduction of rare elements from the ores.

The Tacoma Smelting Co., Tacoma, Wash., has ordered from Allis-Chalmers Co., of Milwaukee, a steel water-jacketed blast furnace for treating copper ores, the dimensions of which are to be 42 x 160 feet at the tuyere line. It will be provided with a specially designed tuyere box, which, it is believed, will prove more satisfactory than any tuyere box yet constructed.

The Sociedad de Minas y Fundiciones, Carrizal, Bajo, Chanarcitos, has ordered from Allis-Chalmers Co., of Milwaukee, the complete equipment for a 50-ton copper smelting plant, to be erected at Carrizal, Bajo, Chili, South America. The installation will include a blast furnace and blower, with auxiliary apparatus, and a Bullock type "B" generator to supply current for an electric light plant. Mr. William M. Martin, the manager of the Chilian company, was in Chicago recently for the purpose of purchasing this machinery.

The Arizona Smelting Company has ordered for its plant at Prescott, Ariz., the complete equipment for a sampling mill. The contract also calls for an electrically-operated converter and two reverberatory furnaces with auxiliary apparatus. This very large order is another indication of the increasing activity in the copper fields of the Southwest. The entire equipment will be furnished by Allis Chalmers Co., of Milwaukee.

We received from The Mine & Smelters Supply Co., of Denver, Col., a voluminous and profusely illustrated, complete catalogue of machinery and supplies. In spite of the conciseness and brevity of the descriptions of the various machinery dealt with, the catalogue contains 891 pages. This already indicates the completeness with which the mining and metallurgical field is covered. The catalogue is divided into nine principal sections, dealing respectively with the following subjects: Engines, boilers and feed-water heaters, hoisting engines, pumps, water-wheels and hydraulic machinery, mining, milling and smelting machinery, tools and supplies, assayers' supplies, electrical machinery, iron and wood-working machinery, ready reference tables and useful information. As an example the contents of the fourth section, devoted to mining, milling and smelting machinery, and covering 114 pages, may be briefly summarized. There are given in this section concise illustrated descriptions of air compressors, rock drills, crushing and grinding machinery, samplers, elevators, screening machinery, ore feeders, classifiers, concentrating tables and jigs, amalgamated fans, settlers, clean-up pans and barrels, amalgamated sieves, chlorination barrels, steel tanks, furnaces, lathes for making zinc shavings, cyanide process machinery, ore cars, conveyors, aerial wire rope tramways, dryers, roasting furnaces, smelting furnaces, etc. The book is concluded by a very useful alphabetical index covering 30 pages.

The New Castle Portland Cement Co., New Castle, Pa., has placed an order with Allis-Chalmers Co., of Milwaukee, for four rotary kilns, each 100 feet in length and 7½ feet in diameter at the large, and 6½ feet at the small end. These kilns, which will have a large capacity, are of extra heavy design throughout, and when completed will represent the highest type of machinery of this class in use. The tendency of late in the cement business has been to lengthen the kilns. A few years ago kilns measuring 60 feet were practically the standard, but kilns which are now being erected range from 80 to 125 feet in length, 100 feet being about the standard. In addition to the kilns Allis-Chalmers Co. will furnish a crushing plant consisting of one No. 6 "K" Gates rock and ore breaker, two No. 4 Gates breakers, and one No. 6 elevator. The operation of these is as follows: The No. 6 breaker receives the limestone as it comes from the quarry and does the preliminary crushing. The discharged product is divided and sent to the two No. 4 breakers, which prepare it for the grinding.

Buffalo Meeting American Chemical Society.—There were some interesting exhibits at the recent convention of the American Chemical Society in Buffalo. The Buffalo Dental Manufacturing Company exhibited in the convention hall their full line of laboratory appliances, including muffle and crucible furnaces, gas burners of many types for laboratory purposes, laboratory foot blowers, gas furnaces, tube furnaces, vacuum pumps, gasoline gas generators, water heaters, etc. The Bausch & Lomb Optical Co. exhibited some of their Baloc glassware for general laboratory purposes.

Articles of incorporation have been filed in Albany by the Bausch, Lomb, Saegmuller Co., manufacturers of engineering, astronomical, physical and other instruments of precision. The Bausch & Lomb Optical Co. is to be the sales agent of the new corporation, whose manufacturing plant is to be in the north end of the recently completed large addition to the

Bausch & Lomb factory in Rochester. George N. Saegmuller, at Washington, D. C., has been well and favorably known since 1890, when he succeeded Fauth & Co., old established makers of engineering and astronomical instruments. Many observatories in this country and abroad have been equipped by Mr. Saegmuller. One of the specialties of the firm has been the production of accurate graduations. The two automating dividing engines which are just being erected in the fireproof vault at the new Bausch & Lomb building, and to which soon a third will be added—are among the most accurate ever constructed. Since the circles which are used on engineering instruments are graduated on the same engines, their excellence has long been recognized. Mr. Saegmuller is perhaps best known to the engineering world as the inventor of the Solar attachment which bears his name. The work of transferring the plant from Washington to Rochester has been begun. Already five carloads of machinery and instruments have been unloaded, and are being placed in the new building. Associated with Mr. Saegmuller, who will continue as heretofore in active charge, are his sons, John L. and Fred. The latter is in Europe gathering facts relative to the latest developments in scientific instruments. It is the intention to establish a scientific bureau for computation and research on the lines of the celebrated Carl Zeiss Works, of Jena, the results to be available to both the Bausch & Lomb Optical Co. and the Bausch, Lomb, Saegmuller Co.

Personal.

On June 28, Amherst College conferred upon **Henry Noel Potter**, Westinghouse special engineer, the degree of Doctor of Science, in recognition of his achievements in scientific research and his contributions to the practical as well as theoretical knowledge in the field of electric furnace work, electric lighting and kindred subjects. Dr. Potter graduated from Amherst in 1891, receiving at that time the first B. S., cum laude, given by that institution. Since that time he has been connected with the Westinghouse companies. Dr. Potter has been instrumental in the introduction of the Nernst lamp in this country. He has done a large amount of work in the electric furnace field, especially on tube furnaces. Quite recently he has developed a new process for making silicon in the electric furnace, and has devised, in conjunction with Mr. A. B. Albro, a method of using silicon in the silicothermic reaction.

Digest of U. S. Patents.

Compiled by Byrnes & Townsend, Patent Lawyers, National Union Building, Washington, D. C.

Electric Smelting and Reduction Processes

(Continued.)

No. 602,747, April 19, 1898, Charles K. Harding, Chicago, Ill.

Produces phosphorus from phosphate rock containing tricalcic phosphate. The rock is first pulverized and boiled for several hours with an excess of sulphuric acid of 20° Be., being continuously agitated. The liquid is filtered from the resulting sulphate of lime and boiled down to a syrup. The phosphoric oxide is then mixed with carbon and highly heated in a reverberatory furnace for several hours, in the presence of an excess of oxygen, thus driving off moisture and eliminating sulphur. More carbon is then preferably introduced into the furnace and parts of the oxygen are smelted out. The product, brought into a granular form by stirring and consisting of phosphoric or phosphorous oxides, or a mixture of the two, is then mixed with carbon and electrically smelted. The electric furnace has side walls of masonry, with a carbon lining at the upper portion serving as the anode, a non-conducting packing being interposed. The lower part of the lining and bottom are of silica, the bottom having a central hole through which passes the tubular carbon cathode. Around the cathode is an annular trough for the slag, leading to a tap-hole. The furnace rests upon a water-cooled metal base. During the smelting, gasoline is forced into the furnace through the cathode and is said to be electrolyzed as it enters the furnace, forming a zone of hydrogen gas around the end of the cathode, through which the current passes as an arc. The supply of gasoline is controlled by an electromagnetic valve in the main circuit. The gasoline serves as a reducing agent for the phosphorus in the slag. The vapors escaping from the furnace are suddenly expanded and cooled, being delivered to a condenser in which the phosphorus is liquified. The partial elimination of the sulphur in the roasting furnace and that of the remainder by combination with hydrogen from the gasoline, prevents the production of carbon disulphide in the electric furnace, which would otherwise dissolve the phosphorus and interfere with the reduction.

Nos. 602,975 and 602,976, April 26, 1898, Guillaume De Chalmot, Leaksville, North Carolina.

Produces the ferrosilicides Si_2Fe_2 and Si_2Fe and alloys of the two. A charge of silica or a silicate, *e. g.*, felspar or mica, mixed with iron or an iron ore and carbon, in the form of coke, coal, charcoal or pitch, is smelted in an electric furnace, the ferro-silicide being either tapped out, allowed to cool in the furnace and then separated from the slag, or allowed to settle on the bottom of the furnace, the slag being tapped off. A flux such as calcium or magnesium carbonates, hydrates or oxides, or soda and potash or other bases or their salts, may be added. With the minimum proportion of silica the product will be Si_2Fe_3. As the silica is increased, the product contains Si_2F in increasing amount, up to a product containing from 40 to 50 per cent of silicon. The materials should be pure to keep the slag low. A product low in silicon may be resmelted. A ferro-silicide containing from 10 to 13 per cent of silicon may be substituted for the iron or iron ore. An excess of the silicon compound is necessary to compensate for vaporization and incomplete reduction. The silicides are crystallized, those containing from 25 to 30 per cent of silicon being white, and those containing from 40 to 50 per cent being gray. They scratch glass, but not chrome steel, ruby or diamond. The alloys low in silicon can be melted in a combustion furnace and produce exact castings, which take a fine polish and do not tarnish in the air. The products are resistant to the action of ordinary acids and acid oxidizing agents, especially as the percentage of silicon increases, but dissolve in hydrofluoric acid. Those containing a small percentage of silicon are more resistant to aqueous alkaline solutions. The silicides are suitable for the production of anodes and chemical apparatus.

No. 656,353, Aug. 21, 1900, C. B. Jacobs, East Orange, N. J.

Produces alkaline earth-metal silicides $CaSi_2$, $BaSi_2$ and $SrSi_2$. A mixture of, *e. g.*, lime 56-60, quartz sand 120-130, and ground coke or anthracite, 60-70, is electrically smelted, preferably in the Bradley rotary furnace of patent 597,945, and the product accumulates on the hearth, where it is allowed to solidify. The silicides are white or bluish white substances of metallic appearance, resembling aluminium silicide and silicon. They have a crystalline fracture, like cast zinc. They oxidize slowly in the air, more rapidly when hot, yielding silicon dioxide. They decompose water, yielding hydrogen. The calcium compound decomposes more rapidly in hot water. The barium silicide decomposes in dilute acids, giving silicon hydride SiH_4, with free hydrogen. The calcium silicide reacts on dilute acids to give silico-acetylene, Si_2H_2, a bright yellow crystalline product. The strontium silicide reacts on dilute acids with the production of silicon hydride, free hydrogen and silico-acetylene.

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New York, August, 1905.

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Ten Years of Electrochemical Activity at Niagara Falls.

On Aug. 26, 1895, the Niagara Falls works of the Pittsburg Reduction Company started operation. On Oct. 19 of the same year the current was turned on at the plant of the Carborundum Co. Thus it is now just ten years that electrochemical activity started at Niagara, and in this short space of time more than a dozen varied electrochemical industries have grown up and are flourishing within a radius of 2 miles from the Falls. In the first two issues of our first volume Dr. J. W. Richards gave an extensive review of the electrochemical industries of Niagara, and later developments have been recorded from time to time in these columns, especially in a paper by Mr. F. A. J. FitzGerald in our last issue. It must, therefore, suffice to give here a brief summary of the principal products. In electric furnaces are made at Niagara artificial graphite, siloxicon, silicon, carborundum, alundum, calcium carbide, phosphorus and various ferro-alloys. By electrolysis of fused electrolytes are made aluminium, sodium (for the production of various important derivatives), and caustic soda and chlorine. The latter two products are also made by electrolysis of aqueous solutions; while other products of such processes are caustic potash and hydrochloric acid and chlorates. The only example of a process using electric discharges through gases on a commercial scale at Niagara is the production of ozone for the manufacture of vanillin. As the latest developments we may mention that ground has recently been broken for the caustic soda and chlorine works, which will use the Townsend diaphragm cell, while in the old barn in which Mr. Rossi made his pioneer experiments on ferrotitanium, Mr. Ruthenburg's process of agglomerating in the electric furnace magnetic iron concentrates, now undergoes an experimental trial. This is certainly a splendid development, and while as a necessary concomitant to the success achieved there have been commercial failures, yet their number is remarkably small. The most notable one is probably that of the Atmospheric Products Co., on the success of which very great hopes had been founded. When the company ceased operation, we understood from a reliable source that the process was in a "relatively" better shape than ever before, but presumably not in such a shape as to warrant the expenditure of further capital. In the interest of the problem it is greatly to be regretted that no exact information on the causes of the failure has been published. Certainly, the problem itself cannot be considered to be dead forever.

The foundation of the success of Niagara Falls as a center of electrochemical industries is the cheap power and the geographical location. The cost of power is far less than in any steam-generated plant; nevertheless, it is not low enough to warrant, for instance, the manufacture of ferrosilicon at Niagara in competition with some European producers. The advantages of the geographical location of Niagara are manifold: Its location on the frontier of the United States and Canada, its easy accessibility by railways as well as by the waterway over the great Lakes, with their 3600 miles of shore line and navigable to the very docks at Niagara. Concerning the raw materials of the electrochemical industries of Niagara an interesting article by Mr. M. M. Green will be found in our present issue. But all these natural advantages and resources would not have been exploited without the ingenuity and fostering care of American electrochemists and engineers, who here proved that good engineers can be good business men. After all, in the wide sphere of industry it is the man who does things who counts more than he who knows things only. To push forward toward a fixed goal is specially promising in electrochemistry, and Mr. Acheson's single original experiment on the reaction between carbon and silica led, step by step, to the industries of carborundum, graphite, siloxicon, silicon. May the future of Niagara's electrochemical industries be as bright as the success of the first decennium.

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Data for Metallurgical Calculations.

For his presidential address before the Society of Chemical Industry, Dr. William H. Nichols selected a subject which is of the most fundamental importance in modern industrial life, and which Dr. Nichols, as the foremost captain of chemical industry in the United States, is qualified to discuss as nobody else. This subject is the management of a chemical industrial organization, and an abstract of this address will be found on another page of this issue. While we hope to make some remarks on Dr. Nichols' address as a whole in our next issue, we wish to call attention in this note to one special point. This is the Statistical Department, which Dr. Nichols considers as absolutely necessary to intelligently run a large enterprise. This department is organized to compile and classify all the facts concerning the operation of the works; it advises the officers of the company each month of the cost of every product and step, and also of the profit or loss on each article and the total profit or loss of the company. These records represent an amount of information the value of which cannot be overestimated. With respect to this point it would seem that the metallurgical and chemical engineer has still very much to learn from the business man.

It is impossible to think of a business man who does not keep books, and who does not regularly make up his balance sheet of profit or loss. In metallurgical or chemical processes the energy balance sheet will necessarily always show a loss; for even if we had a perfectly ideal process, the principle of the conservation of energy requires that no more energy can be obtained from a process than has been put into it. Now, there will always be losses. But in this case it is just as important as for the business man to know exactly the efficiency of the process, not only of the whole process, but of each separate step and to locate the losses of energy. This is possible only by making out a complete energy balance sheet. No business man would be satisfied with information to the effect that business is good or pretty good; he wants exact figures in dollars and cents. No metallurgical or chemical engineer should be satisfied before he can state exactly in figures what becomes of the calories or of the kilowatt-hours which he puts into his process. Nobody will say that this has been done in the past by engineers to anything like the same extent as is the rule of the business man. But this is not wholly the fault of the engineer.

The making out of a metallurgical or chemical energy balance sheet requires the knowledge of principles of physics and chemistry—these principles can be acquired, and we have been glad to learn from a good many of our readers how valuable and useful in this respect they consider the serial on metallurgical calculations by Dr. J. W. Richards, which is now being published in our columns. But to make out the energy balance sheet far more is required than mere principles of calculation. We need exact figures of specific heats, latent heats of fusion and volatilization, etc., and with respect to them, as a matter of fact, there is a woful lack of data. In many cases—important cases of chemical and metallurgical practice—the most fundamental figures are lacking which would be required for the calculation of the balance sheet. Interest in these matters has now been aroused, and this should go a far way towards getting the necessary data. But people should not expect to get those important and valuable data for nothing. It is usual to expect the best investigators and brightest experimenters to do this work for nothing, to pay the expenses of the work themselves, and then give the information to the world—all for glory. Undoubtedly investigators are willing to give their time and efforts, but they are usually unable to pay the expenses. They should, therefore, be assisted. The Carnegie Institution is now doing splendid work along these lines. So are many universities, by paying the expenses from special funds. So are many broadly-run industrial plants. But in the latter case there is still too much secrecy and not sufficient recognition of the fact that the self-interest of our manufacturers should animate them to give to the world, in the spirit of "do ut des," data of this kind obtained in their research laboratories. If every one will do his duty to the profession, then we will soon be able to collect the data which one would expect to get from a Statistical Department of chemical and metallurgical industries.

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Sodium Sulphate Electrolysis.

The great success of electrochemistry in the industrial electrolysis of sodium chloride is well known and undisputed. Also well known is the commercial difficulty, due to the production of caustic soda and bleaching powder in equivalent proportions as long as all the chlorine is used for making bleaching powder, while the commercial demand for bleaching powder does not warrant this. Compared with sodium chloride electrolysis very little attention has been paid in the past to sodium sulphate electrolysis. The obvious reason is the difference in the available quantities of the raw materials. Nevertheless, there are sufficiently large deposits of sodium sulphate which call for exploitation; and an indication of developments in this direction is an interesting patent granted to Dr. Kelly which is noticed in our Analysis of Current Electrochemical Patents in this issue. A comparison of sodium chloride and sulphate electrolysis suggests itself, although not much more can be attempted here than a discussion of the general aspects of the problem, especially as long as nothing of a definite nature is known concerning the purity of the sodium sulphate deposits; to prepare and maintain cheaply a pure electrolyte during operation is, of course, just as important in this case as in any other electrolytic process.

The cathodic reaction is the same in the electrolysis of sodium chloride and sodium sulphate; it is the production of caustic soda and evolution of hydrogen. As anodic reaction we have in the case of sodium chloride the evolution of chlorine gas which is to be worked up into further products. In the case of sulphate electrolysis, we have the production of sulphuric acid with the simultaneous evolution of oxygen gas. To compare the two cases we will assume that the same amount of caustic soda is to be produced. Then for the production of 2NaOH there are required in one case $1Na_2SO_4 + 3H_2O$, and in the other case $2NaCl + 2H_2O$ as starting materials. The products of the process are, besides the caustic, in the sulphate electrolysis $1H_2SO_4 + 2H + O$, and in the chloride electrolysis $2Cl + 2H$. In the latter case, by far most of the chlorine is still being worked up in practice into bleaching powder, while for the reason already mentioned there exists a distinct tendency to use the chlorine for better paying products. The only case of a commercial plant in which the anodic chlorine and the cathodic hydrogen are combined for the production of hydrochloric acid is that of the Roberts Chemical Co., of Niagara Falls, although in this case potassium chloride is the electrolyte. Electrolysis of sodium sulphate has the undisputable commercial advantage of yielding sulphuric acid for which there is absolutely no trouble to get a market. Besides, hydrogen and oxygen are produced in equivalent proportions; whether it would pay to compress, store and sell them for the uses of the oxy-hydrogen flame is a question which would depend mostly on the geographical position of the plant.

So far we have supposed that electrolysis of sodium sulphate offers no greater engineering difficulties than electrolysis of sodium chloride. The main point in both problems is, of course, to prevent the anodic and cathodic products from reaction with each other. Dr. Kelly, in his patent mentioned above, tries to solve this problem in a way which, as far as we know, has not been used in commercial sodium chloride electrolysis. He employes a three-compartment cell. The middle compartment is to contain the undecomposed solution of sodium sulphate, while the anolyte and catholyte are the resulting solutions of sulphuric acid and sodium hydroxide respectively. It is very doubtful whether diaphragms are alone sufficient without other special devices for removing the anodic and cathodic products from the electrolytic sphere of action. The reason is that these anodic and cathodic solutions formed by the action of the current will at once begin to act themselves as electrolytes. There will, therefore, be a traveling of hydrogen ions from the anolyte into the middle department, and they will meet there SO_4 ions traveling in the opposite direction, thus forming sulphuric acid in the central part. On the other hand, from the catholyte OH ions will travel into the middle compartment, and will there meet sodium ions coming toward them, which results in the formation in sodium hydroxide. Thus both sulphuric acid and sodium hydroxide will be formed in the middle compartment. The circulation of the electrolyte will, of course, result in neutralization with the production of heat; this represents a loss of energy and an decreased efficiency of the process.

Moreover, the probability is small that equivalent quantities of sulphuric acid and sodium hydroxide are formed in the central compartment by the process just sketched. The proportion of the amounts of sulphuric acid and sodium hydroxide formed in the middle compartment depends essentially on the numerical values of the mobilities of the ions concerned. Without going into details we may say that calculation shows a slight tendency of sulphuric acid accumulating in the middle compartment, because a greater amount of sulphuric acid is formed in this department than sodium hydroxide. While any amount which is formed of the latter will be taken care of by being neutralized by an equivalent quantity of sulphuric acid, yet there will probably be left a balance of sulphuric acid in the central compartment, and this amount will increase during the operation. If this is so, the central compartment will get richer and richer in sulphuric acid and the main purpose of maintaining a neutral compartment will not have been accomplished. However, there is another action, that is, endosmosis, and almost nothing can be predicted *a priori* as to what it will do, especially since it depends on the nature of the diaphragms. This effect may be greater than that of ionic migration.

For sodium chloride electrolysis we have an abundant number of different means of removing the cathodic products of the current from the sphere of action. To mention only a few, in the Castner-Kellner process the sodium alloys with mercury, and is carried away by movement of the latter. In the Acker process, with its molten electrolyte, the sodium is absorbed by lead. Mr. Townsend's new diaphragm cell represents an ingenious method of carrying off by simple mechanical means the sodium hydroxide as soon as formed. Any of these or any other suitable means may, of course, also be applied at the cathode in sodium sulphate electrolysis. But at the anode we have a somewhat different problem. The anodic product, in case of an inert electrode, is sulphuric acid. This we may remove mechanically; or if we do not want its formation in the cell itself, we can catch the SO_4 ion by means of an anode forming an insoluble sulphate, like lead sulphate. In this case the sulphuric acid would be obtained afterwards. There can be no doubt that the problem can be solved if there is enough cheap and suitable raw material available to warrant the trouble of its solution. For the present it would be interesting to know what results have been obtained with Dr. Kelly's cell in practice.

Autumn Meeting American Electrochemical Society.

The eighth general meeting of the American Electrochemical Society is to be held at Bethlehem, Pa., on September 18, 19 and 20 (Monday, Tuesday and Wednesday), as was already announced in these columns.

Earlier than ever the preliminary programme has been issued, and while it not yet contains a list of the papers to be presented, yet in its general arrangement as to sessions, social functions, visits to plants and excursions, the programme is unusually elaborate and attractive.

While Monday is often considered a bad day for beginning a meeting, in the present case the programme has been so arranged that members in New York or Philadelphia may attend to their mail on Monday morning and leave New York on the Black Diamond Express at 11.55, or Philadelphia at 12.30, and arrive in time for the first session at 3.00 p. m.

Professional Sessions.—There will be three sessions for the reading and discussion of papers. On Monday at 3.00 p. m., on Tuesday at 9.00 a. m., and on Wednesday at 9.00 a. m These three sessions will be held in the lecture room of the Physical Laboratory of Lehigh University, in South Bethlehem.

Visits to Plants and Excursions.—These should prove extremely interesting and profitable to all members. On Tuesday afternoon the plant of the Lehigh Zinc Co., in South Bethlehem, will first be visited, which is one of the oldest and one of the most interesting zinc works in the United States. The Bethlehem Steel Co.'s world-renowned works will next be visited. For Wednesday afternoon an option of either of two excursions is offered. The first is by trolley to one of the big cement works in the vicinity, the other an excursion to Mauch Chunk, including a trip around the Switch-Back. Any who remain in Bethlehem Wednesday afternoon are cordially invited to attend the opening exercises of Lehigh University.

Social Functions.—A smoker will be held at one of the commodious club houses in Bethlehem on Monday evening. A lunch will be tendered to the visiting members by the authorities of Lehigh University, in the gymnasium, on Tuesday, September 19, at 12.30 p. m.

A subscription dinner will be served at the Eagle Hotel, Tuesday evening, September 19. The Eagle Hotel is to be the convention headquarters.

Entertainments for Ladies.—Besides the attractions which the beautifully situated historic town of Bethlehem offers in itself, a special programme of entertainment is provided for the visiting ladies. For Monday afternoon a carriage drive, starting from the Eagle Hotel at 3.00 p. m., has been arranged through the Bethlehems to South Mountain, with a visit to the grounds and buildings of Lehigh University. In the evening there will be an informal reception by the ladies' committee to the visiting ladies and their escorts at the residence of Professor and Mrs. J. W. Richards. On Tuesday morning an excursion will be made to Easton, with a visit to the large works of the Standard Silk Co.; lunch will be taken at Huntington, followed by a visit to Wygatt Mountain and grounds and buildings of Lafayette College, while in the evening a subscription dinner will follow, as already mentioned. For Wednesday morning an excursion is planned to Nazareth, with visits to the quaint Moravian buildings, Nazareth Hall, school, Indian burying ground and the Whitefield house. For the afternoon the option of either an excursion to Mauch Chunk or the attendance of the opening exercises of Lehigh University is offered.

Dr. Joseph W. Richards, who has just sailed for Europe, is the chairman of the local committee; the secretary is Mr. Walter S. Landis, from whom any further information may be obtained.

The Management of a Chemical Industrial Organization

The official feature of the visit of American members of the Society of Chemical Industry to England was the annual meeting, held on July 10, in the University College, London, where Dr. Wm. H. Nichols delivered his presidential address, an abstract of which is given below. Concerning the trip through England and its social features and visits of plants—which we are informed is turning out the big success which had been expected—we will report when it is over, in our next issue. The number of members is now 4326, compared with 4134 at the last annual meeting; these members live in sixty-four different countries, though the great divisions, the British Isles, United States, Australia, Canada, etc., are only reckoned as one each. The new president is Dr. E. Divers.

William H. Nichols.

The subject of Dr. **William H. Nichols'** extremely interesting and suggestive presidential address was "The Management of a Chemical Industrial Organization." Sir Henry Roscoe, later on, in proposing a vote of thanks to Dr. Nichols, characterized his address as "a lesson from America—a wonderful lesson, a most valuable lesson."

Dr. Nichols' plan of management is applicable in a large and rapidly growing country like the United States. The organization is divided into the following general departments: Purchasing, Sales, Transportation, Finance, Construction, Operating, Research (or Investigation) and Statistical; and yet all of these must be so closely interwoven and work in such harmony that best results shall be obtained without delay. To attain this object, Dr. Nichols has found it necessary to arrange two committees, composed as follows: First, a manufacturing committee, consisting of the managers of the Operating, Construction, Purchasing and Investigation Departments, the chairman being the chairman of the executive committee of the board of directors. This brings all departments having to do with the turning out of products, present and prospective, in regular and systematic touch with each other and with the executive committee. Second, a sales committee, composed of the managers of the Sales, Operating and Purchasing Departments, together with a member of the executive committee. This also results in a close touch of the department distributing products with the others, and with the active governing body.

Concerning sales, Dr. Nichols expressed his firm belief that if the goods are well and economically produced—but not overproduced—at the right geographical points, their sale follows almost as a matter of course. "I do not pin my faith very strongly to that method of doing business, by means of which manufacturers by agreement fix prices without much reference to cost, and thus court the competition which is sure to ensue. I believe that the best results in the long run are obtained by following natural laws willingly, rather than by combating them, invite the disaster which is certain to follow the temporary benefit so derived."

The main part of Dr. Nichols' address dealt with the organization of the Operating, Construction, Investigation and Statistic Departments.

Operating Department.

The Operating Department is one of great complexity and importance, involving, as it does, all the details of the management of the various works in connection with their corps of superintendents and chemists and the foremen, processmen and laborers under them. Its headquarters should be in the main office of the company, in close touch with the officers. A very complete and systematic organization and hearty coöperation with the other departments is necessary.

The administration of this department is carried on by a manager and assistant manager with a suitable clerical force. Through membership on committees and by frequent consultation with department heads and by correspondence with branch managers, the manager and assistant manager of the department are kept informed regarding the general policy and manufacturing needs of the company, and regulate manufacturing so far as possible in such manner as to provide a sufficiency of products wherever required.

This department naturally requires the services of a number of chemists, used for the daily routine of the work. The selection of these men is highly important, as from their number may come those who will go into the research laboratory of the company, or possibly fill its superintendents' positions, and in later years its offices. It is not sufficient that they have received a good education and have completed a technical course, but they must evidence such personal characteristics as lead to the belief that they are capable of advancement to positions requiring the exercise of thought and judgment, and the assumption of responsibility. It is desirable that they should have some knowledge of mechanical engineering and the general principles of construction.

The apprentice confines his duties for a number of months, and frequently for years, to a works laboratory, and incidental to his analytical work he gains a certain knowledge of the general routine which obtains at that plant. If he here displays ability, he is promoted to a position which will bring him into direct contact with the manufacturing process, and his duties will gradually change from those of analyst to those of a manufacturing assistant. Further promotions lead to assistant works superintendent and superintendent. The chemist must have a clear, logical mind, a singleness of purpose, and he must be able to separate the essential from the non-essential. This is very important.

Construction Department.

The Construction Department has entire charge of the designing and construction of all new plants, or parts of plants, and must act in the closest touch with the Investigation Department. The course of procedure was sketched in Dr. Nichols' address.

Investigation Department.

The Investigation Department deals with all the new propositions of a technical nature. A new proposition is in its control until sufficient data has been obtained to enable the Construction Department to design the necessary plant, if one be authorized by the executive committee. The organization of the Investigation Department should be sufficiently broad to permit the consideration of a manufacturing proposition from the points of view of the business man, the chemist, the engineer and the patent attorney. It consists of the manager, a chemical council, composed of the manager, the chief chemical engineer, the chief chemist, who is director of the Research Laboratory, and such consulting chemists and engineers as the company employs. A corps of chemists on research laboratory work, an abstractor of current chemical literature, patent experts and a small office force complete the department staff.

The work of the Investigation Department originates from sources which may be divided into three classes: (*a*) The probability of reducing manufacturing costs; (*b*) a decision to produce well established products not previously manufactured by the company; (*c*) new applications of science to industry.

The commercial side of a new proposition calls for consideration of the following points: Its relation to the interest of the company, the market, manufacturing costs, investment necessary, source of raw materials, transportation.

On the technical side a study must be made of the process, other processes, raw materials, quality of product required.

The Research Department would not be complete without a laboratory plant, large enough to work out processes on a small manufacturing scale.

Statistical Department.

The Statistical Department, which is absolutely essential in a company operating a number of plants, has to do with the compilation of facts and the deductions from them. It advises the officers within a reasonable time after the end of each month of the cost of every product and step, and also of the profit and loss on each article and the total profit or loss of the company. Dr. Nichols stated that the results compiled by the Statistical Department of his own company have been so exact that for several years the profits determined by public accountants at the end of the year have not varied 1 per cent from those which had been worked up in the Statistical Department month by month.

Copper Refining at the Kedabeg Copper Mines.

A long paper, read by Mr. Gustav Köller, before the Institution of Mining and Metallurgy in London, on March 16, 1905, gives an interesting description of this Russian electrified refinery.

The installation for the electrolytic process is situated in Kalakent. It was established in 1889 on a small scale and comprised originally only 36 electric baths, from which an output of 216,000 pounds of electrolytic copper was obtained annually from 90 per cent. raw copper. The slimes deposited at the anodes were low in precious metal, the maximum tenour never exceeding 2 per cent. of argentiferous gold. The process using raw copper was subsequently abandoned, and refined copper was substituted, the capacity of the works being also increased in 1893 to an annual production of 900,000 to 1,080,000 pounds of pure copper.

The plant now consists of 102 electric baths, which are supplied with current from a shunt-wound dynamo generating 700 amperes at 35 volts. These baths are grouped in four rows, and the arrangement in general differs from that usually adopted in other places only in that the baths are entirely independent of each other with regard to the circulation of the liquors. The circulation is effected by means of Siemens' geyser-pumps, two pumps being attached to each bath. This arrangement enables the baths to be worked either independently or in groups, and facilitates the control of each separate bath. They are made of pine wood, the length inside being 2 m., the depth 1.15 m. and the width 0.96 m. The joints are rendered tight with canvas lining, soaked in asphalt, a method which cannot be highly recommended, but which was adopted on account of the enormous price of lead in Russia.

In preparing the canvas lining, there are several precautions to observe, especially with regard to the quality of the asphalt or creosote. If the latter is insufficiently boiled before being applied, gases consisting of oxygen and carbon dioxide are attacked by the action of the free acid in the bath, and the cathode copper becomes oxidized by these.

The connectors which convey the current along the bath in the direction of its length have a cross section of 500 square mm., and the main conductors are slightly larger. The baths are charged with 15 anode and 14 cathode plates, beginning and closing with anodes. The electrodes in each bath are, of course, fixed in parallel, while the baths themselves, as whole units, are connected in series. The distance between the electrodes is 5 to 6 cc. The effective area of the anodes is 19 to 20 square m., and that of the cathodes 19 square m.

The anodes are of cast refined copper, containing 0.07 to 0.09 per cent. of precious metal, of which the 1-14th part is gold. The weight is 288 to 360 pounds, and they measure 90 by 80 cm., with a thickness of 2 to 2½ cm. The cathode sheets are 2 mm. thick, and before using are heated to redness to get rid of grease, and are provided with suspension hooks and metal strips to serve as terminals.

As already stated, two geyser-pumps are connected to each bath for circulating the electrolyte, an operation which must be carried on vigorously to ensure uniform deposition of the copper. The liquor is raised by atmospheric pressure, equal to about 70 mm. in the mercury column, through the delivery pipe from the neighborhood of the bottom of the bath and flows laterally into a distributing channel to mix again with the electrolyte at the surface. In this manner 800 litres of the liquor are circulated per hour in each of the baths.

The current density does not exceed 25 or 30 amperes per square m. of the cathode surface, the electromotive force across the baths being 0.18 to 0.25 volt. The current efficiency has been found to amount to 99 per cent.

The electrolyte is maintained at 75 grm. free H_2SO_4 per litre and the copper at 35 grm. per litre.

The purification of the electrolyte proceeds along orthodox lines by saturating with copper from calcined matte filtration through lean ore, evaporating with excess of anode scrap aided by air blows. The excess of copper brought into the liquor is crystalized out after evaporation.

The finished electrolytic copper has a degree of purity of at least 99.9 per cent., the average of the analyses giving 99.93 per cent. The weight of the finished plates is about 300 pounds.

The slimes are collected from the baths towards the end of the year, and by decanting with water are freed from the residual liquor, and are then screened and washed. In this condition they contain 35 to 40 per cent. of copper, of which amount 25 per cent. to 30 per cent. is extracted by heating 50° or 55° C., and treating with exhausted acid liquor. After washing again and drying they still retain 10 to 15 per cent of copper, and from 25 to 30 per cent. of precious metal, and in this form they are sold, without further preparation, to the North German Refinery at Hamburg. In the year 1899 there were produced 863,136 pounds of electrolytic copper, and 306,612 pounds of anode slimes containing 722 pounds of silver and 55 pounds of gold.

CORRESPONDENCE.

Electric Furnace Experiments.

To the Editor of Electrochemical and Metallurgical Industry:

SIR: I beg to call attention to one point in Mr. Francis A. J. FitzGerald's very interesting article on "Materials for Resistors," published in the December number of 1904 of your journal. In referring to the curve in Fig. 4, on page 493, he says: "Since the voltage at the terminals of the resistor was constant, the curve represents the changes in resistance of the resistor as a whole."

But the figures that he gives in the table on the same page show that the voltage at the terminals of the resistor was not constant, unless the resistance of the two carbon terminals was at least .2 ohm, which could not possibly have been the case with a furnace as shown in the diagram in Fig. 3 on page 492. According to the diagram the resistance of T1, T2 and M would be negligible, and the resistance of the resistor as a whole would be the sum of the resistance of G1 and G2. Now, the table on page 493 shows that the sum of the two voltages at G1 and G2 was not a constant, but in one case was 72 and in another as high as 84 volts.

This, of course, affects the shape of the resistance curve in Fig. 4, and an examination of the figures will show that the effect of making this correction would be to bring the points E and G lower down and B higher up; in other words, to simply accentuate the difference in the resistance caused by the weights, making the maximum total change in resistance a change of 55 per cent instead of 50 per cent, as the curve in Fig. 4 indicates.

Chicago, Ill. **JAMES J. DOYLE.**

To the Editor of Electrochemical and Metallurgical Industry:

SIR: Mr. Doyle, in making his criticism, has overlooked the fact that the volts given in the table under G1 and G2 were not measured between T1 and M and T2 respectively, but between the rods A and B and the rods C and D. Therefore, the sum of the volts given under G1 and G2 at any time is the voltage between T1 and T2.

Although Mr. Doyle is mistaken in his criticism, there is an error in Fig. 4, for the Y-axis ought to be designated "reciprocals of current," or "resistance," instead of "ohms," and the upper figure marked on the axis should be 0.015, while the figure at the origin should be simply 0.

Niagara Falls, N. Y. **F. A. J. FITZGERALD.**

Crystalline Structure of Electrolytic Copper.

To the Editor of Electrochemical and Metallurgical Industry:

SIR: The letter of Mr. Addicks, published in your issue of July, with photograph of crystalline electrolytic copper, recalls to my memory some experience I had many years ago, in 1883, while precipitating impure copper solutions obtained from the treatment of ores, when crystals similar to those described by your correspondent were obtained in the iron plates used for cementation.

The cold copper solutions were treated in vats filled with iron plates, chiefly cast iron; in one of the vats all the cement copper was deposited in crystals, instead of the usual form of cascara, and exactly in a fern-like grouping. The following observations were made as to the precipitation.

The copper deposited was more abundant in the edges than in the faces of the iron plates.

When two iron plates were placed in the copper solution facing each other, the copper deposit was more regular and adherent in the large plate, but more abundant, and forming crystalline groupings in the small plate.

In two iron plates placed, one showing its edge to the other face, the deposit was more abundant in the plate placed edgewise. The deposit was crystalline and the larger part of it toward the edges.

In some of the iron plates the deposit was very irregular; the copper being deposited in crystalline parallel lines.

Some of the plates from old scrap had holes drilled in; the crystals were chiefly deposited in the edges of the holes, sometimes in the shape of spherical groupings.

When the solutions contained a considerable quantity of iron salts, most of the deposit was in crystals, some very large.

The high density of the current originated by the action of the solution on the elements forming the cast-iron plates, may have caused this crystalline deposition. This opinion was sustained by Bucholz in his classical experiment many years ago.

New York City. **J. BAXERES DE ALZUGARAY.**

Heat Insulating Qualities of Refractory Materials.

The eminent importance of efficient heat insulation for the design of furnaces has been the subject of various discussions in our journal. Mr. F. A. J. FitzGerald has shown repeatedly how great a saving of electrical energy may be accomplished by paying due attention to this point in the design of electric furnaces (our Vol. II., pp. 345, 439, etc.). A discussion of various refractory materials by Mr. E. K. Scott may be found in our Vol. III., p. 140. Mr. W. McA. Johnson recently (Vol. III., p. 214) discussed some principles of the use of refractory materials in general metallurgy, and pointed out how little exact knowledge we have concerning the heat-insulating quality of our ordinary refractories. A remarkable indication of the backward state of our knowledge of refractories is the fact that we use fire-clay generally as material for heat insulation as well as for transferring the heat from the fuel to the charge. This is, of course, really nonsensical. What is most thoroughly needed is exact data on the heat-insulating qualities of refractory materials.

With respect to this problem, a recent Faraday Society paper, by Messrs. R. S. HUTTON and J. R. BEARD, contains much useful information. The authors point out that the measurement of thermal conductivities of refractories is by no means difficult to carry out, at any rate to within an accuracy of 2 or 3 per cent, by means of a method and apparatus devised by C. H. Lees and J. D. Chorlton (*Philosophical Magazine*, June, 1896 [5], Vol. XLI., p. 495 to 503). Messrs. Hutton and Beard have used this apparatus in their tests, carried out in the Physical Laboratories of Manchester University.

All the substances dealt with were either in the state of well-defined granular powders, or, in some few cases, where bricks were examined, these were previously broken up and finely powdered. The apparatus, as shown in Fig. 1, consists essentially of two thick brass discs, A and B, between which the material is placed whose thermal conductivity is to be measured. The upper disc B is surmounted with a steam jacket S which is covered with felt, to diminish the loss of heat. The lower disc A is allowed to cool by free conduction and radiation. This disc is suspended from a horizontal support by three strings. Both discs have the same dimensions, the diameter being 11.4 cm. and the thickness 1.3 cm. The powders are kept in position by a thin ring of red fiber, about 0.36 cm. high. A and B are provided with projecting pegs, the distance apart of which gives a measure of the thickness of the insulating material.

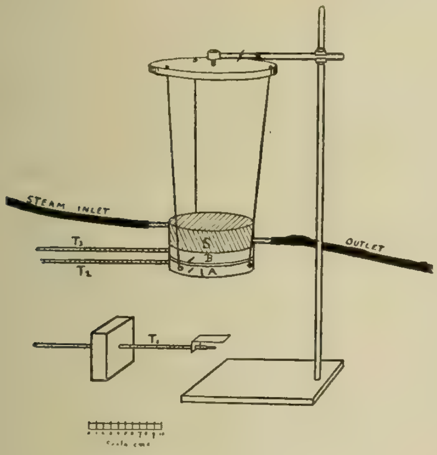

FIG. 1.—MEASUREMENT OF HEAT INSULATION.

Three thermometers are used. T_1 registers the temperature of the air; it is provided with a screen to protect it from radiated heat from the lower disc. T_2 and T_3 indicate the temperature of the lower and upper disc respectively, and are inserted into radial holes drilled in the brass. When steam is passed through the steam jacket the temperature of the upper disc is raised to nearly 100° C., heat flows through the material experimented on, and thus raises the temperature of the lower disc above that of the surrounding air. The lower disc loses heat by conduction and radiation to the air. Eventually a steady state is reached when this loss of heat is equal to the heat received through the material experimented on.

The following equation enables the thermal conductivity K to be calculated from the result of such an experiment, provided h_1, the external conductivity or emissivity, has been previously determined (from the rate of cooling of the lower disc at different temperatures):

$$K = \frac{t_2 - t_1}{t_3 - t_2} b \left(h_1 + h'_1 + \frac{p h_1 b}{2q} \right)$$

Here t_1 t_2 t_3 are the temperatures indicated by the thermometers T_1 T_2 T_3 respectively, and b denotes the thickness of the layer of powder in cm.; p is the perimeter and q the area of cross-section of the plate. Under the conditions of the experiment (*i. e.*, with the dimensions of the apparatus mentioned above) $h'_1 = 0.026 \sqrt{\frac{p h_1 k_1}{q}}$, where k_1 is the internal conductivity of the metal disc (0.25 for brass). The conductivity k is the quantity of heat in gram-calories, which is transmitted per second through a plate 1 cm. thick per square centimeter of its surface, when the difference of temperature between the two faces of the plate is 1° C. According to the arrangement of the experiments, the conductivities refer to the temperature range from 20 to 100° C.

Messrs. Hutton and Beard found by this method the following conductivities, most of the granular powders tested just passed through a sieve with 600 meshes per sq. cm:

Sand, white Calais	K = 0.00060
Carborundum, fine	0.00050
Carborundum, coarse	0.00051
Quartz "enamel"	0.00036
Quartz, fused	0.00039
Firebrick	0.00028
Retort graphite	0.00040
Lime	0.00029
Magnesia, fused	0.00047
" "Mabor" brick	0.00050
" calcined Greek	0.00045
" calcined "Veitsch"	0.00034
" Pattinson's light calcined	0.00016
Kieselguhr (infusorial earth)	0.00013

RELATIVE VALUE OF DIFFERENT HEAT INSULATING MATERIALS AT HIGH TEMPERATURES.

Messrs. Hutton and Beard point out that it is not advisable to choose a material suitable for furnace uses from the above data alone.

Firstly, it is always necessary to know the general effect upon the substance under investigation of such temperatures as are likely to be experienced. Several, even amongst those materials mentioned in the above table, are unsuitable for subjecting to high temperature on account of the physical changes, such as shrinkage, which they undergo. Chemical action, especially oxidation, also renders many insulators unsuitable for a large number of purposes.

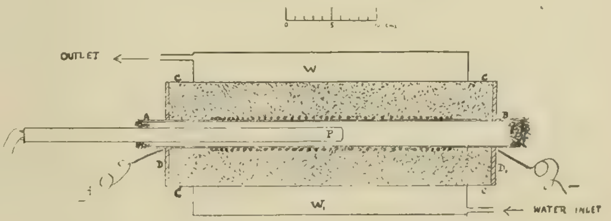

FIG. 2.—ELECTRIC FURNACE FOR TESTING REFRACTORIES.

Secondly, having by such considerations excluded all unsuitable materials from the choice, it becomes necessary to compare the relative efficiency of those which remain under conditions more nearly approaching those experienced in practice. Messrs. Hutton and Beard have made the following test:

An electric tube furnace, as shown in Fig. 2, was employed. A B is an unglazed porcelain tube with a heating coil of nickel wire wound on its central portion. The tube is kept in position at the center of the cylinder of sheet iron CCCC by washers of asbestos card DD_1. The cylinder C is surrounded by the water jacket WW_1. The temperature in the tube is indicated by the thermoelectric pyrometer P. The granular resistance material under test fills the space between the porcelain tube and sheet-iron cylinder.

The electric energy expended in the heating wire of nickel is kept constant over a considerable period by careful adjustment of an external resistance and constant observation of a precision wattmeter. The temperature at the center of the porcelain tube is noted at regular intervals, and from the observations a curve is plotted to show the relation between the rise of temperature and the time during which the heating has been in progress. The temperature of the water flowing through the external jacket was kept fairly constant; generally it varied from 2° C.

In such an experiment it would require a very long time for the temperature to attain a constant value. It was, therefore, found preferable, after the rate of increase in temperature had become rather slow, to reduce the power by a small but definite amount. Under these conditions the temperature falls, at first very rapidly, but soon reaches a constant value, or at any rate, falls so slowly that the variation is almost negligible. It is from a comparison of the heating curves with different materials and the same expenditure of power, and particularly from the nearly horizontal portion of the curves with lower power expenditure that valuable conclusions can be drawn.

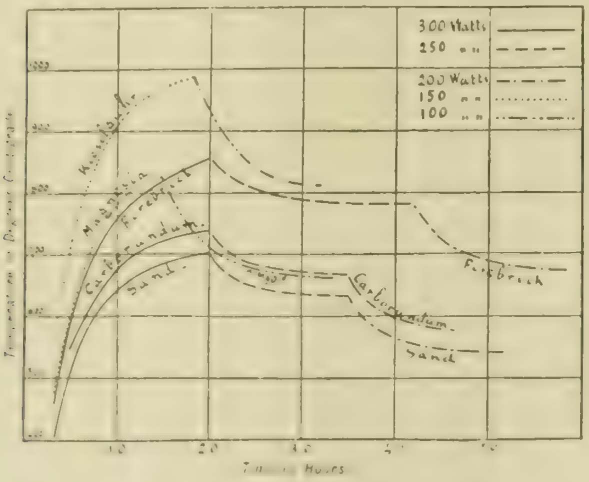

FIG. 3. TESTS OF VARIOUS REFRACTORIES.

Fig. 3 illustrates some of the results obtained by Messrs Hutton and Beard:

In the case of firebrick, carborundum and sand, the preliminary heating was effected with 300 watts, and after the lapse of about two hours the power was reduced to 250 watts, being kept constant at this for a period of one and a half to two hours, by which time the temperature had become almost constant. The curves for kieselguhr or infusorial earth (see FitzGerald, our Vol. III., p. 55) and light magnesia clearly show the very much smaller conductivity of these materials. With these two substances the preliminary heating was effected with an expenditure of only 150 watts, but despite this the rise of temperature [illegible] rapid than with any of the other materials [illegible]

Unfortunately, neither light magnesia nor infusorial earth can withstand any high temperatures for long without losing their efficiency to a considerable extent. Both substances exhibit large shrinkage at high temperature, and their chief uses are likely to be limited to external jacketing of furnaces lined with some more permanent but less good insulation.

Tests were then made to see what influence the preliminary heating might have on the results. Two similar experiments were carried out with the various substances. In the first case the preliminary stage of the heating was effected with 300 watts, in the second with 250 watts. In both cases the power was subsequently reduced to 200 watts, and then to 150 watts. The results showed that in all those cases in which no permanent alteration is brought about in the materials under test, the horizontal portions of the curve are practically identical, whatever the preliminary treatment had been. In earlier experiments, in which no water cooling had been used to maintain constant the temperature of the outer jacket of the enclosure, some anomalous results were obtained.

In their conclusions Messrs. Hutton and Beard point out that while bricks and general jacketing materials for furnaces should be of low thermal conductivity, it is of no less importance to choose crucibles, retorts and other containing vessels which are heated externally of as high conductivity as is feasible in conjunction with their other necessary qualities.

By a judicious use of the different materials and a stratified form of construction, it should be possible usefully to employ such excellent insulators as infusorial earth without risk of the damage which is inevitable if they be permanently subjected to too high a temperature.

On the Fundamental Formulæ of Chemical and Electrical Energy.

By Prof. F. Haber, Ph. D.

In the development of our knowledge of the relation between heat of reaction and energy of reaction, three different stages may be clearly distinguished. The oldest view is that which assumed the energy of reaction to be equal to the reaction heat. This view has been defended by Berthelot under the name of Principe du travail maximal for a long time. Sir William Thomson (Lord Kelvin)[1] has applied it to galvanic cells by connecting the reaction heat Q of the reaction which generates electric current, with e.m.f. E of the galvanic cell and with the valence *n* of the material involved in the reaction by means of the equations

$$(1)\quad A = Q$$

and

$$A = 23{,}110\ En.$$

Where A represents the energy of the reaction, and 23,110 follows from Faraday's electrochemical equivalent and from the electrical equivalent of heat.

The second stage of the development was inaugurated by Helmholtz, who derived from the principles of thermodynamics the result that A and Q differ from each other by variable quantity according to the equation

$$(2)\qquad A = Q + T\frac{dA}{dT}$$

or

$$(3)\quad E = \frac{Q}{23{,}110 \times n} + T\frac{dE}{dT}$$

where T represents the absolute temperature, while the quantity $\frac{dE}{dT}$ or $\frac{dA}{dT}$ is called the temperature coefficient. The term $T\frac{dA}{DT}$ represents that portion of the energy of reaction which is produced at the temperature T by the change of a certain quantity of external heat q into work. By considering an experimental arrangement, which has also been realized in practice, it is easy to make these relations clear.

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[1] Helmholtz' first opinion was the same.

We place a galvanic cell, together with its circuit, into an ice-calorimeter and measure the heat change which takes place when an electric quantity of 96,540 n coulombs passes through the circuit. The heat developed is equal to Q, that is, equal to the reaction heat of the chemical change taking place in the cell. The production of electrical energy is in this case only an intermediate process, and the resultant heat is the same as though the same reaction had taken place to the same extent, in the ordinary chemical manner, by direct transformation of chemical energy into heat at zero degree C. without any generation of electric current.

We now modify this experiment by using two ice-calorimeters instead of one. One calorimeter contains the cell, the other one the external circuit, the resistance of which is made extremely large compared with the internal resistance of the cell; 96,540n coulombs may again pass through the circuit. Now, several different cases are possible. The first case is that in the second calorimeter the heat Q is set free, while the first calorimeter, which contains the cell, shows no heat change. Then we have again $A = Q$. But this is not necessarily so, since the heat set free in the second calorimeter may differ from Q by a positive or negative quantity q. If in the second calorimeter the heat $Q + q$ is set free, the law of conservation of energy requires that the same amount of heat q disappears in the first calorimeter. The galvanic cell, which is placed in this calorimeter, will then have the property of changing at constant temperature the heat q into electrical energy. In this case the electrical energy A, which is changed into heat in the second calorimeter, is

$$(4)\quad A = Q + q$$

The former formula (2) contains, instead of the calorimetrically measurable heat q, the term

$$T\frac{dA}{dT} \text{ or } T\frac{dE}{dT} \text{ in (3)}$$

The identity of q and $T\frac{dA}{dT}$ follows from the Carnot-Clapeyron principle as to the work which can be obtained from heat by reversible cyclic processes, generally called the second principle of the mechanical theory of heat.

In order to apply the formula of Helmholtz to experimental investigations, the differential quotient $\frac{dA}{dT}$ is replaced by the quotient of the differences $\frac{\Delta A}{\Delta T}$ which is obtained if A is determined at two temperatures, near together. This, for the reactions which produce electrical energy in galvanic cells, may be done in a very simple way, since, in this case, it is only necessary to measure the e.m.f. of the reversible cell at different temperatures which are near together. In this way it has been possible to confirm Helmholtz' formula in many cases.

This formula does not state, however, anything concerning the influence of a change of concentrations; but it is easy to show experimentally that any change in the concentrations of the substances used causes a change of the temperature coefficient. A deeper insight into the relation between free energy, or e.m.f. and the temperature, was then obtained by van't Hoff, who developed the formula

$$(5)\quad 23{,}110.\ En = RT\left[nK - \ln \frac{C^{v'}_{for'}\,C^{v''}_{for''}\,C^{v'''}_{for'''}\ \ldots}{C^{n'}_{dis}\,C^{n''}_{dis''}\,C^{n'''}_{dis'''}\ \ldots}\right.$$

Here C is the concentration, for', for", for''' . . . . . . represent the substances formed during the reaction, dis', dis", dis''' . . . . . represent the substances consumed or disappearing during the reaction, v', v", v''' . . . and n', n", n''' . . . . . the corresponding numbers of molecules. This formula of van't Hoff is valid only as long as the materials follow the gas law, *i. e.* for solutions only as long as they are not too concentrated. The numerous investigations which were undertaken to confirm by experiment the formulæ (5) and (2), I would consider as the second stage in the development, while the first stage is characterized by the faith in the validity of Thomson's rule (equation 1). Placing the equations 3 and 5 side by side—

$$23{,}110\ En = Q + 23{,}110\,n\ T\frac{dE}{dT}$$

$$23{,}110\ E\,n = RT\ln K - RT\ln \frac{C^{v'}_{for'}\,C^{v''}_{for''}\,C^{v'''}_{for'''}\ \ldots}{C^{u'}_{dis'}\,C^{u''}_{dis''}\,C^{u''}_{dis'''}\ \ldots} \quad (5a)$$

the question is suggested: may Q and RTlnK be interchanged? Experience shows that this is only possible in some cases. In other cases results are erroneous. The third stage of development is characterized by the attempts to elucidate by theory and experiment the relations between Q and RTlnK.

From Helmholtz' fundamental paper, it may already be seen that the difference in the specific heat of the compounds decomposed and formed rules the relation sought. Lewis and Richards in the United States, and van't Hoff in Germany, have recently brought this half-forgotten view to the front, and inaugurated the third stage of development, to the theory of which this paper may perhaps add some slight simplification.

The equation (2) of Helmholtz is a differential equation, and can be integrated. For this purpose we may write it as follows:

$$-\frac{A}{T^2} + \frac{1}{T}\frac{dA}{dT} = -\frac{Q}{T^2}$$

The left hand is now a complete differential, and we may write the whole equation at once as follows:

$$(6)\quad \frac{d\left(\frac{A}{T}\right)}{dT} = -\frac{Q}{T^2}$$

The equation (6) is exactly identical with the formula of Helmholtz, the difference being only in the manner in which the equation is written. By integrating equation (6) we get

$$(7)\quad A = a\,T - T\int\frac{Q}{T^2}\,dT$$

In equation (2) of Helmholtz the concentrations are assumed to remain constant while the temperature changes. In the integrated equation (7) the term a, therefore, depends on the concentrations. In order to calculate the integral it is now necessary to express Q as function of T. For this purpose we write

$$(8)\quad Q = Q_0 + \sigma T$$

where Q_0 is the reaction heat at the zero point of the absolute temperature scale, while σ represents the difference of the mean specific heats of the consumed and formed substances at constant volume. Here we assume with van't Hoff that it is sufficiently approximate to consider σ as constant with varying temperature. The calculation can easily be made more exact by putting

$$\sigma = \sigma' + \sigma''T + \sigma'''T^2 + \ \ldots$$

By making the above approximate assumption, and introducing the term of Q from (8) into 7, we get

$$(9)\quad A = a\,T + Q_0 - \sigma T \ln T.$$

Now, going back to 5a such concentrations may be chosen that the second term on the right hand vanishes, then

$$(10)\quad A = 23{,}110\ En = RT\ln K = a'T + Q_0 - \sigma T\ln T$$

Here a' is the special value of a corresponding to the concen-

[illegible] For any concentrations the equation becomes:

$$(11)\ A = [illegible] = Q - \sigma T \ln T - RT \ln \frac{C^{v'}_{for'}\ C^{v''}_{for''}\ C^{v'''}_{for'''}}{C^{v'}_{dis'}\ C^{v''}_{dis''}\ C^{v'''}_{dis'''}} + a'T$$

Now, T. W. Richards and van't Hoff agree in the view that a' is zero, at least when the number of molecules formed and decomposed is the same as e. g. in

$$Zn + Cu\ So_4 = Zn\ So_4 + Cu$$

and in similar cases. The equation then becomes

$$(12)\qquad RTlnK = Q - \sigma TlnT$$

The equation (12) is remarkable, since it shows the possibility of calculating the equilibrium constant K of the reaction from the reaction heat, the temperature and the difference of the specific heats of the substances formed and decomposed during the reaction. If σ is zero, the reaction heat is independent of the temperature and equal to RTlnK. The difference between the reaction heat and RTlnK on one hand and the change in the reaction heat by the temperature on the other hand, control each other, so that, by determining one, the other may be calculated.

On the assumption of Richards and van't Hoff that a' is zero, we get also from 11—

$$(13)\quad \frac{dE}{dT} = -\sigma(1 + \ln T) - R \ln \frac{C^{v'}_{for'}\ C^{v''}_{for''}\ C^{v'''}_{for'''}\ \ldots}{C^{v'}_{dis'}\ C^{v''}_{dis''}\ C^{v'''}_{dis'''}\ \ldots}$$

By help of 13 the numerous investigations of dE/dT may be used to calculate σ when the concentrations are known too.

Unfortunately, there are very few determinations of the specific heat of dilute solutions. It appears very important to systematically study this matter, and we may hope that a general endeavor to this effect will prove not less fruitful than the work done in the investigation of the effect of concentration.

The equation 11 has also a large field of application in gaseous reactions. The discussion of this application will be found in my "Thermodynamische Vorlesungen über Technische Gasreactionen."

Techn. Hochschule, Karlsruhe, Germany.

Electric Induction Furnace for Making Steel.

Mr. Viktor Engelhardt, chief chemist of the Siemens & Halske Co. in Berlin, made a visit to Sweden towards the end of last year to study and report upon the electric manufacture of steel in the induction furnace, as operated at Gysinge, Sweden, and his observations and conclusions are embodied in a voluminous report published in *Stahl und Eisen* of Feb. 1, 15 and March 1. Since the induction furnace has been so often described and noticed in these columns (Vol. I., p. 141, 283, 376, 462, 526, 546; Vol. II., p. 479; Vol. III., p. 134), only new details will be mentioned in the following:

The patents of Kjellin for the induction furnace are now owned by the Metallurgiska Patent Aktiebolaget, in Stockholm (which also owns the patents for Groendal's process of treating iron ores by means of magnetic separation and briquetting). This company has given licenses to the new owner of the Gysinge plant, to a "prominent English steel works," and to a French firm (at Voiron), which has already a 61-kw furnace in operation. The Siemens & Halske Co. have acquired the exclusive rights for Germany and Austria-Hungary.

At Gysinge, a 58-kw Kjellin furnace, finished in November, 1900, had an output of 600 to 700 kg steel in 24 hours, thus requiring 2140 kw-hours per ton of steel.[1] Later on a 170-kw [illegible] the following refer to this furnace.

The construction of this furnace was already shown in the drawings on page 577 of our first volume. That vertical portion of the magnet core which is inside the primary coil carrying the alternating current has not an exactly square cross-section, but the four corners of the square are taken off to provide vertical air chambers for cooling purposes.

The cross-section of the chamber, containing the metal, with its lining, is shown in Fig. 1. It will be seen that the cross-section of the metal chamber is essentially a rectangle with a rounded bottom. This is the only detail of construction which has not given complete satisfaction in practice. The trouble experienced is that the opening for charging is too narrow, and that it is difficult to repair the linings of the vertical walls of this chamber. It is, therefore, intended to use inclined walls in future, so that the chamber has a larger cross-section at the top than at the bottom. The cross-section of the chamber will, therefore, be in future more of the form of a triangle, the bottom being again rounded.

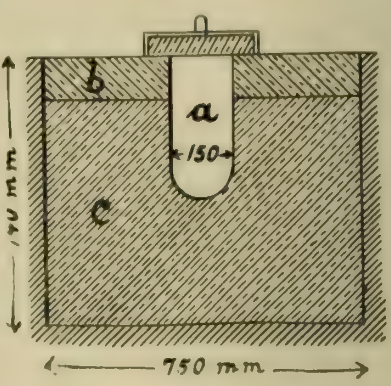

FIG. 1.—CONNECTION OF METAL CHAMBER AND LINING.

The first lining used in Gysinge was a silica lining, but this was not durable, and gave up too much silicon to the steel. It has, therefore, been replaced by a basic magnesite lining, stamped in a thickness of 300 mm.; 500 kilograms of sintered magnesite are mixed with 10 kilograms of finely ground caustic magnesite; then 40 kilograms of Holland clay are made up to a paste with water, and the whole is then mixed and stamped in. The entire furnace lining requires 2700 kilograms of sintered magnesite and the corresponding additions. Details of the cost[2] are given as follows:

First cost of lining.....................	$121.86
Cost of repairs during twelve weeks.....	37.98
Removal of the lining..................	13.40
Total	$173.24

On this one lining 285 tons of steel were made during a twelve-weeks run, making a total cost of lining amounting to 6.18 cents per ton of steel.

The operation is as follows: The charge being made up, it is placed behind the furnace, and as soon as the previous charge is run out the new one is put in, in two lots, the second half following 60 minutes after the first, and each charging consuming 15 minutes. The whole charge is molten in about two hours from the start, and a sample is then taken for carbon by the color test. The carbon being regulated to the desired point, the heat is increased, and 10 to 15 kg of 12 per cent ferro-silicon are added about 15 minutes before tapping. On tapping, if the steel is not perfectly quiet, a slight amount of aluminium (about 50 grams for a whole charge) is put into the metal in the casting ladle, and the steel is then cast into ingots of 85 to 200 kgs each. In the presence of Mr. Engelhardt, eight ingots were obtained from one charge, namely, four ingots each of 85 kg, three each of 100 kg, and one of 200 kg; the capacity of the furnace was 1350 kg, the total metal tapped 840 kg, and the duration of the operation 4 hours.

According to the record book of the plant an average of about 5000-kg ingots were cast in 24 hours, with an average power of 167.1 kw, corresponding to a consumption of 802 kw-hours per ton of steel. As an average of forty-eight further charges in 8 days at 170 kw, with a 4-hour duration of each charge, a power consumption of 770 kw-hours per ton is recorded. These figures refer to cold-starting materials. When 650 kg molten pig iron were introduced into the empty furnace and 1300 kg cold pig iron and scrap were added, the power consumption was 650 kw-hours per ton of steel.

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[1] In this article one ton means metric ton = 1000 kg = 2204 pounds.

[2] The original figures of Mr. Engelhardt's report are given in Swedish crowns. In converting the figures to United States money, 1 crown is [illegible] the figure used by the United States Treasury Department in 1903 for estimating the value of all foreign coins [illegible] imported into this country.

The figures 800 kw-hours for cold-starting materials and 650 kw-hours for molten starting materials are plotted in Fig. 2 for the 170 kw-furnace. These two figures therefore represent experimental results. The diagram shows how the power consumption per ton of steel depends upon the size of the furnace. The results obtained with the 61-kw furnace in France are also used in the diagram, the figure being 1220 kw-hours per ton. From the report of Mr. Engelhardt it would seem that these three figures for power consumption are the only ones in the diagram which were found by experiments, while the power consumption for larger sizes of furnaces are apparently estimated. It seems that a 736-kw furnace is in course of construction, and that for this furnace, with a capacity of 3740 kg and 2000 kg of tapped metal, the power consumption per ton of steel is estimated as 590 kw-hours when the starting material is cold pig iron, the daily output being 30 tons. If molten pig iron is the starting material the daily output is estimated to increase to 36 tons, corresponding to 490 kw-hours per ton. It is, of course, evident that the losses due to radiation and convection of heat (in percentage of the total power) are reduced by increasing the size of the furnace.

The figures for the power consumption, amounting to 489 or 331 kw-hours per ton with cold or molten starting materials respectively, are theoretical figures of Mr. Engelhardt, their calculation being given in detail in the original report.

If pig iron and ore are treated the power consumption is higher, and amounts in the average to 1200 kw-hours.

Mr. Engelhardt gives a great many tables giving the complete record of several charges carried out in his presence. Two of them, representing two different types of treatment, may be mentioned here. First, the "scrap process." The whole operation, using 400 kg of iron and 425 kg of open-hearth steel scrap, lasted 3 hours and 21 minutes. Tapping lasted 6 minutes. The scrap was introduced in two portions, 200 kg being introduced 10 minutes after starting and 225 kg 1 hour after starting; 16.5 kg of 98 per cent Cr were introduced 2¾ hours after starting, 10 kg of 12 per cent ferro-silicon 3 hours after starting. The total electrical energy used was 567.4 kw-hours. The analysis of the steel gave 1.71 per cent C, 0.13 Si, 0.30 Mn and 2.35 Cr.

As an example of the "ore process" the following run may be given: There were charged 850 kg of pig iron, and at intervals during 4 hours 85 kg of iron ore briquettes (introduced in four different portions), and 11 kg of limestone (in six portions), while just before tapping, nearly 6 hours after starting, 15 kg of 12 per cent ferro-silicon and 2 kg of 85 per cent ferro-manganese were added. A total of 1002.90 kw-hours was consumed in this run. The tapping lasted 5 minutes and the analysis of the steel gave:

	%C	Si	Mn	P
Beginning of tapping..........	1.19	0.17	0.24	0.102
Middle of tapping............	1.11	0.18	0.25	0.086
End of tapping...............	1.18	0.13	0.24	0.150

The raw materials used in commercial work at Gysinge are given in Table I.

TABLE I.

Raw Materials Used in Gysinge.

Pig Iron, Dannemora	C 4.5% Si 0.08 S 0.015 P 0.018 Mn 1.00 Cu 0.015 As 0.035	For making carbon and chrome steel by the "scrap process."
$22.00 per ton in Gysinge.		
Guldsmedhütte	C 4.0% Graphite 1.16 Si 0.73 P. 0.051 Mn 0.11	For the "ore process" since richer in P.
Scrap from basic open-hearth furnace in Hohendahl. Assumed 0.1%.		For making carbon and chrome steels by "scrap process."
$25.50 to $26.80 per ton in Gysinge.		
From their own forge in Gysinge. Assumed 1.0%.		For making carbon and chrome steels by "scrap process."
Assumed $26.80 per ton.		
Ferro-silicon made in blast furnace; 12% Si.		In every charge before tapping, to prevent blowholes.
$30.80 per ton in Gysinge.		
Ferro-manganese made in blast furnace; 85% Mn.		In "ore process" added at the end of the charge.
Ferro-chrome, manufacturer unknown; 66% Cr, 8% C.		For making chrome steel.
Chromium, probably made by aluminothermic process; 98% Cr.		For making chrome steel.
Ore briquettes from Groendal process of magnetic separation and briquetting.		60 to 65% Fe. For "ore process." 69.5% Fe. To be used later in "ore process." Was not yet available in Gysinge in sufficient quantity.
Assumed price $4.30 to $4.55 for 70% Fe free port.		

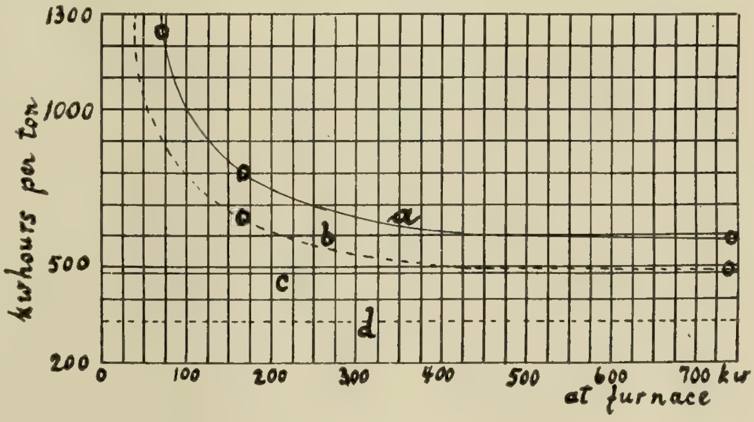


FIG. 2.—INFLUENCE OF SIZE OF FURNACE ON POWER CONSUMPTION.

We may again return to Fig. 2 and Mr. Engelhardt's theoretical calculations. From the data obtained, Mr. Engelhardt calculates that a furnace of 1000 horse-power (736 kw) should have a capacity of 3740 kg and run out regularly 2-ton charges each 1½ hours when using cold pig iron, or each 1 1-3 hours using melted pig iron, with a total output of 30 or 36 tons per day, using respectively 590 or 490 kw-hours per ton of steel. "Since there are theoretically needed 489 and 331 kw-hours per ton, in the two cases," the efficiencies to be attained with this size of furnace are 83 and 68 per cent respectively, and it is evident from the curves that larger furnaces would not attain any greater efficiency.

The abstractor calls attention to the fact that Mr. Engelhardt has calculated all his efficiencies too high; that the heat on the melted steel running from the furnace would not be over 340 calories per kilogram of steel at the most, whereas, it has been [illegible] 680 calories in his calculations, and the efficiencies stated as attained are to be at least cut in half. There is, however, a comforting compensation in the fact that this criticism also halves the heat or power stated as theoretically necessary to melt the steel, and so gives to the designers and operators of electric furnaces a hundred per cent greater chance for improvement and final economy of power than Engelhardt's figures promised.

At Gysinge a 300-horse-power turbine is directly coupled with a dynamo giving single-phase alternating-current of 3000 volts at 15 periods per second. The phase difference between the e.m.f. and the current changes in different runs with the quantity of materials treated, and it also changes in a single run according to the amount of material which has been charged into the furnace. With small charges (1350 kg) the power factor (cosine of the phase difference) is up to 0.80; with larger charges (like 1800 kg) it decreases in the average to 0.68.

Concerning the waste, it is stated that from the 23d to 28th of October 27,069 kg raw materials were charged into the furnace, and 26,131 kg steel were obtained, while there were at least 400 kg left in the furnace, so that the difference of 27,069 and 26,531, that is, 538 kg, or 2 per cent, represented the loss in this period. Mr. Engelhardt considers a 2 per cent waste as a fair average.

In the "scrap process" there is no possibility of removing certain elements which are not wanted. The process is of such a nature that whatever is introduced into the furnace is found afterwards in the product. The only exception is carbon. At Gysinge it is customary to calculate the amount of carbon introduced into the charge by the following method: From the amount of carbon calculated from the carbon contained in the metal left in the furnace, and from the new raw materials, a certain empirical percentage is taken off to account for the burning of carbon in air during charging, or by the oxygen contained in the raw materials, etc. This constant percentage is between 0.4 and 0.5 per cent, and is assumed in general as 0.45 per cent, while the exact adjustment is made by proper additions of pig iron, or ore, after the content of carbon has been determined by the color test, as follows:

Scrap Process.—The metal left in the furnace from the previous charge contains 1 per cent carbon, the next charge is desired also to contain 1 per cent carbon.

Old Charge:	Kg C.
400 kg steel at 1% C..................	4
New Charge:	
300 kg pig iron at 4.5% C..............	13.5
500 kg wrought iron at 0.1% C.........	0.5
75 kg scrap at 1% C..................	0.75
1275 kg, total	18.75

18.75 ÷ 12.75 = 1.47% C, subtracting 0.47% yields 1% C.

Mr. Engelhardt gives numerous mechanical tests of Gysinge steel. Some of them, made in the Institute of Technology in Stockholm, and on material showing less than 0.02 per cent P, 0.011 per cent S, and with carbon from 0.18 up to 0.91 per cent, the following figures were obtained:

Elongation, 9.0 to 23.8 per cent.

Limit of proportionality, 30.2 to 44.3 kg per sq. mm.

Elastic limit, 33.4 to 50.7 kg per sq. mm.

Tensile strength, 43.3 to 97.6 kg per sq. mm.

The quality is, therefore, close to the best of crucible steel.

Mr. Engelhardt then makes some comparisons of the induction furnace process, first with the crucible steel process and then with the open-hearth process. In comparison with the crucible steel process he concludes that the Kjellin furnace produces a steel equal in quality to crucible steel, but is able to furnish it at a lower cost of production. Since this has been pointed out repeatedly in our columns, and since no new arguments are brought forward, this reference may suffice.

As far as concerns competition with the Siemens-Martin open hearth steel, Mr. Engelhardt assumes an average cost of $18.75 to $20.00 per ton for the basic process, and a cost of $21.25 to $22.00 per ton for the acid process. With a 736-kw electric induction furnace he estimates a total cost of $17.85, or $17.01 per ton, with cold or molten pig iron respectively as starting material. The only big item in this cost, which depends essentially on local conditions, is the cost of power; this is based on the assumption of 0.5 cent per kw-hour. Mr. Engelhardt, therefore, concludes that the electric induction furnace can compete successfully with the open-hearth furnace as long as the kw-hour does not cost more than 0.5 cent.

The Sources of Raw Materials for Niagara's Electrochemical Industries.

By Morris M. Green.

The development of hydroelectric power at Niagara Falls for electrochemical plants brings with it sundry interesting points regarding the cost of the various raw materials which are brought there for manipulation.

Among the most prominent of the materials are salt, used for making caustic soda in the Acker and Castner processes; coke, used in the graphite, carborundum, siloxicon and carbide industries; pure glass sand, used for graphite, siloxicon and carborundum manufacture; anthracite coal, used for making graphite; lime for bleaching powder and carbide. The most convenient sources and the approximate cost of these materials will now be discussed in detail.

Salt.

Fine beds of rock salt, suitable for manufacturing the purest evaporated salt, by first dissolving the rock salt, then evaporating by steam heat, are found in western New York near Warsaw, Retsof, and other towns, in the Genesee Valley.

There are two well-known methods of evaporating the salt from the brine, the first by "grainers" or vats open to the air, which produce a coarse salt; the other, a vacuum process in closed vacuum pans, producing a fine grained salt, at reduced consumption of fuel. With bituminous coal of fair quality, the relative fuel required per ton of salt, by the two processes is about as follows:

"Granier" process	1300 pounds per ton of salt
Vacuum pan process..	900 pounds per ton of salt

Associated with chloride of sodium in the rock salt, are small quantities of sulphate of lime and chlorides of calcium and magnesium, all of which linger as impurities in the evaporated salt. Treatment of the brine with carbonate of soda will throw down carbonates of these bases, leaving soluble salts behind. This is desirable for some electrolytic processes, but also raises the cost of the salt.

Ordinary evaporated salt may be said to contain 98½ chloride of sodium.

With coal at $2.50 per ton, the cost per ton, f. o. b. works, of salt may be said to run about as follows, including all expenses:

Grainer salt	$2.12 per ton
Vacuum salt	1.70 per ton

The freight rate on salt per ton from the Warsaw district to Niagara Falls is 70 cents per ton for an 80-mile haul.

At Cleveland, Ohio, salt can be evaporated with coal costing not over $1.25 per ton, but the distance to Niagara Falls is 200 miles, and the freight rate is $1 per ton.

Coke

The coke used in the electric furnace plants must be as pure as possible, as impurities require consumption of power for their volatilization.

Ordinarily the best grade of massive Connellsville beehive coke is bought, averaging 90 per cent fixed carbon.

The selling price of Connellsville coke at the ovens for the five years ending 1903 has averaged about $2.30. Adding freight charges on the 300-mile haul to Niagara Falls would bring the price to about $4 delivered.

It is unfortunate that the large amount of breeze coke, produced by the by-product coke oven plants should be so high in ash as to debar it for electric furnace work, as it is sold cheaply to find a market.

Glass Sand.

Large quantities of the purest kind of glass sand are used at Niagara Falls in the carborundum, graphite and siloxicon plants. As impurities are objectionable for the reason mentioned for coke, the sand is shipped great distances. At one time Niagara Falls was supplied with glass sand, averaging 99 per cent silica, from a district near Pittsburg, Penn., 300 miles from Niagara Falls. At present the Carborundum Company is shipping sand 600 miles, from Wedron, Ill., to Niagara Falls. This deposit of sand has great natural advantages for mining cheaply, and is unusually pure, testing 99½ per cent SiO_2. This sand has been delivered in Niagara Falls for $3.50 per ton, three quarters of this being transportation charges.

Anthracite Coal.

This material, used in egg size for making artificial graphite, by the Acheson process, costs about $4 a ton at the mines in the Lehigh Valley. Its cost at Niagara Falls, including freight, fluctuates between $6 and $7 a ton.

This material, being controlled by railroad interests, must be bought in the market.

Lime.

As the purest lime is required for electrochemical industries, only certain quarries of limestone, averaging 99 per cent $CaCO_3$ are available.

Pure limestone can be quarried for 40 cents a ton, where strata are thick and well exposed, and where quarrying is carried on on a large scale. If the stone is burned with coke for fuel, 200 pounds of coke per ton of stone, or about 400 pounds per ton of lime, are required. If bituminous coal is used, 500 pounds per ton of lime are used.

Coal is much cheaper fuel; also lime can be burned with it without mixing with the ash of the coal, whereas if coke be mixed directly with the limestone, the ash of the coke tends to contaminate the lime.

The limestone for a ton of lime costs $1, at a minimum; coal from 50 to 75 cents, so that lime seldom costs less than $3 a ton, at a minimum. Four dollars a ton is a medium price, at the kiln.

Industrial Resistance Furnaces

(Gin, Colby, Kjellin.)

By F. A. J. FitzGerald.

In attempting to discuss various forms of industrial furnaces of the resistance type we are at once met with the difficulty that, as a rule, details of the working of such furnaces, whether these are in the commercial or experimental stage, are lacking. If we desire to study the working of the blast furnace or the open-hearth furnace we have a voluminous literature, for these furnaces have reached a stage in their evolution where those who use them are not afraid of giving away something valuable if they say anything about them. A study of the patent literature in connection with electric furnaces shows that all experimenters are anxious to monopolize discoveries as far as possible, and in a new art like that of the working of electric furnaces such discoveries are naturally frequent. It thus happens that the experimenter, on overcoming a difficulty in connection with his work, at once conceives the idea of monopolizing his discovery by an application to the patent office.

This attitude on the part of the experimenter is natural, and it is right that he should be rewarded for his work; but, unfortunately, it happens that from his lack of knowledge of the work of others he continually obtains patents on details that are already well known, or that have been patented by others, or in his anxiety to get a patent of some kind he obtains claims on well-known principles which cannot be regarded in any way as inventions or discoveries. Thus, on the one hand, we find a patent depending on the grading of carbon grains for the purpose of varying the resistivity of the resistor which they are intended to form; on the other hand, we find patents on obtaining increased temperature in a resistor by diminishing its sectional area or on adjusting the potential difference between the terminals of a furnace, so as to obtain a definite rate of generation of heat energy in the furnace.

Therefore, although some furnaces have been described with considerable detail as regards their construction and working, the majority of those in commercial use are more or less unknown quantities, and of those proposed for commercial use we have little information as to the experiments on which they are based. The only method of studying such furnaces is to apply general principles to their consideration, and attempt in that way to form some notion of their working.

Gin Furnace.

Looked at from a purely ideal point of view the simplest form of electric furnace is that in which the current is passed through the material to be heated, the heating effect being produced by the resistance of the substance itself, or, in other words, the furnace in which the charge constitutes the resistor. An example of this form of furnace is found in that proposed by Gin for the production of steel by melting cast iron and scrap or ore. The furnace is built by forming a long, narrow, shallow channel of refractory material such as chromite; the channel is filled with molten cast iron, and the current passed through the latter, the ends of the channel being supplied with water-cooled iron terminals for that purpose. A detailed description of the furnace may be found in an article by Bennie, *Electrochemical Industry,* Vol. II., No. 1, p. 20.

G. Gin.

The heat generated in the channel of metal per second may be calculated from the equation:

$$JH = I^2R = \frac{E^2}{R} \qquad (1)$$

Where J is the mechanical equivalent of heat, or the number of ergs given to a gram of water in raising it 1° C. in temperature, H, the number of calories (gram — degree C.), I is the current, E the potential difference between the ends of the channel and R the resistance, all expressed in absolute units. Since $J = 4.2 \times 10^7$, then if I, E and R are respectively amperes, volts and ohms, equation (1) becomes

$$H = \frac{I^2R}{4.2} = \frac{E^2}{4.2R} \qquad (2)$$

If the specific heat of the metal is c, its weight in grams is w, and the increase in temperature per second is Θ, then

$$H = cw\Theta \qquad (3)$$

hence from (2) and (3)

$$cw\Theta = \frac{I^2R}{4.2} = \frac{E^2}{4.2R} \qquad (4)$$

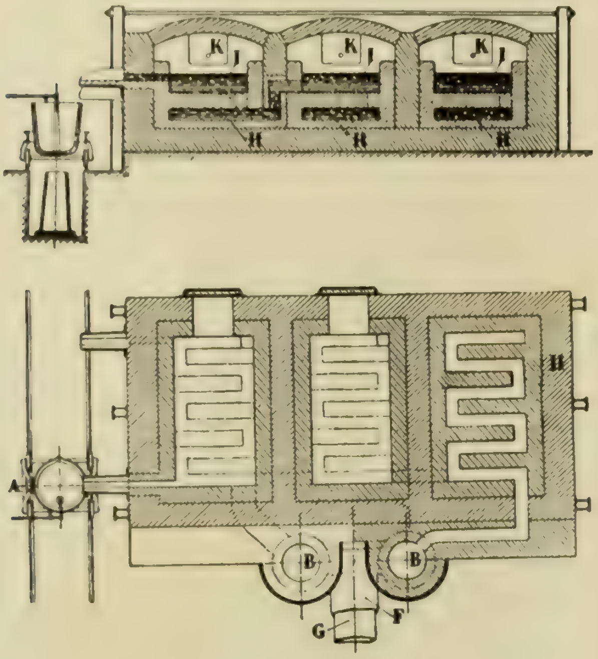
assuming that there is no loss of heat. Then the increase of temperature per second is

$$\Theta = \frac{I^2R}{4.2cw} = \frac{E^2}{4.2Rcw} \qquad (5)$$

If W is watts we also have this equation:

$$W = 4.2cw\Theta \qquad (6)$$

Now, assuming that it is desired to heat 1 kg of iron from 0° to 1600° in one hour, and that the average value of the specific heat of iron is 0.3, then from (6)



FIGS. 1 AND 2.—GIN FURNACE.

$$W = \frac{4.2 \times 0.3 \times 1000 \times 1600}{3600} = 560$$

Therefore, provided there is no loss of heat, 560 watt-hours are required for 1 kg of iron, or 560 kw-hours for 1 metric ton.

The resistivity of molten iron, according to Gin, is 0.000175 ohm for a centimeter cube, therefore, if the length of the channel is *l*, its sectional area A, the current is 11,200 amps. 50 volts, and the density of the molten iron is 7, then we have

$$0.000175\,\frac{l}{A} = \frac{50}{11200}$$

$$7lA = 1000000$$

from which we get the following values:

$$l = 1910 \text{ cms}$$
$$A = 75 \text{ square cms}$$

for a furnace that will fuse 1000 kg of iron per hour, assuming no loss of heat.

It is plain that in a furnace of such great length, compared with such a small cross section, the heat losses would be very great. To overcome this objection the channel is bent on itself several times, thus greatly diminishing the radiating surface.

In the furnace proposed by Gin it is not intended to start with cold iron in the channel, but to run in molten iron from a blast furnace or cupola, and then raise its temperature and treat it with ore or scrap iron, or both, so as to produce the desired grade of steel. It would be difficult to start a furnace of this kind using cold iron for the charge unless some device for regulating the voltage was used. Probably the best form in which to have the charge would be scrap iron in small pieces, which would offer numerous high-resistance points of contact; but a furnace charged in this way would certainly be difficult to manipulate. Once the furnace was filled with molten metal the practice probably would be to tap off only a portion of the charge and then add more iron to the furnace.

Here it it may be well to consider a possible contingency, viz.: the freezing of the metal in the furnace. The resistivity of iron at ordinary temperatures may be assumed as 0.00001 ohm. Then, in order to use 560 kw for melting the frozen iron in the channel of the furnace, we have imagined we should require the volts (E) to be

$$E = \sqrt{\frac{0.00001 \times 1910 \times 560000}{75}}$$

and the current I would be

$$I = \frac{560000}{12} = 47000 \text{ amps.}$$

The electrical apparatus necessary for a contingency of this nature would be, at the present time, somewhat expensive.

In actual practice the current used in the furnace would be larger than appears in the calculations, since the assumption has been made that there are no heat losses. Even with the furnace built of dimensions given above and the channel so arranged as to avoid heat losses, the furnace would be highly unsatisfactory on account of the small sectional area of the channel. To shorten the channel and increase its cross-section it would be necessary to lower the volts and increase the amperes of the current, and then electrical difficulties will be encountered, for in using such a large alternating current the self-induction of the furnace circuit becomes great and the power factor low.

EDWARD A. COLBY.

Another objection that may be raised in connection with this furnace is the probable difficulty that would be experienced in keeping walls of the channel in repair. With an ideal refractory material such troubles might be avoided; but this material has still to be found.

In the simple channel furnace it would not be at all easy to carry out the necessary work of treating the molten iron with the materials required for producing the desired grade of steel. In a later design Gin has elaborated his furnace with the object of avoiding this objection, and the destructive action of the slag on the walls of the channel. This modified furnace is shown in vertical and horizontal section in Figs. 1 and 2. We have here three reservoirs I connected in series by the channels H, the terminals of the channels being at B, B. The heating of the charge is obtained by the passage of the current through the molten metal in the channels, while the reservoirs are used for charging the furnace and treating the iron to produce the desired grade of steel. Thus, in treating the iron by the oxidation method a furnace with three reservoirs is used, the molten iron being charged into the first reservoir and drawn off from the third. In the first reservoir silicon and manganese may be removed in the second carbon and phosphorous, while in the third reservoir recarbonization is carried out and the steel tapped off. As the metal is drawn off from the third reservoir the metal which has been heating in the channel between the second reservoir and the third flows into the third reservoir, while the contents of the second reservoir flows into the channel. Meanwhile the contents of the other channels and reservoirs are flowing through the system in a similar manner.

Ignoring the objections that may be raised on *a priori* grounds to this type of furnace for smelting iron, it must be admitted that the ideal method of heating any substance by an electric current is to pass the current through the substance itself. But until some definite information on the actual working of such a furnace is obtained it would be well to suspend judgment thereon.

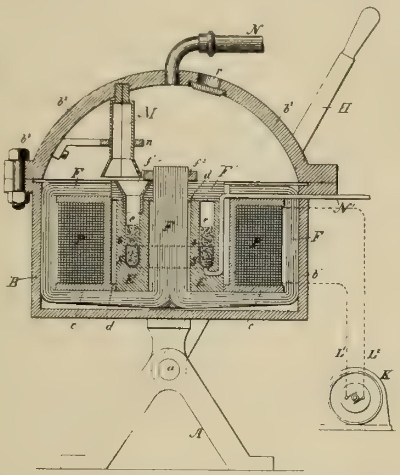


FIG. 3.—DIAGRAM OF KJELLIN FURNACE. (FROM PATENT 428,552, MAY 20, 1890.)

THE COLBY AND KJELLIN FURNACES.

The furnace patented by Colby[1] and the similar furnace recently worked out by Kjellin, in Sweden,[2] heat the material under treatment by passing the current through it directly. In these furnaces the heated material forms the secondary circuit of a transformer, and, consequently, the use of terminals is avoided. In the manufacture of steel in the electric furnace the use of carbon terminals in contact with the molten steel must be avoided, and thus various devices, with this end in view, are employed. In the Gin furnace water-cooled steel terminals are used; but obviously doing away with terminals altogether, as in the Kjelllin and Colby furnaces, is best. In this furnace, as in the Gin furnace, it is necessary to use a channel of considerable length as compared with the cross-section, and the difficulty of obtaining a suitable refractory material for its construction probably arises. A more serious disadvantage is found in the low power factor of the apparatus. Thus, in the case of the 225-kw furnace shown to the Canadian commission the power factor was found to be about 0.6, and if it was desired to use larger furnaces the power factor would be still less. To correct this objectionable feature to a certain extent the frequency of the current may be decreased.

F. A. KJELLIN.

In spite of these obvious objections, the Kjellin furnace has been worked with considerable success and excellent steel produced by it. This furnace would compete with the ordinary crucible furnace for steel, and the very poor efficiency of the latter is well known, so that the induction furnace might be used with important economic advantages.

Metallurgical Calculations.—VI.

By J. W. Richards, Ph.D.

Professor of Metallurgy in Lehigh University.

In several of the preceding installments we have given the heats of formation of alloys and compounds and the thermophysics of the elements. Before passing to the thermophysics of alloys and compounds and problems involving their use, we will consider a few simple cases of the application of data so far given. Such include operations in which metals are melted or volatilized, or amalgams retorted. A few words may be in order, to clear the ground, regarding what is to be regarded as the efficiency of a furnace.

Efficiency of Furnaces.

Under this term we must distinguish a generic sense and a specific sense, the first referring to furnaces in which the object is to maintain a certain temperature for a certain time with the minimum consumption of fuel, the second, in which the object is to perform a certain thermal operation with the smallest consumption of fuel. In the first case, one furnace may be compared with another, and thus comparative efficiencies calculated; in the second case real or absolute efficiencies can also be calculated. A few examples will illustrate this difference, which is an essential difference as far as making calculations is concerned.

Cases of Specific Efficiency: Whenever it is desired to melt a metal for the purpose of casting it, a certain definite amount of heat must be imparted to the metal, and the ratio between this efficiently utilized heat and the heating power of the fuel consumed, is the efficiency of the furnace. If the furnace is electric the theoretical heat value of the electric energy used is the divisor. If, in addition to the heat required to raise the substances to the desired temperature, there is also heat absorbed in chemical reactions, this amount can be added in as usefully applied heat, and the sum of this and the heat in the final products be regarded as the total efficiently applied heat. If a blast furnace takes iron ore and furnishes us melted pig iron, the sum of the heat absorbed in the chemical decomposition of the iron oxide and the sensible heat in the melted pig iron is the efficiently applied heat, because it is the necessary theoretical minimum required; all other items are more or less susceptible of reduction, but these are necessary items and, therefore, measure the net efficiency. If my dwelling requires 200 cubic feet of hot air per minute at 150° F. to keep it at 65° F., while the outside air is at 0° F., the ratio of the heat required to warm the 200 cubic feet of air from 0° F. to 150° F., to the calorific power of the fuel used per minute, measures the *specific* efficiency of the "heater;" the question whether this amount of hot air keeps the temperature of the rooms at 65° F. is a question of the *general* efficiency of the construction of the house.

Cases of Generic Efficiency: Such are those in which practically all the heat generated eventually leaves the furnace by radiation or conduction, or useless heat in waste gases; this is the case when a certain temperature has to be continuously maintained for a given time, and where the *time* element is the controling one, and not any definite amount of thermal work is to be done. Examples are numerous. An annealing furnace, where steel castings, let us say, are to be kept at a red heat for two days, or a brick kiln, where several days slow burning are required, or a puddling furnace, where the melted iron must be held one to two hours to oxidize its impurities. In all these cases we may say that one furnace keeps its contents at the right heat for the right time with so much fuel, another does the same work with 10 or 25 per cent less fuel, and is, therefore, 10 or 25 per cent more efficient; but we cannot, in the nature of the case, speak of the absolute or specific efficiency of the furnace, because there is no definite term, expressible in calories, to compare with the thermal power of the fuel.

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[1] U. S. Patents, Nos. 428,378 428,379, May 20, 1900.
[2] *Electrochemical Industry*, Vol. I., No. 11, p. 376.

In many cases the two efficiencies are mixed in the same process or operation, and then the calculation of absolute or specific efficiency can be made for that portion of the operation wherein a certain definite amount of thermal work is done. Thus, in an annealing kiln, 50 tons of castings may be brought up to annealing heat in 24 hours, starting cold, and the heat absorbed by the castings compared with the calorific power of the coal burnt during this period is a measure of the real efficiency of this part of the operation. During the rest of the operation, while the castings are simply kept at annealing heat, there can be no calculation of the absolute or specific efficiency of the furnace, because one of the terms necessary for the comparison has disappeared; in that part of the process we can only speak of relative efficiency compared to some other furnace doing a similar operation.

It goes, almost without saying, that we can, of course apply the conception of efficiency in its relative or general sense to the whole operation or to any part of it.

Problem 6.

The Rockwell Engineering Co. state in their current advertisements that their regenerative oil-burning furnace melts 100 pounds of copper with the consumption of less than 1.5 gallons of oil. Assume that 1.5 gallons of oil is used, and that the copper is heated from 25° C. to melted metal 100° C. above its melting point.

Required: The "efficiency" of the furnace; *i. e.,* its specific efficiency as calculated from the net heat utilized.

Solution: One gallon of fuel oil averages in weight 7.5 pounds, and its calorific power 11,000 Calories per kilogram, or 11,000 pounds Calories per pound. The calorific power of the fuel used in melting 100 pounds of copper is therefore:

Heat generated $11,000 \times 7.5 \times 1.5 = 82,500 \times 1.5 = 123,750$ pounds Calories.

The heat imparted to the copper is as follows, taking the data from Article V. of these calculations:

Heat in 1 lb. melted copper at melting point	=	162 lb. Cal.
Heat in 1 lb. solid copper at 25° C	=	2 "
Heat required to just melt the copper	=	160 "
Heat to superheat liquid copper 100° C. 0.133 × 100	=	13 "
Total heat expended on each pound of copper	=	173 "
Heat usefully applied per 100 pounds	=	17,300 "

$$\text{Net efficiency of furnace} = \frac{17,300}{123,750} = 0.14 = 14\%$$

It is proper to remark that although this efficiency appears low, yet it is considerably greater than is attained in simple melting holes or wind furnaces, and yet the calculations show what a large margin for improvement and greater efficiency exists in even some of the best and relatively most efficient metallurgical furnaces.

Problem 7.

In the distillation of silver amalgam in iron retorts, 1000 kilos. of amalgam, containing 200 kilos. of silver, is retorted with the consumption of 550 kilos. of wood, the mercury vapor passes off at an average temperature of 450° C., and the silver is raised towards the end of the operation to 800° C., in order to expel the last of the mercury. Assume the calorific power of the wood 3000 Calories.

Required: The net efficiency of the furnace.

Solution: The heating power of the wood is $550 \times 3000 = 1,650,000$ Calories.

The heat utilized is that absorbed in separating the silver from the mercury, plus the sensible heat in the mercury vapor at 450°, plus the sensible heat in the silver at 800°. These are calculated as follows:

Heat to decompose amalgam = 2470 Calories per 108 kilos. of silver $= 200 \times (2470 \div 108) = 200 \times 238 = 47,600$ Calories.

Heat in silver at 800°, using Pionchon's formula, is $800\,[0.05758 + 0.0000044\,(800) + 0.000000006\,(800)^2] \times 200 = 10,390$ Calories.

Heat in 800 kilos. of mercury vapor at 450° is

(*a*) heat to boiling point (Naccari) $356\,[0.03337 - 0.00000275\,(356) - 0.0000000667\,(356)^2] \times 800$	=	14,699 Cal.
(*b*) heat to vaporize 77.5×800	=	62,000 "
(*c*) heat in vapor at 450° = $0.025 \times (450 - 356) \times 800$	=	1,880 "
Total	=	78,579 "

Heat usefully applied:

In decomposing amalgam	=	47,600 Cal.
In mercury vapor, as sensible heat	=	78,579 "
In silver, as sensible heat	=	10,390 "
Total	=	136,569 "

$$\text{Efficiency of furnace} = \frac{136,569}{1,650,000} = 0.083 = 8.3\%$$

Problem 8.

In a zinc works, impure zinc is refined by redistillation in fire-clay retorts, a bank of retorts distilled 970 kilos. of zinc with the expenditure of 912 kilos. of small anthracite coal. Assume that the zinc vapors pass out of the muffles at the boiling point (930°).

Required: (1) The net efficiency of the furnace.

(2) The electrical power which would be required, in horse-power-hours, to do the same work, assuming the heating efficiency of the electric furnace is 75 per cent.

Solution: (1) The small anthacite may be assumed to have a calorific power of 7850 Calories; therefore, the total heat which should be developed is $7850 \times 912 = 7,159,200$ Calories. The heat in 1 kilo. of zinc in the state of vapor at its boiling point can be calculated from the thermophysical data supplied for zinc as:

(*a*) In solid zinc to melting point (420°).....	47.34 Cal.
(*b*) Latent heat of fusion....................	22.61 "
(*c*) Heat in melted zinc to boiling point (930°)	65.05 "
(*d*) Latent heat of vaporization.............	370.15 "
Total.	505.15 "
Heat required for 970 kilos.	= 489,995. "

$$\text{Efficiency of furnace} = \frac{489,995}{7,159,200} = 0.068 = 6.8\%$$

(2) One electric horse-power-hour	=	644.0 Cal.
Efficiently applied heat = 644 × .75	=	483.0 "

$$\text{Electric horse-power-hours required} = \frac{489,995}{483} = 1013 \text{ E.H.P. hours.}$$

One metric ton of zinc requires $\frac{1013}{0.970} \doteq 1044$ E.H.P. hours.

Cost of power, at $20.00 per E.H.P. year $= \frac{20.00}{8766} \times 1044 = 0.00228 \times 1044 =$ $2.38.

This cost of electric power would replace the use of $\frac{0.912}{0.970}$ metric tons of small anthracite, equal to a cost of $2.53 for electric power sufficient to replace a metric ton of coal for this purpose.

Many other examples could be given of the technical use of the thermophysical data concerning the elements, but the problems given illustrate the methods of calculation.

When one is acquainted with some of the ordinary metallurgical operations, such as melting and distilling the metals, it is surprising to notice how little is known or thought of the efficiency or lack of efficiency of the furnaces used. One man melts 100 pounds of metal by the use of 150 pounds of coal, he builds a new furnace and does it more cheaply by using 100 pounds of coke, which is certainly relatively more efficient; but it is seldom that the operator knows that in one case he is getting probably only 7 per cent efficiency from his fuel and in the other case only 10 per cent. It is the knowledge of these absolute efficiencies which tells the practical man just what he is accomplishing, and shows him how much room there still remains for improvement.

THERMOPHYSICS OF ALLOYS.

There does not exist, in technical literature, much data of this nature concerning alloys. There is here a wide and interesting field for metallurgical research, whose cultivation would yield results both of high practical and high theoretical interest, and yet it is comparatively untouched. What is wanted is complete data concerning the specific heat of solid and liquid alloy, and latent heat of fusion. These, combined with the determination of the heat evolved in the alloying, would furnish a sound basis for a practical theory of alloys, besides enabling workers with these alloys to control the efficiency of their furnaces and, in general, to know with scientific exactness what they are accomplishing.

ALLOYS OF TIN AND LEAD.

Per Cent of Tin.	*Sm.*	*Latent Heat of Fusion.*
4.8 ($Pb^{16}Sn$)	....	5.5 (Mazotto)
10.2 ($Pb^{5}Sn$)	....	8.0 at 307° (Spring)
12.5 ($Pb^{4}Sn$)	....	8.3 at 292° (Spring)
16.0 ($Pb^{3}Sn$)	....	9.1 at 289° (Spring)
22.2 ($Pb^{2}Sn$)	....	9.5 at 270° (Spring) 7.9 (Mazotto)
36.3 ($PbSn$)	0.04073 (12° — 99°) (Regnault)	11.6 at 241° (Spring) 9.4 (Mazotto)
50.0 total heat to 0° in 1 kilo. melted metal		18.0 from 202° (Ledebur)
53.3 ($PbSn^{2}$)	0.04507 (10° — 99°) (Regnault)	10.5 at 197° (Mazotto)
63.1 ($PbSn^{3}$)	....	15.5 at 179° (Spring)
69.5 ($PbSn^{4}$)	....	17.0 at 188° (Spring)
83.0 total heat to 0° in 1 kilo. melted metal		21.5 from 205° (Ledebur)
90.1 ($PbSn^{16}$)	....	12.9 (Mazotto)

ALLOYS OF TIN AND BISMUTH.

Per Cent of Tin.	*Sm.*	*Latent Heat of Fusion.*
6.7 ($Bi^{8}Sn$)	....	11.4 Cal. (Mazotto)
22.1 ($Bi^{2}Sn$)	....	11.2 Cal. (Mazotto)
36.2 ($BiSn$)	0.0400 (20° — 99°) (Regnault)	11.6 Cal. (Mazotto)
53.1 ($BiSn^{2}$)	solid, 0.0450 (Regnault) liquid, 0.0454 (146° — 275°) Person	11.6 Cal. (Mazotto)
69.1 ($BiSn^{4}$)	....	11.1 Cal. at 140° (Mazotto)
82.7 ($BiSn^{8}$)	....	12.6 Cal. (Mazotto)
90.1 ($BiSn^{16}$)	....	12.8 Cal. (Mazotto)

ALLOYS OF TIN AND ZINC.

Per Cent of Tin.	*Sm.*	*Latent Heat of Fusion.*
78.4 $ZnSn^{2}$)	....	23.5 Cal. (Mazotto)
92.7 ($ZnSn^{7}$)	....	16.2 Cal. at 197° (Mazotto)
95.6 ($ZnSn^{12}$)	....	16.3 Cal. (Mazotto)
97.3 ($ZnSn^{20}$)	....	15.1 Cal. (Mazotto)

ALLOYS OF TIN AND COPPER.

Bell metal (20 per cent tin) Sm (14° — 98°) = 0.0862 (Regnault).

Bronze (15 per cent tin).

Total heat in melted metal (to 0°) = 130 Cal. (Ledebur).

If strongly superheated =143.5 Cal. (Ledebur).

ALLOYS OF TIN AND ANTIMONY.

Britannia metal (90 per cent tin) requires to melt it, starting cold, 28.0 Calories per kilogram (melting point 236°); with 82 per cent of tin, 25.7 Calories; melting point 205° (Ledebur).

ALLOYS OF TIN, BISMUTH AND ANTIMONY.

$BiSn^{2}Sb$ (bismuth 34.3, tin 41.9, antimony 23.8 per cent).
Sm (15° — 100°) = 0.0462 (Regnault).

ALLOY OF TIN, BISMUTH, ANTIMONY AND ZINC.

$BiSn^{2}SbZn^{2}$ (bismuth 29.8, tin 34.0, antimony 17.3, zinc 18.9 per cent).
Sm (15° — 100°) = 0.0566 (Regnault).

ALLOYS OF LEAD AND BISMUTH.

Per Cent of Lead.	*Sm.*	*Latent Heat of Fusion.*
11.1 ($PbBi^{8}$)	....	10.2 (Mazotto)
33.2 ($PbBi^{2}$)	....	6.4 (Mazotto)
39.9 ($Pb^{2}Bi^{3}$)	solid, 0.03165 (16° — 99°) Person liquid, 0.03500 (144° — 358°) Person	
42.7 ($Pb^{3}Bi^{4}$)	....	4.7 at 127° (Mazotto)
49.9 ($PbBi$)	....	4.0 (Mazotto)
66.6 ($Pb^{2}Bi$)	....	3.6 (Mazotto)
88.8 ($Pb^{8}Bi$)	....	4.9 (Mazotto)

ALLOYS OF LEAD AND ANTIMONY.

With 63.0 per cent of lead, Sm (10° — 98°) = 0.0388 (Regnault).

With 82.0 per cent of lead, heat in 1 kilo. melted metal = 15.6 Calories (Ledebur).

With 90.0 per cent of lead, total heat in 1 kilo. melted metal = 13.8 Calories (Ledebur).

ALLOYS OF LEAD, TIN AND BISMUTH.

D'Arcet's Alloy, containing 32.5 lead, 18.5 tin, 48.7 bismuth:

Sm solid (5° — 65°) = 0.0372 (Mazotto)
Sm solid (12° — 50°) = 0.049 (Person)
Sm solid (14° — 80°) = 0.060 (Person)
Sm liquid (107° — 136°) = 0.047 (Person)
Sm liquid (120° — 150°) = 0.0399 (Mazotto)
Sm liquid (136° — 300°) = 0.0360 (Person)
Latent heat of fusion = 5.96 Cal. at 96° (Person)
= 5.77 Cal. at 99° (Mazotto)

Rose's Alloy, containing 24.0 lead, 27.3 tin, 48.7 bismuth
Sm solid (5° — 65°) = 0.375 (Mazotto)
Sm fluid (119° — 338°) = 0.0422 (Person)
Latent heat of fusion = 0.85 Cal. at 90° (Mazotto)

Fusible Alloy, containing 31.8 lead, 36.2 tin, 32.0 bismuth:
Sm solid (18° — 52°) = 0.0423 (Person)
Sm solid (11° — 98°) = 0.0448 (Regnault)
Sm fluid (143° — 330°) = 0.0400 (Person)
Latent heat of fusion = 7.03 Cal. at 145° (Person)

Wood's Alloy, containing 25.8 lead, 14.7 tin, 52.4 bismuth, 7 cadmium:
Sm solid (5° — 50°) = 0.0352 (Mazotto)
Sm fluid (100° — 150°) = 0.0426 (Mazotto)
Latent heat of fusion = 7.78 Cal. at 75° (Mazotto)

Lipowitz's Alloy, containing 25.0 lead, 14.2 tin, 50.7 bismuth, 10.1 cadmium:
Sm solid (5° — 50°) = 0.0345 (Mazotto)
Sm fluid (100° — 150°) = 0.0426 (Mazotto)
Latent heat of fusion = 8.40 Cal. at 75° (Mazotto)

ALLOYS OF COPPER AND ZINC.

Red Brass	S at 0°	= 0.0899 (Lorenz)
	S at 50°	= 0.0924 (Lorenz)
(Copper 85%)	S at 75°	= 0.0940 (Lorenz)
Yellow Brass	S at 0°	= 0.0883 (Lorenz)
	at 50°	= 0.0922 (Lorenz)
(Copper 65%)	at 175°	= 0.0927 (Lorenz)

Heat in 1 kilo. of melted, somewhat superheated, brass = 130 Calories (Ledebur).

ALLOYS OF COPPER, ZINC AND NICKEL.

German Silver
(74Cu. 20Zn. 6Ni) Sm (0 — t) = 0.0941 + 0.0000053t (Tomlinson)

ALLOYS OF COPPER AND ALUMINIUM

Copper 88.7% Sm (20° — 100°) = 0.10432 (Luginin)

ALLOYS OF SILVER AND PLATINUM.

Silver 66.7% Sm (0 — t) = 0.04726 + 0.0000138t (Tomlinson)

ALLOYS OF MERCURY AND TIN.

HgSn (37.1% Sn) Sm (— 30° — 15°) = 0.04083 (Schuz)
(— 25° — 15°) = 0.04218 (Schuz)
(22° — 99°) = 0.07294 (Regnault)
HgSn (54.1% Sn) Sm (25° — 99°) = 0.06591 (Regnault)
HgSn² (74.7% Sn) Sm (— 16° — 15°) = 0.05039 (Schuz)

ALLOYS OF MERCURY AND LEAD

PbHg (50.9% Pb) Sm (— 60° — 20°) = 0.03458 (Schuz)
Sm (23° — 99°) = 0.03827 (Regnault)
Pb²Hg (67.4% Pb) Sm (— 72° — 20°) = 0.03348 (Schüz)

ALLOYS OF CADMIUM AND TIN.

CdSn (67.8% Sn) Sm (— 77° — 20°) = 0.05537 (Schuz)
whence we have Sm (0 — t) = 0.0557 + 0.0000366t (Schuz)

ALLOYS OF IRON AND CARBON.

Soft Steel (0.15% carbon) Sm (20° — 98°) = 0.1165 (Regnault).

Hard Steel (1.00% carbon) Sm (20° — 98°) = 0.1175 (Regnault).

Total heat in 1 kilo. melted steel at 1350° = 300 Calories (Ledebur).

Cast Iron (4.0% carbon) Sm (0 — 1200°) = 0.173
Sm (0 — t°) = 0.12 + 0.000046t

Total heat in 1 kilo. melted at 1200° = 245 Calories (Ledebur).

Total heat in 1 kilo. coming from blast furnace = 2?0 to 325 Calories (Akerman).

Problem 9.

A steel melting crucible contains 110 pounds of steel, which is melted in a wind furnace with the use of 150 pounds of coke. Assume the coke to be 90 per cent fixed carbon, and the steel to be superheated 100° C. above its melting point.

Required: The net efficiency of the furnace.

Solution: The calorific power of the coke may be assumed as 90 per cent that of pure carbon, and therefore:

= 150 × (8100 × 0.90) = 150 × 7380 = 1,093,500 lb. Cal

Heat in steel at melting point:
110 × 300 (Ledebur) = 33,000

Heat to superheat 100°:
110 × 100 × 0.15 (assumed) = 1,650

Total 34,650 lb. Cal.

$$\text{Efficiency of furnace} = \frac{34{,}650}{1{,}093{,}500} = 0.032 = 3.2\%$$

Problem 10.

A Siemen's regenerative furnace holds eighteen steel crucibles, each containing 100 pounds of steel. Assume that the efficiency of utilization of the heat for melting the steel is 5 per cent, and that the furnace is fed by natural gas, having a calorific power of 512-pound Calories per cubic foot.

Required: The number of cubic feet of natural gas required per furnace heat of eighteen crucibles = 1800 pounds of cast steel.

Solution: Heat in steel = 1800 × 315 = 567,000 lb. Cal.

$$\text{Heating power of gas required} = \frac{567{,}000}{0.05} = 11{,}340{,}000 \text{ lb. Cal.}$$

$$\text{Cubic feet of gas required} = \frac{11{,}340{,}000}{512} = 22{,}150 \text{ cubic feet.}$$

Gas required per ton of steel, 2000 lbs. = 24,610 cubic feet.
Cost of gas, at $0.08 per 1000 cubic feet = $1.97.

Problem 11.

In a malleable-casting foundry the pig iron is melted in a reverberatory air furnace, 3000 kilos. being melted in two hours by the combustion of 1200 kilos. of bituminous coal, having a calorific power of 8500 Calories.

Required: The melting efficiency of the furnace.

Solution: Calorific power of coal used:

1200 × 8500 = 10,200,000 Calories.

Heat in melted iron at foundry heat:
3000 × 250 (Ledebur) = 750,000 Calories.

$$\text{Efficiency of furnace} = \frac{750{,}000}{10{,}200{,}000} = 0.0735 = 7.35\%$$

Problem 12.

In an iron foundry cupola 14 metric tons of pig iron are melted in one hour, using 1.5 tons of coke (90 per cent carbon). The gases passing away contain by volume CO 13 per cent, CO^2 13 per cent, nitrogen 76 per cent, and leave the cupola at 500° C. The body of the cupola is 1.5 meters in diameter outside and 4 meters high.

Required:

(1) The net melting efficiency of the cupola.
(2) The proportion of the calorific power of the coke lost.
(*a*) By the sensible heat of the hot gases escaping.
(*b*) By the imperfect combustion of the coke.
(*c*) By radiation from bottom and walls of the cupola.
(3) The amount of heat in Calories radiated, on an average, from each square meter of outside surface per minute.

Solution:

(1) Calorific power of the coke:

$$1500 \times 0.90 \times 8100 = 10{,}935{,}000 \text{ Calories.}$$

Heat in melted iron:

$$14{,}000 \times 250 \text{ (Ledebur)} = 3{,}500{,}000 \text{ Calories.}$$

$$\text{Efficiency of melting} = \frac{3{,}500{,}000}{10{,}935{,}000} = 0.32 = 32\%$$

(2, *a*) Weight of carbon escaping $= 1500 \times 0.90 = 1350$ kilos.

$$\text{Volume of CO and } CO^2 \text{ escaping} = \frac{1350}{0.54} = 2500 \text{ m}^3$$

(Because 1 m^3 of either gas carries 0.54 kilos. C.)

$$\text{Volume of escaping gases} = \frac{2500}{0.13 + 0.13} = 9615 \text{ m}^3$$

Volume of nitrogen (by difference) $= 7115 \text{ m}^3$

Sensible heat of nitrogen and CO

$$(7115 + 1250) \times [0.303\ (500) + 0.000027\ (500)^2] = 1{,}323{,}760 \text{ Cal.}$$

Sensible heat of CO^2

$$1250 \times [0.37\ (500) + 0.00022\ (500)^2] = 300{,}000 \text{ ''}$$

Total sensible heat in gases $= 1{,}623{,}760$ "

Proportion of calorific power of the fuel thus lost:

$$= \frac{1{,}623{,}760}{10{,}935{,}000} = 0.1485 = 14.85\%$$

(2, *b*) Volume of CO escaping $= 1250 \text{ m}^3$

Calorific power of this gas $= 1250 \times 3062 = 3{,}827{,}500$ Cal.

Proportion of calorific power of the fuel thus lost:

$$= \frac{3{,}827{,}500}{10{,}935{,}000} = 0.35 = 35\%$$

(2, *c*) Heat in pig iron = 3,500,000 Cal.
Heat in waste gases = 1,623,760 "
Lost by imperfect combustion = 3,827,500 "

Accounted for 8,951,260 "
Calorific power of the coke = 10,935,000 "

Difference, loss by radiation = 1,983,740 "

Proportion of calorific power of the fuel thus lost:

$$= \frac{1{,}983{,}740}{10{,}935{,}000} = 0.1815 = 18.15\%$$

(3) Area of bottom of cupola $= (1.5)^2 \times 0.7854 = 1.77 \text{ m}^2$
Area of sides of cupola $= 1.5 \times 3.14 \times 4 = 18.85 \text{ m}^2$

Total radiating area $= 20.62 \text{ m}^2$

$$\text{Heat radiated per sq. m. per hour} = \frac{1{,}983{,}740}{20.62} = 96{,}200 \text{ Cal.}$$

$$\text{Heat radiated per sq. m. per min.} = \frac{96{,}200}{60} = 1{,}603 \text{ ''}$$

In general, we may state that the net efficiency in melting, etc., of metals is very various, from, say, 2 or 3 per cent in a crucible steel-melting wind furnace to about 10 or 15 per cent in reverberatory furnaces, 20 to 30 per cent in regenerative open-hearth furnaces, 30 to 50 per cent in shaft furnaces, where material to be heated and fuel burned are in direct contact with each other, 50 to 75 per cent in steam boilers and hot-blast stoves, and 60 to 85 per cent in large electrical furnaces.

[The next instalment of these calculations will take up the thermophysics of compounds such as occur in metallurgical practice.]

The Electrochemistry of the Metallic Arc.

By Isador Ladoff.

The object of this article is to review the literature treating of the metallic arc, scattered on the pages of various scientific and technical publications in the United States and foreign countries, and relate concisely the results of my own work in this field.

Berthold Monasch's book, "Der Electrische Lichtbogen," etc., published in 1904, by the Berlin firm of Julius Springer, was a great help to me owing to the excellent bibliographical data furnished by it.

Davy found that the metallic arc is not as luminous as the carbon arc. Grove claimed in 1840 that various metals produce arcs of different luminosity, and arranged the metals accordingly in series, in which each foregoing metal is alleged to produce a more brilliant arc than the next following metal, namely, K, Na, Zn, Hg, Fe, Sn, Pb, Cu, Ag, Au, Pt. Later investigators doubted the correctness of Grove's series. Guye and Monasch, for instance (Drud. Ann. 1, p. 702, 1900), found that an arc produced by Bi-electrodes with a current of 0.06 amps. could not be maintained for any considerable length of time. Arons had a similar experience with zinc electrodes. Wurts (Ann. Inst. El. Eng. 9, p. 102, 1893) found that no permanent arc can be maintained by zinc, bismuth and antimony, probably due to the insulating properties of the oxides of these "non-arcing" metals. As a rule, metallic arcs are unsteady and flickering, the metallic electrodes producing the arc being easily softened or melt and oxidize in the air. De La Rive noticed the formation of brownish-red oxides by the iron arc at the usual pressure in the air and dark oxides in a rarified atmosphere. Gold (Berichte d. Wiener K. Acc. d. Wiss. 104, IIa, p. 815, 1895) and Arons (Wied. Ann. 57, p. 199, 1896) confirmed these observations. According to Guy and Monasch (Ecl. El. 34, p. 305; 35, p. 18, 1903) aluminium showed the highest degree of oxidation in the arc. Lead sends out into the arc a whitish blue vapor, condensing into a powder of the same color on the inner surface of the globe surrounding the arc. At the same time the products of oxidation settle on the pencil in colored circles; a dark circle of sub-oxide of lead and a yellow of lead oxide are followed by a circle of brownish red color. An antimon arc behaves in about the same way and manner as a lead arc. A cadmium arc does not produce any white vapors, but other products of oxidation. Magnesium electrodes, similar to aluminium, produce in the arc a thick crust of insulating earthy oxides. Nickel electrodes cover under the action of the arc with a film of oxides. Copper electrodes, between which an arc is maintained for some time by a current of 6 amps., cover with a thick crust of black oxide. However, when the current does not exceed 0.06 amps. only a thin layer of black oxide is formed. Guye and Monasch noticed a blackish sublimation even on silver and platinum electrodes after an arc was kept up between them for some time.

In connection with the behavior of different metallic electrodes in the air it will be instructive to study their behavior in other gases.

Arons (Drud. Ann. 1, p. 700, 1900) generated nitrogen by carefully heating a concentrated solution of $(NH_4)_2SO_4$ and $2NaNO_2$, and dried the gas in which he produced metallic arcs. The most remarkable phenomenon observed by him was that no lasting arc could be maintained between silver electrodes with low-tension currents while employing the usual pressure in an atmosphere of nitrogen. Arons explains this phenomenon by the small affinity existing between silver and nitrogen. He advances the theory, that in order to keep up the arc some chemical combinations between the material of the electrodes and the surrounding atmosphere are essential or necessary. I. Thomson concluded from certain phenomena produced by discharges in gases, that the formation of certain chemical combinations precedes the passage of electricity from the gases to metallic electrodes. Guye and Monasch produced an arc between silver electrodes with currents of high tension in an atmosphere of nitrogen containing about 3.50 per cent impurities. Owing to these impurities the experiment could not be pronounced as conclusive. Arons proved that some metals, as, for instance, aluminium and magnesium, produce nitrides in an atmosphere of nitrogen under the influence of the arc.

Grove succeeded, in 1840, to produce a carbon arc in an atmosphere of hydrogen, but failed to produce a lasting arc between metals in the same gas. Herwig (Pogg. Ann. 149, p. 523, 1873) and Stenger (Wied. Ann. 25, p. 31, 1885) repeated the experiment and confirmed Grove's conclusion. Arons used a current of 105 volts for the production of metallic arcs in hydrogen. He concluded that some metals do not arc at all in hydrogen, while others produce arcs only under special conditions. At a pressure exceeding 400 mm. Hg. column, no metals produce an arc in hydrogen.

Copper and aluminium easily producing an arc in nitrogen did not arc at all in hydrogen. Platinum and silver arcs demanded very high currents, endangering the electrodes. The same applied to iron, and especially to lead, due to the latter's melting point. Cadmium, zinc and magnesium showed in hydrogen the most favorable results.

Crew and Basquine (Proc. Am. Acad. 33, p. 335, 1897, 1898), who succeeded to produce in hydrogen arcs between iron, zinc and magnesium electrodes, call attention to the reducing effect of the gas on iron electrodes previously oxidized in the air.

It was noticed many times that it is easier to start an arc between metallic electrodes the readier they vaporize.

A metallic arc is more readily extinguished and harder to relight than a carbon arc. W. Duddell (The Elect. 46, p. 311, 1900) maintains an arc between copper pencils of 6 mm. diameter with a current of 3 amps. However, the arc was extinguished when put in parallel with a condensator of 0.6—5.4 mikrofarad. At 0.6 mikrofarad no extinction followed. When the copper pencils were replaced by cored carbons, the arc could be maintained with a current of 3 amps. in parallel with a capacity of 5.4 mikrofarad.

Le Roux (Comptes Rendus Hebdomedaires de Seanses de l'Academie des Siences, Paris 65, p. 1149, 1867) succeeded to start again a carbon arc which was kept extinguished 1/25 of a second by starting the current. However, Duddell (The Elect. 46, p. 311, 1900) found that it was impossible to relight an arc between copper electrodes after it was kept extinguished more than 1/27000 of a second by starting the current. Edlund (Prog. Ann. 134, p. 250, 1868) already noticed that he could not relight an arc between silver electrodes by simply turning on the current, while he succeeded to accomplish it with ease with carbon arcs. The heat conductivity of carbon is about thirty-seven times weaker than of the poorest conductor of heat among metals. The metallic arc consists of gases cooling down and solidifying rapidly after the extinction of the arc, while carbon, whose temperature of evaporation is higher than that of metals, produces in the arc gases which keep up longer their conductivity for electricity than metallic ones. The formation of insulating oxides likewise make the relighting of the metallic arc harder than the relighting of the carbon arc This explains the reason why it is easier to extinguish and harder to relight a metallic arc than a carbon arc.

The same explanation is applicable to metallic electrodes and their arcs at low-tension alternating currents.

At low-tension direct currents easily produce metallic arcs in most cases; however, this is not so with alternating currents. Zuchirstian (Sitzungsb. d. K. Acc. d. Wiss. in Wien 102, IIa., p. 567, 1893) ascribes the impossibility to produce an arc between metals at low-tension alternating currents to the high heat conductivity of metals, owing to which property the last rapidly cool down at each half period. Zuchristian produced an unsteady arc with carbon pencils of 17 mm. in length, inclosed in copper alloy holders. *He did not succeed at all to produce an arc between pencils of pure metals.* Sahulka (Sitz-b. d. K. Acc. d. Wiss. in Wien, 103, IIa, p. 925, 1894) confirmed Zuchristian's conclusions. Arons (Wied. Ann. 57, p. 185, 1896) tried to produce an arc between metallic electrodes, and expressed the results of his experiments as follows:

"I can affirm that it is impossible to produce an arc between two metallic electrodes by the means of an alternating current of 200 revolutions per second and at medium tension (which tension is from ten to twelve times larger than is necessary for the production of constant tension), although, when the metallic pencils were replaced by carbons of approximately the same dimensions, an arc of about 1 cm. length and high luminosity was easily produced. Blondel proved that the arc is extinguished at the end of each half period of the alternating current. The relighting is produced by a discharge emanating from the negative electrode. Arons arrived at the conclusion that it is necessary to employ high-tension alternating currents in order to produce a metallic arc.

The discontinuity of the arc between metals (as well as carbon arc) produced by a direct current seems to be established by many investigators. Lecher, for instance, proved in 1887 the discontinuity of the arc produced by a direct current between iron and platinum electrodes. Owing to the crude methods, not allowing the recording of very rapid changes in the arc, he, however, did not succeed to prove the discontinuity of the arc between silver and copper electrodes. Arons demonstrated (Wied. Ann. 58, p. 73, 1896) the discontinuity of the mercury arc.

The discontinuity of the metallic arc may, produced by a direct current, account, partially at least, for the unsteadiness of this arc on the same principles as it is thought to hinder the relighting of the alternating-current metallic arc.

Edlund (Pogg. Ann. 134, p. 250, 1868) connected silver electrodes with a galvanometer of high resistance 1/80 of a second after the arc between them was extinguished, and noticed no deflection of the needle of the galvanometer, while under exactly the same conditions carbon pencils produced a considerable deflection. Lecher (Sitzungsberichte d. Kais. Acc. d. Wiss. in Wien 95, IIa, p. 992, 1887) could not ascertain any difference of resistances between cathodes and anodes of platinum, iron, silver and copper. He found that the potential difference between a metallic electrode and any part of the gaseous arc is equal to about one-half of the potential difference between cathode and anode. As an explanation of this phenomenon he advanced the supposition that the difference of the metallic cathode and anode is not so pronounced as in the carbon cathode and anode.

(To be concluded.)

Notes on Electrochemistry and Metallurgy in Great Britain.

(From Our Special Correspondent.)

Institution of Mining and Metallurgy

Gold Metallurgy.

The Institution of Mining and Metallurgy at their meeting on Thursday, April 13, had five new papers before them for discussion. The first place was given to Mr. T. Kirke Rose's paper on *Refining Gold Bullion and Cyanide Precipitates with Oxygen Gas.* The paper was a lengthy one, the best summary being perhaps afforded by the author's conclusions.

"(1) It has now been shown that base metals can be removed from gold and silver bullion by means of a current of oxygen passed through them. (2) The losses due to volatilization and projection are insignificant, and those in the slag are moderate and can be kept low by stopping the process before the refining is complete. The initial losses in the slag, etc., including all beads, varied from 0.03 per cent of the values in Experiment No. 21, to 1.70 per cent in Experiment No. 18, when the slag was pasty, the mean being 0.73 per cent. Most of this could be brought to account at once by crushing and sieving. (3) The various impurities are equally harmless, provided that some readily oxidizable metal is left in the bullion to protect the silver. It is probable that copper, tellurium and bismuth have little action in protecting silver from oxidation. (4) Pure oxygen gas and air have the same effect, and there is little doubt that, if a deep body of metal were used, their efficiency would be, at the lowest, 80 or 90 per cent. (5) The fluxes required for fusion are borax and silica, and these are sufficient. The slag formula 2-3 ($Na_2\,O\,B_2\,O_3$) + 6RO 2-3 ($B_2\,O_3$) $3SiO_2$ appears to fulfill all requirements, except that of cheapness. The cost of the fluxes in such a slag would be about ½d. per ounce of fine gold recovered. The proportions indicated by the formula may undergo considerable variations without any disadvantage. On the other hand, when 20 per cent of both the borax and the silica are omitted, the slag becomes somewhat pasty. The replacement of even 90 per cent of the borax by sand appears to be more than a possibility, and would greatly reduce the cost of the fluxes. The addition of iron oxide or lime may be useful if borax is omitted. (6) The gold can in great part be recovered from the slag by concentration, but the silver cannot be recovered in that way. Almost all the values can be recovered by fusion with carbon and iron, the lead and copper being reduced and carrying down the gold and silver. (7) In the treatment of zinc-box precipitate from the cyanide process, it is convenient to drive off part of the zinc by volatilization in order to save fluxes and to avoid pasty slags. (8) The metals are oxidized in succession, each in turn partially protecting the metals which are less easily oxidized than itself. Owing to mass-action, this protection is not absolute. Silver is not easily oxidized, and gold is protected by all other commonly occurring metals. (9) There is *prima facie* evidence that the cost of treatment by air is less than that of other refining processes. The relative losses are more doubtful, and cannot be determined in the laboratory. The total losses observed in the experiments, including the values left in the slag tailings, varied from 0.008d. to 2.4d. per ounce of fine gold recovered, the mean loss being 1.36d. On a large scale these should be much less."

VARIOUS PAPERS.

The second paper to be discussed dealt with *Wood Gas for Power Purposes and Gas Generators.* In it Mr. George M. Douglas described the gas-driven power house of the Moctezuma Copper Company, at Nacozari, Sonara, Mexico. (See our Vol. II, p. 104.) Coke is used on setting away the gas producers, after which wood fuel, consisting mostly of an inferior scrubby oak. An average sample of the wood gas contains 19.5 per cent H, 13.45 per cent CO, 2.48 per cent CH_4 15.45 per cent CO_2, 48.5 per cent h, 0.34 per cent $C_n\,H_{2n}$ and 0.25 per cent O. Its calorific value is 135.6 B. T. U. per cubic foot. Under normal working conditions the average is about 2.5 pounds per B. H. P. hour, and 0.11-pound coke.

The third and fourth papers may be almost described as *Prospectors Reports.* In the third, Mr. Philip Poore supplies some notes on the Prestea District, Gold Coast Colony, while Mr. R. O. Ahlers, in the fourth, deals with the new Dharwar Gold Fields. They are both partly geological, partly antiquarian, and the former is partly economic. They both form valuable contributions to the bibliography of mining districts, but do not call for other comment in these pages.

The fifth, and last, paper was purely geological, being entitled *The Cause of Border Segregation in Some Igneous Magmas.* The author, Mr. James Park, ascribes border segregation "mainly to differences of osmotic pressure in the magma, with perhaps convection currents as a contributing cause." It is suggested that if a molten magma be regarded as a mass of rock material in solid solution, the first crop of minerals separated by cooling being inert and unable to offer ionic resistance will be carried by the transference of osmotic energy from the interior to the borders which form the coolest parts of the magma. "Manifestly this unequal struggle between different potentials of osmotic energy will continue so long as the difference of pressure exists, but a point will be reached where the pressure will be neutralized by the increasing viscosity of the magma. The minerals which first crystallize are generally basic, and these being carried outward as they form, necessarily leave the interior more silicious, or acid than the borders." Finally it is suggested that molecular concentration may be the result of differences of osmotic pressure due to unequal temperature in a homogeneous solution.

REFRACTORY AURIFEROUS SULPHIDES.

The most important of five papers which were presented at the May meeting of the Institution of Mining and Metallurgy was that by Mr. F. B. Stephens, on "The Treatment of Refractory Auriferous Sulphides at the Cassilis Mine, Victoria, Australia." The surface ore in this district is very rich, giving as much as 10 ounces to the ton, but this proportion falls rapidly with an increase in depth, so that improved methods of extraction alone permitted work to be continued in depth. The main body of ore comprises a band of dense sulphides, consisting of arsenical pyrites, iron pyrites, zinc blende, galena and carbonate of iron, whilst antimony, manganese, copper and tin occur in proportions usually of less than 1 per cent each. The gold values range from 1 to 10 ounces, averaging from 3 to 5 ounces. The plant used for treatment consists of a jaw crusher, 16 inches by 9 inches, self feeders, twenty head of 1000-pound stamps, silvered copper plates, four three-cone hydraulic classifiers, six Wilfley tables, and four Berdan grinding pans. Tailings go to settling tanks for future cyaniding, and concentrates are roasted for chlorination, whilst the galena is shipped to smelters after grinding. A drawback to chlorination in this district has always consisted in the large percentage of galena carried by most of the ores, and copper is also occasionally troublesome. Zinc blende, after roasting, is quite amenable to chlorination, our concentrates sometimes carrying 20 per cent of zinc. The galena is separated on the Wilfley tables and subjected to special treatment, thereby getting over the trouble in the furnaces due to fusion of the sulphides, a matter of some consequence with galena, running sometimes as high as 7 per cent on the ore crushed. The ore is crushed to pass a 17 by 20-mesh screen, and after crushing, the pulp is run over silver-plated copper plates, given a fall of 1½ inches per foot, to keep the plates clear of fine galena. The amalgam yields a return of one-fifth to one-seventh of smelted gold. Before the employment of classifiers great losses of gold and mercury took place, but since their introduction the gold and mercury is caught in the galena streak on the Wilfleys. Only 20 per cent of the total gold is obtained by amalgamation. With the adoption of classifiers no occasion was found for use of Raff wheels, which were thrown out. At the end of the tables an inner sliding box was arranged in the usual concentrates box, for the purpose of catching the galena concentrates. These concentrates contained a quantity of arsenical pyrites, and were daily reconcentrated on a separate table reserved for this purpose, and three products were made. Of these, the heads containing 40 per cent of lead went to the grinding pans; the middlings, containing 15 to 20 per cent of lead, 15 to 20 per cent of arsenic, and 8 ounces of gold per ton, were shipped direct to the smelters, and the thirds went back to the ordinary concentrates for chlorination. Smelters, by the way, paid for gold, silver and lead, but deducted 1s. per unit for arsenic. The galena headings were ground in Berdan pan with mercury and a little cyanide for the period of twelve hours or less, and the ground pulp, when removed from the pans, was stirred in a tub and allowed to settle, when the top sludge was removed, put into jute bags to drain, and shipped to smelters. Settlings at the bottom of the tub, after removal of the bulk of the mercury and amalgam, were returned to the next charge, and in this way the mercury loss was kept at a minimum, averaging for all operations at the mill from .75 to 1.05 ounce per ton crushed. The lead sludge from the grinding carried about 15 ounces gold, 40 per cent lead and 10 per cent arsenic when shipped; the gold value before grinding being sometimes as high as 60 ounces. The stamps were run at 96 to 100 drops per minute, with a 6-7 inch drop.

As it was usual to make from 15 to 20 per cent of concentrates, running from 3 to 4 ounces of gold, and containing over 50 per cent of the total gold value of the ore crushed, the chlorination work had to be run as cheaply as possible.

It was found that short hand-rabbled reverberatories were quite unsuited for the work, on account of the concentrates fusing too easily. As the galena could not all be separated some form of mechanical furnace was absolutely necessary. Chlorine solutions generated from bleaching powder and sulphuric acid were used in 10-ton open wooden vats, and after comparative trials against gas, this method was retained as offering many advantages, being actually cheaper for this ore, while the extractions were the same for both.

The chlorine solutions run on had a strength of .09 to 1.2 per cent of chlorine, and from .5 to 1 per cent of sulphuric acid over and above the quantity required to combine with the bleaching powder. In this way many deleterious compounds were leached out or rendered harmless, most of the copper present being leached out in the first solution, leaving the ore in a good condition for the next solution. The first solution usually had all its chlorine used up, but on well-roasted ore the second solution ran off, showing chlorine freely. The solutions were not allowed to remain long in contact with the ore, as otherwise the chlorine became used up and the gold already dissolved was precipitated in the vat. Solutions were run on for six days, or until the off-flowing solutions failed to react well for gold. The solutions were allowed to settle in an intermediate vat before precipitation. This vat was fitted at plugs at every 6 inches for drawing off small portions for assay, and an extra check on the amount of gold due at the clean-up was obtained. Ferrous sulphate was used as a precipitant, and was made from old battery screens. A considerable amount of basic iron sulphate, etc., settled in the precipitating vats, and at clean-up time was taken into solution with sulphuric acid. Copper gave very little trouble, but it seldom went over 1 or 2 per cent in the concentrates. Owing to the use of such weak solutions not much chlorine passed into the atmosphere. Less chlorine was consumed than if dry gas had been used on this ore. An extraction of about 85 per cent was obtained on well-roasted ore, the loss amounting to 1 to 2 dwt. per ton crushed. Shipping to smelters and treatment charges amounted to £6 per ton, whilst the costs, plus the value of gold left in, did not amount to half that. The drawback to the use of 10-ton vats was the fact that a little carelessness on the part of a furnaceman spoilt the extraction of a whole vat.

Cyanide works were erected to treat the accumulated tailings, but much difficulty arose in leaching these owing to the large percentage of talc and mica; further, the slimes running high in concentrates gave rise to clogging, and the solutions could not be allowed to remain in contact with the sands. Solutions which ran off quite clear, on standing became quite milky with magnesia hydrate, and this precipitate settled thickly on the zinc and spoiled its efficiency. Extractions ranged between 60 and 65 per cent, and the consumption of cyanide kept at the moderate amount of 75 pounds. A treatment of seventy-two hours was given, but a longer treatment gave no better extractions. The use of an oxidizing agent in experiments increased the extractions 10 per cent. The water supply destroyed permanganate readily; organic matter in the water may have accounted for this, as the water came from peat bogs. If the ore were leached with distilled water a heavy consumption of permanganate followed on the magnesium sulphate, which seems rather strange.

For assaying the cyanide solutions an electrolytic method is employed. For a weekly output of 270 tons of ore milled and mined, 54 tons of concentrates are chlorinated, 300 tons tailings are cyanided, and four tons of galena are shipped for smelting. The cost per ton of ore crushed is stated to be £1.8.0 (excluding directors' fees and Melbourne office expenses). When it is remembered that 27 to 30 per cent bleaching powder cost £18 per ton, chamber sulphuric acid £11 per ton, and wood—the only fuel available—cost from 14s. to 18s. per cord of 125 cubic feet, and that explosives varied from £3.2.6 to £4.2.6 per case, it will be realized that a cost of $7 per ton of ore mined is very satisfactory.

Of the remaining papers none calls for abstracting, although each was of interest. Mr. F. J. Stephens' "Notes on a Low-Grade Copper Ore Deposit in the Himalayas," does not betoken any prospect of the immediate cheapening of this metal. Containing only 2 per cent of copper, poor transport facilities are likely to defer its development, "notwithstanding the undoubted mining facilities which exist, such as abundant and cheap timber, and some of the cheapest labor in the world." Mr. H. Brelich's "Chinese Methods of Mining Quicksilver" is an acount of what we should call in England a threatened and decaying industry once worked on a much larger scale. Yet threatened industries—like threatened men—live long, and Mr. Brelich anticipates the maintaining of a constant output for many years to come. Much of what the author says respecting the peculiar habits of the Chinese miner would, if the locale were the Transvaal, make an admirable anti-Chinese labor pamphlet. Mr. A. N. Mackay, in his paper, "Reprecipitation of Gold from Cyanide Solutions and Absorption of Gold by a Wooden Leaching Vat," deals with two different subjects, the one due to the use of too alkaline a solution with a too slow percolation rate, the latter showing that the red pinewood of a leaching vat absorbed gold to the extent of 12 ounces 8 dwts. per ton of wood. Lastly, Mr. H. R. Sleeman's account of an Egyptian gold mine describes some abandoned ancient workings now being dealt with by modern methods.

THE ROYAL SOCIETY'S CONVERSAZIONE.

The Royal Society's Conversazione held on May 17 was chiefly interesting by reason of the exhibits of scientific value which excited much attention. On the metallurgical side, mention must be made of a series of microscopic slides and photographs illustrative of an ingenious method of detecting the presence of phosphorus in steel, exhibited by Mr. J. E. Stead, who also showed an apparatus for polishing and preparing metals for microscopic examination.

Dr. Glazebrook showed some high temperature furnaces constructed of rare earths, such as are used in Nernst lamps. They are available for temperatures between 800° C. and 2000° C., and one had recently been used to determine the melting point of platinum.

Mr. W. Rosenhain showed some transverse sections of slip bands and other microscopic features of metallic surfaces. The surfaces to be examined in section are first coated with a thick electrolytic deposit of copper or other metal, and a section is then cut through the compound specimen. The line of contact between the original metal and the electro-deposit is perfectly sharp under the highest magnification, and reveals what is claimed to be an accurate sectional view of the original surface.

Mr. R. S. Hutton exhibited new models of laboratory electric furnaces which were interesting. The furnaces consisted of carbon tubes or rods heated by the electric current. The method employed for conveying the current to the carbon by soldering water-jacketed sleeves to the electro-coppered ends of the carbon formed a novel feature of the construction. As will be remembered, this tube type of furnace formed the subject of a paper read before the Faraday Society.

The Faraday Society.

First among London scientific societies to start their session in October last, the Faraday Society has been the last to break up for a vacation. While other societies have held their conversaziones in their own halls, as the Institutions of Civil and Mechanical Engineers, or have gathered around cases of stuffed birds and the vertebra of the *Diplodocus Carnegii* at South Kensington, as did the Institution of Electrical Engineers, or have held a garden party in the Royal Botanical Gardens, like the Society of Arts, the members of the Faraday, eschewing the formal pursuit of pleasure, broke all their own records with a four-paper night on July 3 last. The following was the order of procedure: Mr. Sherard Cowper-Coles, "Some Notes on the Rapid Electrodeposition of Copper." 2. Prof. W. W. Haldane Gee, "The Use of Balanced Electrodes." 3. Messrs. H. D. Law and F. Mollwo Perkin, (*a*) "Electrolytic Oxidation of Hydrocarbons of the Benzene Series." Part II., "Ethyl Benzene, Cumene and Cymene"; (*b*) "Electrolytic Analysis of Antimony." 4. Messrs. R. S. Hutton and J. R. Beard, "Notes on Heat Insulation, Particularly with Regard to Materials Used in Furnace Construction." Nominally, a four-paper night, it will be observed that the two sections of Mr. Mollwo Perkin's and H. D. Law's papers are on entirely different subjects. There was a good attendance of members and the papers were well discussed. As to what was said, I propose to defer my notes until next month. Although societies may temporarily cease from meeting, the publication of technical journals and the duties of their correspondents do not cease. I, therefore, reserve my notes and comments for the time when certain of my sources of news and matter run dry.

Market Quotations.

Nothing particular has to be reported with regard to the prices for chemical substances, save that copper sulphate has receded to £19.10.0 per ton, and shellac is again cheaper, at £7.5.0 to £7.7.0 per cwt.

After the wild gambling of May, Cleveland pig iron has steadied down to £2.5.6 per ton, but the outlook in the iron and steel industries being promising, a rise is expected. Copper was fairly quiet, very little speculative buying being indulged in. The closing price was £66 per ton. Tin has risen to £139.12.6 per ton, on account of lower shipments from the Straits. Aluminium ingots are listed at £130 per ton, aluminium wire and sheet being quoted respectively at £168 and £166 per ton. English lead has been fairly steady, improving to £14 per ton towards the end of the month for ingots, and to about £15 for sheets. Quicksilver is unchanged, but zinc has arisen £1 to £27.7.6 per ton. I have also to chronicle a rise in the price of platinum to £4.4.0 per ounce.

ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS.

By George P. Scholl, Ph. D.

Electric Furnaces and Furnace Products.

Process of Treating Phosphate Rock.—F. J. Machalske, Brooklyn. Patent 789,438, May 9, 1905. Application filed Feb. 25, 1905.

The process comprises the crushing of rock, containing tricalcium phosphate, especially low rock, with less than 70 and down to 50 per cent of phosphate, and the subsequent mixing of the crushed product with an alkali metal chloride of carbon, after which the mixture is heated to a temperature sufficient to decompose the rock and to effect the production of phosphoric chloride and calcium and sodium carbides. The preferred mixture consists of the crushed rock, sodium chloride and an excess of carbon in the form of broken coke. The smelting is preferably effected in an electric furnace; the charge should be free from water, and air is excluded from the furnace. The heating may be done in an electric furnace, either of the arc or the resistance type. A current, preferably alternating, of from 2000 amps at 50 volts to 3500 amps. at 40 volts, is stated to be sufficient for a furnace of the usual size. The phosphoric chloride is led off from the furnace, and may be treated with water for the production of phosphoric and hydrochloric acids. The calcium-sodium carbonate may be treated with water for the production of acetylene, calcium hydroxide and sodium hydroxide.

Process of Treating Phosphate Rock and Producing Compounds of Phosphorus and Nitrogen.—F. J. Machalske, Brooklyn. Patent 789,439, May 9, 1905. Application filed Feb. 25, 1905.

The initial step of the process consists in the crushing of the low-grade material as above and raising it with alkaline chloride and carbon. The mixture is then smelted at a temperature sufficient to decompose the rock and to effect the production of phosphoric chloride, the smelting being preferably carried out in an electric furnace. Nitrogen or air, or another gas containing nitrogen, is then blown through or brought in contact with the charge in the melting furnace, and the calcium and sodium combine with the carbon and nitrogen to produce cyanamides. The production of the cyanamides is stated to be presumably effected in two stages, the first being the production of the carbides, which then, in the second stage, combine with the nitrogen. The phosphoric chloride is treated as above, while the calcium-sodium cyanamides may be treated with water for the production of calcium and sodium carbonates and ammonia. The cyanamides may also be mixed with sodium carbonate and smelted in an electric furnace for the production of sodium cyanide.

Process of Treating Phosphate Rock and Producing Phosphorus Chlorides and Alkali Metal Cyanides.—F. J. Machalske, Brooklyn. Patent 789,440, May 9, 1905. Application filed Feb. 25, 1905.

The process comprises the same steps as the preceding one, resulting in the formation of calcium-sodium cyanamides. The latter are smelted in an electric furnace, either of the arc or the resistance type, with sodium carbonate. An alternating current is preferable, of from 800 to 1000 amps. at 50 volts. The sodium carbonate is preferably produced by treating another portion of the calcium-sodium cyanamide with water.

Electrolytic Production of Metals and Compounds.

Centrifugal Apparatus for Electrolytic Purposes.—R. V. Hauser, Erie, Pa. Patent 790,055, May 16, 1905. Application filed Sept. 17, 1904.

The apparatus is to be used particularly for the manufacture of persulphate. It consists essentially of a revoluble drum, provided with an anode of lead, corrugated and perforated. The anode is parallel to the outside wall of the drum, a space being, however, left between them. The cathode is preferably in the form of a cylinder provided with holes. Sulphuric acid is supplied by a pipe into the space enclosed by the cylindrical cathode, while an aqueous solution of ferrous sulphate is fed to the space outside the cathode cylinder. These liquids are claimed not to mix, on account of their different densities. The electric current causes the liberation of SO_4 in its nascent state at the anode, and transforms persulphate of iron, $Fe_2(SO_4)_3$. This substance has a comparatively high specific gravity, and when the apparatus is rotated the centrifugal force tends to force it outwards through the holes, and it eventually finds its way to the discharge pipe. In the course of action the anode becomes coated with peroxide of lead, which latter, it is claimed, improves the action.

Electrolytic Apparatus.—F. A. Decker, Philadelphia. Patent 789,721, May 16, 1905. Application filed Feb. 29, 1904.

The invention is stated to relate to electrolytic apparatus designed for the recovery of zinc, magnesium or other apparatus from acid or neutral solutions, such as spent battery fluids and solutions obtained in the reduction of metal from its ores or scrap. The apparatus is divided into two compartments by means of a porous diaphragm. The positive electrode of the cell may be of carbon, peroxide of lead, or other suitable material. The negative electrode is formed by a series of trays of the usual material, which trays are arranged one above the other. A tube extends from the bottom to the top of each tray, thus providing a passage through the trays for the better circulation of the gas and the fluid contained in the compartment.

Apparatus for Obtaining Oxides of Alkali Metals.—C. W. Roepper, Philadelphia, and W. E. Harmon, Mechanic Falls, Me. Patent 790,922, May 30, 1905. Application filed July 22, 1903.

The apparatus is of the so-called mercury type, the circulation of the mercury being effected by a wheel. The patentees have found that the so-called "discharging surface," that is, the place where both the amalgam and water are in joint contact with a conducting surface, and where, therefore, the alkaline element, which has been taken up by the mercury, shows the greatest tendency to disassociate itself from the amalgam, is most usefully provided in the wheel itself. The latter, or at least that part of it which comes into contact with the amalgam, is therefore constructed of iron or nickel or carbon, and in such a manner as to increase the extent of the contact of the amalgam with the discarging surfaces. The slots, or pockets, in the wheel are, therefore, more or less filled with granules of electrically conductive material such as lumps of iron or carbon. The periphery of the wheel is surrounded by wire netting, so as to keep this material in the pockets. One of the forms shows the pockets sub-divided into numerous smaller compartments by wire plates or perforated metal plates forming circumferential flanges. The actual material of the body of the wheel is comparatively immaterial, provided the surfaces which come in joint contact with the water and amalgam are made of electrically-conductive material.

Process of Manufacturing Ammonia.—W. Hoops, Pittsburg. Patent 791,194, May 30, 1905. Application filed Dec. 14, 1904.

The patentee has discovered that when aqueous solutions of ammonium sulphate and another salt are electrolyzed in respectively opposite sides of a porous diaphragm, in a cell having a cathode of carbon, iron or another insoluble material, ammonia is set free at the cathode and sulphuric acid at the anode, and that the compound formed by the liberated acid may then be utilized. The operation is carried out in a two-compartment cell, one containing an iron cathode and the other a carbon anode. The cathode compartment is filled with a solution of ammonium sulphate, and the anode compartment with a solution of common salt. The whole may be heated to about 100° C. The ammonia, in whole or in part, is stated to go off from the cathode compartment with the free hydrogen, from which it may be dissolved by passing it through water, in which it dissolves. The hydrogen is collected and may be used for producing hydrochloric acid with the chlorine liberated at the anode. Ammonium sulphate and sodium chloride are added to the bath from time to time, to replace that which is lost. No further data concerning particulars of electrolysis, current density, etc., are given.

Electrode.—Mataichi, Yasuda, Tokio, Japan. Patent 791,308, May 30, 1905. Application filed Jan. 21, 1905.

The construction provides for carrying off the hydrogen bubbles generated in the electrolysis of alkaline chloride, being so arranged as to permit the entrance of the electrolyte but not the escape of bubbles. The electrode consists of an outer casing of porcelain, glass or other insulating material. On one side is a partition, also of insulating material, forming a passage. The electrodes consist of two plates, one of the plates being rigidly secured to the inner face of the wall, and the other member of the electrode is preferably disposed within the space secured by an offset. The action of the hydrogen bubbles in carrying the solution with them effects a circulation of the electrolyte.

Electrolytic Deposition of Metals.—N. C. Harrison, London, England, and J. Day, Westonsuper Mare, England. Patent 791,341, May 30, 1905. Application filed July 19, 1901.

The invention provides for the use of concentrated solutions and high-current densities in the deposition of metals. This is accomplished by renewing the solution in contact with the cathodes continuously. For carrying out the invention a highly polished cathode of circular cross-section is used, *f. i.*, a cylindrical or conical cathode, mounted vertically in the depositing vat and free to revolve on its own axis. The layer of electrolyte contiguous to the cathode is constantly renewed by means of jets of electrolyte, which are directed tangentially against the cathode. As shown in Fig. 1 the apparatus contains a cathode cylinder x, located in vat a. A series of vertical pipes e e are disposed equidistant from each other and concentrically around the cathode x. These vertical pipes are closed at their upper ends, but provided with holes at frequent intervals, in such a manner that a tangential stream will be directed on the surface of the cathode. It has been found in practice that copper can be successfully deposited on a cathode of about 3 inches diameter, by employing vertical pipes located at or about 2½ inches from the center of the cathode, and spaced apart about 2½ inches from each other concentrically around the cathode. The holes were 3-32 inch, and were bored at intervals of ½ inch.

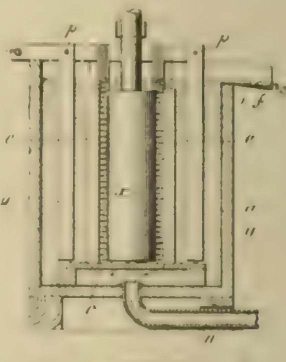

FIG. 1.—DEPOSITION OF METALS.

Process of Extracting Zinc from Its Ores.—A. G. Betts, Troy, N. Y. Patent 791,401, May 30, 1905. Application filed Feb. 20, 1905.

The process consists in first bringing the zinc into solution, for which any of the usual methods may be used, the inventor, however, prefering to use sulphuric acid in lixiviating, thus producing a solution of zinc sulphate. This is electrolyzed with an insoluble anode and with a liquid metal cathode, preferably of mercury or mercury containing zinc. Oxygen is produced at the anode and sulphuric acid remains in solution. Practically all the zinc is removed and the solution of sulphuric acid thus obtained may be used for lixiviating, thus producing a solution of zinc sulphate. This is electrolyzed with an insoluble anode and with a liquid metal cathode, preferably of mercury or mercury containing zinc. Oxygen is produced at the anode and sulphuric acid remains in solution. Practically all the zinc is removed, and the solution of sulphuric acid thus obtained may be used for lixiviating further amounts of ore. The zinc mercury alloy, which preferably contains less than 5 per cent of zinc, is then removed to another electrolytic cell, containing a solution of a zinc salt, preferably one of zinc chloride. Into this solution a zinc cathode is introduced, while the mercury-zinc alloy is connected as anode. An electric current is then passed, whereby the zinc is dissolved from the anode and deposited on the cathode. The mercury, after the removal of a large part of the zinc, is returned, to go through the same process over again.

Electroplating Apparatus.—C. T. Pratt, Frankfort, N. Y. Patent 790,747, May 23, 1905. Application filed Oct. 3, 1903.

The apparatus is intended for plating the inside of ferrules, rings, bands, slip nipples, etc. The articles are strung on the anode bar, which is then inserted in the galvanizing solution. They rest upon a series of V-shaped troughs, along the inner walls of which extend one or more metallic strips. The latter form the connection with the negative terminal of the current, and thus constitute the articles to be plated cathodes. When the current is passed the inside of the hollow articles will be covered with a deposit.

Process of Stripping Tin.—D. W. Hemingway, Walthamstow, England. Patent 791,555, June 6, 1905. Application filed Oct. 29, 1902.

The process depends upon the use of a solution of persulphate of iron, which is contained in the tank A in Fig. 2. The bath is acidified by the addition of sulphuric acid. The tinned iron clippings from which the tin is to be removed are immersed in this solution. After the tin has become dissolved a valve *a* in the pipe *a'* is opened and the solution drawn into the electrolytic vessel B, where it is electrolyzed between iron anodes and copper or tinned copper cathodes. When the greater portion of the tin has been deposited upon the cathode, the electrolyte passes by way of the valve *b* and pipe *b'* into the vessel D. In this vessel the solution is restored by the addition of nitrate of soda or potash, either alone or with sulphuric acid. Other oxidizing agents may be substituted for this mixture. The vessel D has an outlet F for the nitrous vapors, and is surounded by a jacket M, for the purpose of heating its contents. After the electrolyte has become sufficiently oxidized, the liquor is withdrawn through valve *d* and pipe *d'* into the vessel L, where pump N transfers it back into the vessel A. When the quantity of persulphate of iron employed is to be reduced to a minimum, the tin scrap may be added directly to a solution of nitrate of soda, or potash, mixed with sulphuric acid, from which the tin is then recovered by electrolytic or other means.

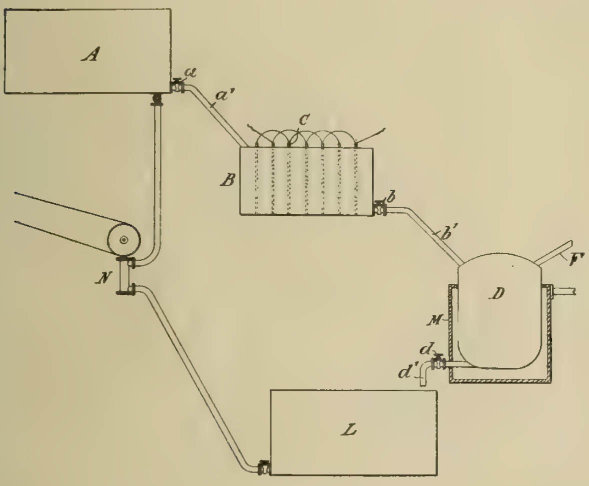

FIG. 2.—STRIPPING TIN.

Method of Making White Lead.—C. P. Townsend, Washington, D. C. Patent 791,956, June 6, 1905. Application filed March 22, 1904.

The process is carried out in an apparatus divided into two compartments by a suitable diaphragm. As shown in Fig. 3 it consists of a wooden vessel 1 lined with sheet lead, the lining being treated with a solution of paraffine. The diaphragm 2 is intended to prevent the compounds of lead present in the solution contained in the anode compartment, the so-called anolyte, from coming into contact with the cathode. Vegetable parchment, or parchment paper, is particularly suited for this purpose, and it is believed to be essential that the membrane should be substantially impermeable as a filter, as no useful results could be obtained by the use of tissues or woven fabrics. The diaphragm divides the vessel into two unequal compartments, namely, a relatively large anode compartment 3, and a relatively small cathode compartment 4. The anode 5 is composed of lead, while the cathode 6 may be a sheet or mesh of any suitable material. The white lead is formed in the anode compartment and is continually drawn off by a pipe 7 to a filter press 8 or another suitable separator. The filtered electrolyte passes into a tank 9, whence it returns to the electrolytic vessel by means of a pump and pipe 10. Carbon dioxide is introduced either directly by a pipe 11 into the anodic solution, or it may be introduced at any point in its circuit. As the cathodic solution apparently does not take part in the reaction by which the pigment is produced, the cathode compartment may be very small as compared with the anode compartment, or may be such that the diaphragm is in substantial contact with the face of the cathode. A pipe 12 serves for washing the pigment collected in the filter press, and a sufficient amount of the first washings should be added to the electrolyte in order to maintain its original volume. The process is not limited to any particular electrolyte, the essential requirement being that it should comprise a mixture of salts in solution, one of them being capable of yielding a solvent for lead at the anode, while the other must necessarily be a soluble carbonate or bicarbonate. As a solvent salt, preferably one of the soluble acetates, such as sodium acetate, is used. The cathodic solution may have the same initial composition as the anodic, but may initially consist of water or of any solution not incompatible with the operation.

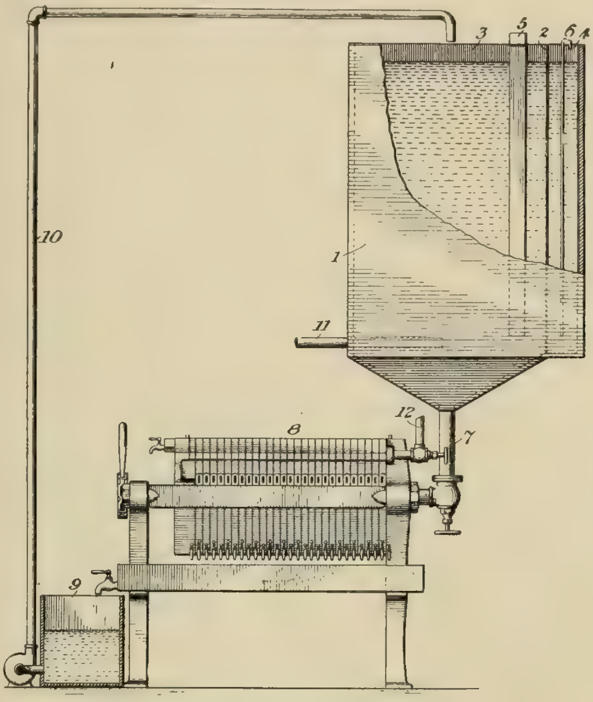

FIG. 3.—PRODUCTION OF WHITE LEAD.

Process of Electrodepositing Antimony.—A. G. Betts, Troy, N. Y. Patent 792,307. June 13, 1905. Application filed May 20, 1904.

The process, evidently intended to be used in the refining of electrolytic slimes, as described by Mr. Betts in ELECTROCHEMICAL AND METALLURGICAL INDUSTRY, Vol. III., p. 141, depends upon the use of an electrolyte consisting of antimony trifluoride. As this compound is not decomposed by water into basic antimony compounds and free acid, a solution can be prepared which is only slightly acid. In addition, there are preferably metallic salts present, which will not deposit on the cathode with the antimony, and alkali sulphates and fluorides are found to be most suitable for this purpose. In case an antimony anode is being used, alkali chlorides may be employed. The presence of sulphuric acid in the electrolyte is stated to be desirable, a suitable solution containing 8 per cent SbF_3, 2 per cent Na_2SO_4, and 4 per cent H_2SO_4. A current density of 10 amps. per square foot of anode surface is stated to give a good deposit of antimony when the solution is pure, *i. e.*, free from metals which would deposit with the antimony. By electrolyzing such a solution, with an insoluble anode of lead, oxygen escapes at the anode, leaving a solution containing hydrofluoric acid. The anode should consist of lead rods, surrounded by a layer of porous material, which is intended to prevent the tendency of the antimony to become oxidized to the pentavalent form at the anode, in which form it has a dissolving action on the metal at the cathode, where it is again introduced to the trivalent compound. Such a layer of porous material consists preferably of a few layers of cloth surrounding each anode rod. If the cloth is absent, resolution is stated to occur to such an extent that the actual yield of metal deposited at the cathode is reduced to 60 per cent of the theoretical, while the yield rises to from 90 to 95 per cent with anodes wrapped in cloth. The area of the anode should not be over half that of the cathode, inasmuch as with the higher anode, current density to formation of pentavalent antimony compounds is reduced. The voltage required, when insoluble anodes are used, is given as usually from 2.5 to 3 volts. As hydrofluoric acid is a good solvent for antimony oxides and basic compounds, the solution, after electrolysis, is in suitable condition for dissolving fresh amounts of basic antimony compounds from which it is desired to prepare metallic antimony.

Process of Treating the Metal Mixtures Produced as a By-Product in Electrolytic Metal Refining.—A. G. Betts, Troy, N. Y. Patent 793,039. June 20, 1905. Application filed May 20, 1904.

The process provides for the treatment of anode slimes, usually containing free or combined lead, copper, arsenic, antimony, silver and gold, with less important quantities of other elements, as applied to the treatment of such slimes, from the electrolytic refining of lead, which consist mostly of metallic lead, copper, arsenic, antimony and silver, it comprises as the first step the agitation of the slime, in a lead-lined tank, with a hot solution of ferric sulphate containing, say, 2 per cent of free sulphuric acid and 4 per cent of iron as ferric sulphate, an excess of the latter salt being present above that required to carry out the reactions. During this operation the copper goes into solution as copper sulphate, and the iron is reduced to the ferrous condition, while lead forms $PbSO_4$, arsenic As_2O_3 + free H_2SO_4, antimony Sb_2O_3 + free H_2SO_4, and bismuth $Bi_2O_3 \cdot H_2SO_4$ + free H_2SO_4. The mixture is then allowed to settle, and the solution drawn off from the residue, which latter is washed by decantation with hot water. The wash water is run over copper so as to precipitate the silver, after which it passes over iron in order to deposit the copper, and is then run to waste. The solution drawn off from the residue is agitated several hours in contact with metallic copper in order to precipitate the silver. The free acid formed during solution of the slimes is neutralized by the addition of copper oxide to the solution. Roasted matte may be used instead, but this is added to the solution when the iron is in the ferric state, that is, before it has been reduced by its action on slime, inasmuch as its acidity is then less and less iron is dissolved from the matte, while the ferric sulphate assists in the solution of any cuprous oxide present. The acidity of the solution is advantageously as low as possible when roasted matte is used. The solution, after treatment to precipitate the silver as described above, is first passed through the cathode compartments of an electrolytic cell, which is provided with copper cathodes, on which latter it deposits the copper which it contains. It then passes through the anode compartments of the cell, by which operation the ferrous sulphate is oxidized again to ferric sulphate and the solution thus regenerated. The arsenic of the slime accumlates in the solution and is kept from reaching too high a concentration by either withdrawing a part of the solution at a convenient stage of the process and cooling it, in order to crystallize out the arsenious acid, or by removing it by chemical methods, or by an occasional purification of the entire solution. The process of reoxidation can also be carried out by other means than electrolysis, for instance, by the use of nitric acid. In this case the too great accumulation of copper and arsenic are prevented by crystallizing them out as copper salt and arsenious acid. Nor is there any limitation as to the use of iron as the carrier of oxygen from the oxidizing tank to the slimes-treatment tank, inasmuch as chromium or manganese compounds in the form of permanganic or chromic acids may be used. They may be reduced by the action on the metals of the slime to manganous salts and trivalent chromium salts, which can be reoxidized in various ways, as, for instance, by electrolysis or reaction with peroxide of lead. Persulphates, produced electrolytically, may also be used as oxidizing agents. The reoxidation of a part of the reduced compound and the reduction of another part from its oxidized condition by action on the metals of the slime in the same solution may be performed as follows: A solution may be used of an acid forming a readily soluble salt of lead, such as fluosilicic acid, in the presence of compounds of metals like iron, chromium or manganese. Ferric salt is reduced to ferrous salt, chromic acid to chromic salt, or permanganic acid to manganous salt by the action on the metals of the slime. At the same time the ferrous salt present is being oxidized to ferric salt, the chromic salt to chromic acid, or the managanous salt to permanganic acid by reaction with the acid in solution and lead peroxide which is added. Only a small amount of the oxygen carrier is stated to be present. The resulting solution of lead salts and salts of metals of the slime can be treated for precipitation of the silver by metallic copper, of the copper, arsenic, antimony, etc., by precipitation with metallic lead. The resulting solution of lead salt can then be electrolyzed with a lead cathode for the deposition of lead, and with an insoluble anode of carbon for the deposition of peroxide of lead. The latter and the acid set free during electrolysis can be used for the treatment of a fresh quantity of slime. When using electrolysis, in order to reoxidize the compound reduced in attacking the slime, the employment of a sulphate solution is not absolutely necessary, inasmuch as there may be electrolyzed a solution of iron and lead fluosilcates, or iron and lead salts of other acids forming readily soluble lead and iron salts, depositing lead on a cathode and oxidizing ferrous to ferric salt at the anode, which ferric salt is then applied to the slime for the solution of silver, copper, arsenic, bismuth, etc. From the solution silver can be precipitated by metallic copper, and the other metals dissolved as well as the copper brought into the solution by the precipitation of the silver, can be precipitated with metallic lead, after which the solution is brought back to the condition of containing iron and lead fluosilicates. It is also not absolutely necessary to use hydrofluoric acid for dissolving the antimony from the insoluble residue left after the extraction of the copper, etc. Other suitable solvents are hydrochloric acid and cream of tartar solution. Only such oxidizing reagents are used for carrying out the process, such as ferric sulphate, *f. i.*, which will not dissolve gold, even if the oxidizing agent is present in excess.

Electrolytic Cell.—J. F. Kelly, Pittsfield, Mass. Patent 792,597, June 20, 1905. Application filed Sept. 9, 1904.

The cell is particularly intended for the electrolysis of alkali sulphate, and is divided by diaphragms into three compartments. The electrolyte is contained in the middle compartment, and one of the other compartments contains the anode and the other one the cathode. There are thus no electrodes in the central compartment which contains the electrolyte. The anode is of lead, while the cathode is of iron. The liquid, which receives the products of the decomposition of the electrolyte is water, and fills the anode as well as the cathode compartments. The diaphragms are composed of one or more sulphates of the alkaline earths, calcium sulphate being chosen on account of its cheapness, though barium sulphate may be used with equal effectiveness, or a combination of the two They are thin plates, secured suitably in the cell. The electrolyte is a saturated solution of sodium sulphate, and is maintained concentrated by placing crystals of the salt at the bottom of the central compartment. When the liquid in the anode and cathode compartments has become sufficiently dense it is withdrawn at a fixed rate.

Storage Batteries.

Storage Battery.—R. Macrae, Chicago. Patent 791,350, May 30, 1905. Application filed Aug. 4, 1904.

The battery is so constructed that the container is formed with double walls, the interior of the container holding one active element, the other active element being without the container. The latter, between its walls, is filled with acid or another electrolyte. The container is made of wood, so as to get the advantage of its yielding qualities for use in automobiles.

Relief Valve for Storage Batteries.—A. P. Perry, Chicago. Patent 789,877, May 16, 1905. Application filed May 22, 1903.

The relief valve is applied to the top of the cell, and consists of a cap, provided with a central opening and an internally threaded lower portion. Within the latter is placed a rubber diaphragm, which has a microscopic central opening. Access to the interior of the cell can be readily obtained if desired.

Galvanic Elements.

Battery.—W. C. Banks, New York. Patent 790,866, May 23, 1905. Original application filed April 11, 1901. Divided and refiled Jan. 5, 1905.

The negative element of the cell is composed of a metallic basket with perforated walls, the bottom of which is formed of a cone-shaped piece of perforated tin, the smaller end of the cone being upwards. It is usually filled to within an inch of the top with black oxide of copper.

Battery Solution.—F. M. Holmes, Marionville, Mo. Patent 790,502, May 23, 1905. Application filed Jan. 25, 1904.

The solution is produced by dissolving a suitable gum in water and mixing it with the acid-exerting fluid of the battery. The gum is stated to form a coating on top of the solution, and to retard evaporation. The solution is claimed to work longer and to be more satisfactory than the old solutions.

Battery.—W. C. Banks, New York. Patent 787,716, April 18, 1905. Application filed Oct. 22, 1904.

The battery is so constructed that it may easily be taken apart and cleaned. It consists essentially of a zinc cylinder, upon which rests an insulating plate, which latter, in its turn, serves as support for a perforated cup, which contains the depolarizer.

Battery.—W. C. Banks, New York. Patent 787,715, April 18, 1905. Application filed Oct. 20, 1904.

This construction is similar to the above, of the zinc-copper oxide type; the zinc electrode, of tubular form resting upon a porcelain support, while it in turn carries the cylinder, usually of perforated tin, which contains the depolarizer. A rubber gasket, placed over the top of the zinc, insulates the latter from the tin depolarizer-basket.

Miscellaneous.

Medical Ozonizer.—M. Otto, Paris, Assignor to American Ozone Co., Niagara Falls. Patent 790,655, May 23, 1905. Application filed June 6, 1903.

The ozonizer consists of a glass tube in which the electrodes are mounted, namely, one electrode inside the tube and the other covering the outside wall of the tube. The tube is provided at its lower extremity with an air-injecting tube 3, by which the air to be ozonized is injected into the chamber, passes between the electrodes and passes out through a trumpet-chaped opening, where it is inhaled by the patient.

Electrical Ozonizer.—C. A. Sahlstroem, Ottawa, Can. Patent 788,557, May 2, 1905. Application filed June 21, 1904.

The positive electrodes in this apparatus are constructed out of a sheet of metal having triangular-shaped tongues stamped out of it. The latter are bent so that the points are in contact with an insulating sheet placed on top of the metallic sheet. The negative electrode is made of wire gauze. The ozonizer is built of successive layers of an insulating plate of glass, a wire gauze electrode, another glass plate, and a positive electrode, with the points in a direction towards the negative plate, this succession being repeated as often as desired. When a sufficiently high voltage is used between the terminals the current passes through the glass plates from the points of the positive electrode to the nearest point where the negative electrode is in contact with the plate. Pressure is exerted upon the end plates, so as to ensure good contact of the respective electrodes with the glass plates. An alternative form shown has circular electrodes, placed around glass tubes, but the principle is the same as in the first one. By means of an electrically-driven fan air is forced through the apparatus.

SYNOPSIS OF PERIODICAL LITERATURE.

A Summary of Articles Appearing in American and Foreign Periodicals.

Industrial Electrochemistry.

Electric Steel Manufacture.—We recently noticed in these columns Prof. W. Borchers' paper presented before the Association of German Iron Metallurgists on the present status of iron and steel manufacturing in the electric furnace. The discussion which followed the paper is given in full in *Stahl und Eisen*, June 15. G. Gin remarked that he has given the license for using his process in Germany to W. Brueninghaus, who has formed the company Deutsche Elektrische Stahlwerke, with the coöperation of Siemens & Halske and two other German firms. This company has installed an experimental plant at Plettenberg, where the furnace is in operation, and Gin hopes to give shortly data of the exact cost. He considers electricity as a "luxury force," that is, one which should be used only for such special purposes where it is distinctly superior to others. To explain the high heat efficiency of electric furnaces he points out that in his furnace a space of 1 cubic decimeter contains a mass of 7 kg, and stores about 2700 calories from 0 to 1800°, while the gases of the open-hearth furnace within the same volume weigh only 2 decigrams, and can store only one-quarter of the above calories. This shows that operations in an electric furnace take place in volumes of small dimensions, and that, therefore, the losses due to heat conduction and radiation are smaller than, for instance, in the open-hearth furnace. On the other hand, operations in the open-hearth furnace are difficult, on account of the reactions of the gases on the metal bath, and it is very difficult to maintain at will a neutral or reducing atmosphere. In the electric furnace the atmosphere is as desired, and its action may be considered as of no account. He claims that with the electric furnace it is possible to make steels of almost mathematically exact composition and properties. He does not think that the electric furnace will revolutionize the metallurgy of iron, but that it will mainly be used for refining purposes. He does not believe that our present steel makers have a right to say that their steel needs no refining. The electric furnace will not be the competitor but the coworker of the present metallurgical furnaces, and its introduction into practice depends on the utilization of the blast furnace gases. Prof. H. Wedding referred to the remark that the electric refining of steel is a luxury; nevertheless, it is possible to see a progress in this direction. He thinks, however, it is useless to make further experiments on the use of the electric furnace for the reduction of iron from ores. For the reduction of iron oxides which cannot be accomplished by dissociation, carbon monoxide should be used, not carbon. By means of carbon monoxide reduction is obtained at low temperatures (up to 700° C.) without considerable loss of heat, but since in the electric arc furnace the temperature is very high the reduction will be accomplished by carbon, and this should be avoided. (He did not mention resistance furnaces.) Eichoff pointed out that for the interests of the army and navy, for automobile manufacture, etc., better qualities of steel are required. He then spoke at some length on the Heroult process, by which about 4000 tons of steel have been made of very best quality. He pointed out that the quality of the product depends intimately on the slag. The more energetic the action of the slag on the bath the more quickly will the deleterious constituents of the bath be removed. In the Heroult furnace the slag has a very high temperature, and its action is very energetic. This explains why by this process steels have been made with properties different from those kinds made at present. "The commercial manufacture has shown that these new properties represent an improvement, that the properties of the steel are bettered, since with the same content of carbon the steel is tougher. For instance, Heroult steel with 0.75 per cent carbon, is as tough as ordinary steels of 0.5 per cent. Another interesting property of electric steel is that it has a high limit of elongation." * * * * "The hope of Heroult that he will be able to work out his process so that its cost will be almost the same as that of Bessemer and open-hearth steel, should be considered interesting, but it must first be proven correct in the practice." Engelhardt spoke of the Kjellin process, and thought that a disadvantage of the Heroult process is that there exists at least a possibility of particles of the carbon electrodes dropping off and spoiling the bath.

Electrochemistry and Engineering.—Prof. F. Haber, well known to American electrochemists from his visit to this country some years ago, recently delivered a suggestive address before the Munich local section of the Association of German Engineers, which is printed in *Zeit. f. Elektrochemie*, April 28. He first discussed the production of electrical energy by means of a carbon cell, explaining the action of the Jacques cell in accordance with the theory given in his St. Louis Congress paper (our Vol. II., p. 401). He thinks that the solution of the problem at a temperature below 400° C. is not impossible, especially since the reaction of carbon monoxide with atmospheric oxygen begins to get very slow only at about 300°; however, nothing of a definite nature has so far been accomplished, and the generation of electric current by means of a carbon cell seems at present little promising. With metals, as combined in primary cells, the generation of electrical energy is rapid, and this fact has often been neglected by engineers not concerned in electrochemistry. He mentions that a certain manufacturer built a hot-water boiler in which both zinc and copper were exposed to the water, which, of course, resulted in the rapid corrosion of the zinc. Similar mistakes have been made in water meters. Of greatest importance is the subject, however, for shipbuilders. The fact should be taken into account that the electrolytic action changes with the temperature and with the composition of the electrolyte. The speaker then discussed the passive state of metals. He thinks its explanation by means of very thin oxide films on the surface is correct for all practical purposes. For the electrolysis of water for the production of oxygen and hydrogen, iron electrodes may be used, but in such installations it sometimes happens that one single iron anode rapidly corrodes. This is due to a failure of circulation, whereby carbonic oxide accumulates in the positive cell compartment with formation of bicarbonate. As soon as the accumulation of carbonic oxide in the anolyte reaches a certain value the passivity of the iron disappears and the metal corrodes quickly. For this reason good circulation is essential. Heating is also effective, since it makes the carbonic oxide escape as gas before it accumulates to a dangerous degree. The worst case of iron destruction by electrolysis is found in the case of the corrosion of water and gas pipes by the stray currents from the rails of trolley systems when used for the return currents. It seems impossible to avoid this as long as the iron pipes are embedded in wet ground containing carbonic acid. The simplest cure would be to place the pipes so that they are always in dry ground, if that is possible. The author discussed the lead accumulator and the new nickel-iron alkaline accumulator, and then gave a review of electrolytic processes. He emphasized that in the case of metal deposition everything may be accomplished with pure solutions and nothing with impure electrolytes, since the difficulty is not to produce electrically the desired metallic deposit but to provide chemical means by which the electrolyte remains sufficiently pure in commercial operation. He dealt with refining of metals and with the electrolysis of sodium chloride. Asbestos paper and asbestos cloth, stiffened with a coat of cement, are a favorite construction of the diaphragm in this country, while in Germany the diaphragms are generally made of cement, which gets the necessary porosity by the addition of a salt, which is later easily dissolved out of the cement. For sodium chloride electrolysis, sodium chloride is, of course, used as this salt. He then reviewed briefly electric furnaces, electric discharges through gases, silent discharges and endosmosis, and referred to the application of endosmosis to the separation of water from peat. The commercial successes of electrochemical engineering have not been devised at the desk of the theorist, but have been developed by experimental work. Nevertheless, a full knowledge of the theory is most essential for industrial work, since it prevents spending much time and work on useless experiments.

Rapid Electrodeposition of Copper.—This is the subject of a paper of Sherard Cowper-Coles, presented on July 3, before the *Faraday Society*. The paper is here abstracted from advance sheets. The reading of the paper was illustrated by lantern slides and the author exhibited specimens of copper sheets, tubes and wire made by the centrifugal process. The various processes for increasing the current densities in copper deposition by using mechanical means for keeping the copper smooth are classified as follows:

(1) Revolving or moving the cathode, as in the processes of Wilde, Cotsworth, Wylie and Grant. The current density employed is comparatively low.

(2) Burnishing the copper during electrodeposition—the Elmore process. The usual current density is under 20 amps. per square foot.

(3) Insulating the growths on the copper so as to prevent further increase. For example, the Dumoulin process, in which sheepskin and other impregnators are employed. The current density is 35 to 40 amps. per square foot.

(4) Rapid circulation of the electrolyte. Examples: Thofehrn's process, in which impinging jets are caused to play on the surface of a rotating cylinder (C.D. 50 to 100); Graham's process, in which the electrolyte is discharged on to a flat cathode surface (C.D. 300, just under the influence of the jets); Poore's process, in which the solution is sprayed on to the cathode, the stream forming the only electrolytic connection; Dessolle's process, used in Paris for coppering ornamental iron work, and finally, Harrison's, likewise an impingement process. The author considers that impingement processes are not likely to be applied commercially until the amount of solution circulated can be greatly reduced.

(5) Revolving mandrel at a critical speed (centrifugal process). This process has been developed by the author and the latest methods of working are here described. The mandrels are suspended vertically, and are provided with Pelton wheels, which are driven by the electrolyte impinging against them. In the most recent form, however, a tubular vat is used, and hollow mandrels suspended on ball bearings, through the middle of which runs the spindles. These are driven by a worm gearing from below. An 8-foot mandrel has only to be driven at about 50 revolutions per minute.

The author claims the following advantages: The copper is refined and manufactured into sheets or tubes in one operation, and is of a hard nature, similar to that which is cold-rolled; the process is at least ten times faster than any existing electrolytic process; a high current can be employed without deteriorating the quality of the copper; there is no risk of lamination; the plant is simple and free from mechanical complications; the amount of copper locked up for a given output is small compared to other processes; finally, anodes of very impure copper can be used as compared to the anode copper used in other systems. By using a mandrel in which a V-groove has been indented, the spiral deposit that results can easily be pulled away, and then drawn down at once into wire, which can thus be made from crude copper in what is practically one operation. The estimated capital expenditure and cost of working the "centrifugal" process are given in the paper.

A. Stanley Elmore said in the discussion the author had not mentioned heating the electrolyte as a method of accelerating the rate of deposition. He knew of one plant working the Wilde process in which the cathode was rotated rapidly, and the current density was by no means "comparatively low." He gave some further details regarding the Elmore process. The C.D. was occasionally pushed up to 30 amps. per square foot, and has been carried up to over 200 amps. A tensile strength as high as 42 tons per square inch had been attained. By the use of a burnisher the same high C.D. could be used with a less expenditure of power than in other processes, and he claimed that by a combination of burnishing and moderate circulation of electrolyte better and more certain results could be obtained than by the author's "centrifugal" method. He asked how the author proposed to make tubes of less than 2½ inches diameter. Milbourne said that all that was claimed for the "centrifugal" process had been borne out by the results of the trials he had made with a practical plant on the Continent. W. C. Prebble pointed out that in the author's process the circulation of electrolyte took place just at the place where it was most needed, namely, the surface of the cathode. J. V. Mackenzie described an arrangement by which he had used current densities up to 45 on electrotyping work. C. J. Steinhart said that capital expenditure was an all-important item; whether that required in the "centrifugal" process remained within the permissible limits remained to be seen. The author had successfully surmounted the "early stage" difficulties of his process. T. C. Cloud criticised the author's theory regarding the growth of nodules. He thought that the toughness of the metal was the result of the particles being drawn out into long fibers as they were being deposited. Cowper-Coles replied to the various points raised. Tubes as small as 2½ inches could not be deposited, but could easily be drawn down.

Nitric Acid from Air.—A very useful review of the various processes which have been devised for the production of nitric acid by electric discharges through air is given by H. R. Carveth and C. L. Rand, in the May issue of the *Sibley Journal of Engineering.* Since the paper is itself a review it cannot be abstracted again. The following table is given of the yields obtained by various experimenters, the figure given being the grams obtained per kw-hour:

1897—Rayleigh	49.1	grams HNO_3
1900—McDougal and Howles (best yield)	33.8	" "
1902—De Kowalski	55.0	" "
1903—Muthmann and Hofer	70.0	" "
1897—Crookes	74.0	" "
1902—Bradley and Lovejoy	83.0	" "
1904—Birkeland and Eyde	110.0	" "

The calculations of Muthmann and Hofer indicate 157.5 grams of nitric acid per kw-hour as the theoretical maximum efficiency. The above results, which in reality are mere approximations, show that the progress has been satisfactory, and indicates that commercial success may yet be attained. The main improvements which may possibly have an advantageous influence on the yield of nitric acid by the electric flame are: (*a*) A vessel so designed as to produce the best hot-cold effect. (*b*) Spark long drawn out. (*c*) Low current and rapid alternations. (*d*) Increased pressure. (*e*) Rapid passage of gases and the introduction of the gases in the correct proportions. (*f*) Temperature below 1200° C. The authors think it is very probable that indirect methods for the fixation of the nitrogen by means of some intermediate compound such as a nitride or cyanide may be found to be more economical.

EXPERIMENTAL AND THEORETICAL.

Electrode Temperature in Electrolysis.—How important the influence of temperature is in many electrolytic processes is well known, but so far one has been satisfied to regulate the temperature of the electrolyte. W. Moldenhauer, in *Zeit. f. Elektrochemie,* May 26, gives the results of an extensive experimental investigation in which he first investigated the heat set free or consumed at the electrodes during electrolysis. Second, the temperature differences between the electrode and the electrolyte under varying conditions. Finally, he investigated what effect in some elec-

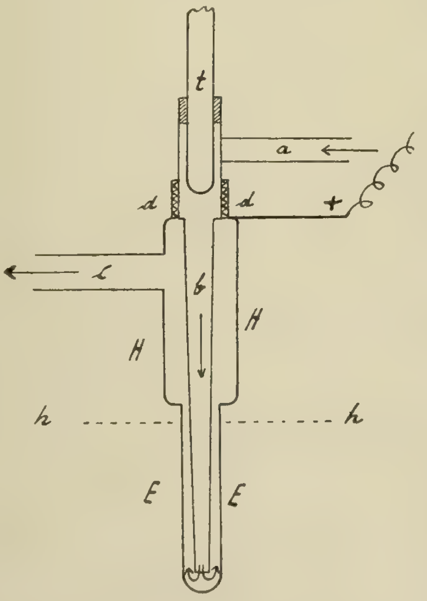

FIG. 1.—COOLED ELECTRODES.

trolytic process is produced by an artificial cooling of the electrode. The arrangement is shown in Fig. 1. E is a platinum tube closed at the bottom, which serves as electrode and is suspended into the electrolyte up to the level *hh*. It is soldered to the brass tube H. The cooling water enters through *a*, passes downwards through *b* and then back and leaves through *l*. The thermometer *t* gives the temperature of the cooling water. In the following we give the results obtained with the production of persulphuric acid from sulphuric acid. Berthelot and Elbs and Schoenfeld found that the best conditions are acid of 1.34 to 1.5 specific gravity, high current density and low temperature. The present author first used a concentration of H_2SO_4 of 1.345 specific gravity, with an anodic current density of 100 amp. per square dc. The results are shown in Table 1. The efficiency represents the amphere-hour efficiency of the production of $H_2S_2O_8$ in per cent:

Table I.

Temperature of electrolyte....	12° C.	12° C.	12° C.
Temperature of anode........	not cooled	11° C.	0 to 2° C.
Efficiency in per cent..........	54.6	66.4	68.2

He then used more concentrated solutions. His results for an acid of 1.6 specific gravity are given in Table 2:

Table II

Temperature of Anode	Efficiency in Per Cent.
Not Cooled.	55.3
10° C.	36.4
2° C.	9.2

Table III.

Current density in amperes per square decimeter..	100.	50.
Efficiency in per cent..........................	26.6	63.9

The current density in Table 2 was 100, while the temperature of the electrolyte was 12° C. Table 3 shows how the efficiency depends on the anodic current density.

These results are easily explained. The formation of $H_2S_2O_8$ is due to the combination of two discharged HSO_4 ions. Since sulphuric acid of medium concentration contains at lower temperature more HSO_4 ions than at higher temperature, the rule is that a lower temperature of the electrolyte aids the formation of persulphuric acid. Table 1 shows how this may be easily accomplished by cooling the anode. Further, a more concentrated acid contains many HSO_4 ions already at ordinary temperature. If in this case a strong cooling of the electrode is employed many non-dissociated H_2SO_4 molecules are formed, which must produce a high resistance at the anode on account of the high current density. This causes a high local heat effect which counteracts the formation of persulphuric acid, as shown in Table 2. If, however, the current density—and, therefore, the local heating effect—is reduced, considerable quantities of persulphuric acid may be obtained even at higher concentrations.

Metallurgy of Copper.

Electric Conductivity of Copper.—A very full and important investigation of the effect of impurities on the electrical conductivity of copper has been made by L. Addicks, the results being given in a paper before the American Institute of Mining Engineers, published in the May issue of the *Bi-monthly Bulletin*. Electrolytic refining has made it possible to produce copper of a very high degree of purity, the metallic impurities averaging but a few thousandths of 1 per cent; oxygen, usually present in the form of compound oxide, bringing the total up to about a tenth of 1 per cent. The author gives in Table I. the amount of various elements which lowers the conductivity several per cent, his researches referring, therefore, to a series of impure coppers rather than alloys.

These numerical results are also given in forms of diagrams by the author, and the author utilizes these curves for plotting a tangent at the zero point of impurity. This gives him the ratio of the lowering of the conductivity to the amount of impurity present, for the case of an infinitely small addition of impurity. This ratio of the percentage lowering in conductivity to the percentage impurity added is given in the column headed "factor" in Table II., where the elements are grouped in their order in the periodic system. In each group the factor decreases with increasing atomic weight.

"The results given in Table I. show what a severe require-

TABLE I

ELEMENT ADDED.	First Sample.			Second Sample.			Third Sample.			Fourth Sample.		
	Added.	Cu	Cond.	Added.	Cu.	Cond.	Added.	Cu	Cond.	Added.	Cu.	Cond.
	%	%	%	%	%	%	%	%	%	%	%	%
Aluminium	.0	99.96	99.6	0.006	99.90	98.6	0.109	99.65	66.8	0.739	99.02	43.5
Antimony..	0.007	99.96	99.6	0.022	99.94	97.2	0.047	99.86	95.4	....	.	....
Arsenic....	0.004	99.96	99.6	0.007	99.95	96.8	0.013	99.94	93.2	0.140	99.82	62.3
Bismuth ..	.0	99.96	99.6	0.028	99.93	99.6	0.045	99.91	99.3	.	....	....
Cadmium..	0	99.96	99.6	0.062	99.90	99.5	0.113	99.87	99.1	0.427	99.55	96.1
Gold	0	99.96	99.6	0.089	99.86	98.9	0.149	99.84	98.4	0.317	99.64	96.4
Iron......	0	99.89	100.5	0.042	99.93	96.8	0.046	99.90	92.9	0.068	99.89	89.6
Lead	0	99.96	99.6	0.083	99.82	99.1	0.052	99.86	98.7	0.347	99.56	98.3
Oxygen ...	0.020	99.98	100.7	0.050	99.95	101.4	0.100	99.90	100.5	0.200	99.80	98.0
Phosphorus	0	99.96	99.6	0.080		52.3			.	.	...	....
Silicon....	0	99.96	99.6	0.007	99.89	99.4	0.007	99.89	99.1	0.042	99.89	99.0
Silver......	0.003	99.89	100.5	0.137	99.81	100.0	0.340	99.60	98.3	0.503	99.49	97.9
Sulphur ...	0	99.89	100.5	0.053	99.93	100.0	0.135	99.83	99.0	0.236	99.75	98.9
Tellurium .	0	99.89	100.5	0.065	99.82	100.4	0.181	99.74	100.2	0.405	99.65	68.7
Tin	0	99.96	99.6	0.052	99.85	97.6	0.097	99.85	92.7	0.295	99.61	79.8
Zinc......	0	99.96	99.6	0.048	99.91	98.3	0.095	99.79	96.3	.	.	...

TABLE II

$$\text{Factor} = \frac{\text{Percentage Lowering in Conductivity}}{\text{Percentage Impurity Added}}$$

Periodic Group	Element Added.	Factor	Atomic Weight.	Remarks
I	Silver....	6	108	
I	Gold	[illegible]	197	
II ...	Zinc	3	65	
II ...	Cadmium	9	112	
III...	Aluminium	[illegible]	27	
IV ..	Silicon....	7	28	
IV ..	Tin	67	119	
IV ..	Lead	3	207	Made metal red short
V	Phosph'r's	[illegible]	31	
V....	Arsenic...	720	75	
V....	Antimony	[illegible]	120	
V....	Bismuth. .	4	208	Made metal red and cold short.
VI .	Oxygen . .	25	16	Large amounts made metal brittle
VI	Sulphur. .	8	32	
VI ..	Tellurium	4	128	Made metal red and cold short.
VIII	Iron	[illegible]	56	

ment the customary 97 or 98 per cent conductivity specification is, especially as copper is usually associated with arsenic; and when it is considered that the average electrolytic refinery daily passes this requirement by a margin of 2 or 3 per cent, frequently using anodes containing 1 per cent or more of arsenic, it will be appreciated that electrolysis would have come as a refining operation even did copper never carry gold or silver."

RECENT METALLURGICAL PATENTS.

Copper.

Pyritic Smelting.—With respect to pyritic smelting, two patents of R. Baggaley (789,648, May 9, and 784,651, March 14) require careful interest. His process produces copper mattes without the need of water concentration and calcining and without the use of carbonaceous fuel, or with only a small percentage of such fuel. The inventor claims that in his process "the raw-ore tonnage, without previous concentration, can be quickly smelted, nine-tenths of it discharged as worthless slag, and the remainder converted into matte." Talcky ores and low-grade ores and ores high in iron can be used. The furnace, as shown in Fig. 1, has a lower dissolving and converting portion 2 and twyers 3. The throat 4 is made variable in width by the sliding sections 6, so as to maintain the heat in the proper condition; 7 are the charging doors, 9 the slag tap, or overflow, and 10 the matte tap with its safety-bot; 12 is the safety-botting device for the matte tap-hole; 17 are water-cooled passages.

For the operation of the process the sulphide ores (relatively low in silica and high in matte-making sulphides, such as pyrite, pyrrhotite, and the various sulphides and arsenides of copper) are separated from the more highly silicious ores (whose gangue is granite, quartz). The low-silica sulphide ores are melted in a suitable furnace to produce a low-grade matte that is high in fuel values, *i. e.*, high in sulphur, iron, etc. The matte and slag are discharged into the usual forehearth, the slag is separated from the matte, and the clean matte is then drawn off when required for use in the furnace, Fig. 1. This furnace having been heated by a wood fire, or by burning gas or oil, is then charged with a sufficiently large bath of such molten matte to submerge the twyers 3 and to extend above the same to a considerable distance, say up to the level of the slag-spout.

Relatively small charges of the above-described silicious, or sulphide ores, are added to the bath from time to time, through 5 and carbonates, oxides, concentrates, mattes or metal may also be added if available. The blast of air, introduced in great abundance through the twyers, oxidizes the combustible ingredients of the bath (sulphur, iron, etc.) and produces an intense heat. The oxidized iron is fluxed by the silica of the added ore, and forms a floating slag, while the metallic compounds of the ores are dissolved in and become part of the molten matte. As the body of matte is enriched by oxidation its exhausted combustible elements and compounds are replaced by addition of the sulphide ores in solid condition. The slag is drawn off from time to time through the tap-hole, or overflow 9, so that the surface of the bath is kept clean for the ready dissolving of ore. During the withdrawal of the slag the blast is shut off temporarily by plugging the twyers. For the regulation and control of the volume of blast, delivered into the bath, iron bars are inserted into the twyers.

When a sufficient body of matte of the desired content of values has accumulated in the furnace, a portion of it is tapped off for further treatment through the matte tap-hole 10, and the bath and its heat-producing power are then restored by additions of sulphide ores. It is important that during the progress of the operation the enrichment of the matte should not proceed too rapidly or beyond certain limits. It should never be allowed to exceed 60 per cent in values, and it should preferably not exceed 40 per cent.

R. Baggaley and C. M. Allen (789,133, May 9) refer to the difficulty experienced in pyritic smelting, that when the metallic sulphides melt by a process of liquation out of the silicious shell, the furnace charge is apt to form scaffolds on the walls of the furnace. The slags produced in the converting of copper mattes are necessarily basic, and are specially adapted to dissolve those accretions. The inventors pour the corrosive converter slags, when molten, into the blast furnace, and distribute the same around the walls, thus dissolving all troublesome accretions and at the same time furnishing additional heat. E. W. Lindguist (789,160, May 9) describes details of apparatus for feeding and distributing the molten converter slag into the furnace.

Wet Process.—G. Gin (793,186, June 27), best known by his work in electrometallurgy, has devised a wet process for the extraction of copper from its sulphuretted ores. His scheme is to utilize the sulphurous acid resulting from the roasting of copper ores afterwards for extraction of the copper. The process is based on these three reactions:

(1) $3CuO + 3SO_2 = 3CuSO_3$
(2) $3CuSO_3 + CuO = Cu_2SO_3 . CuSO_3 + CuSO_4$

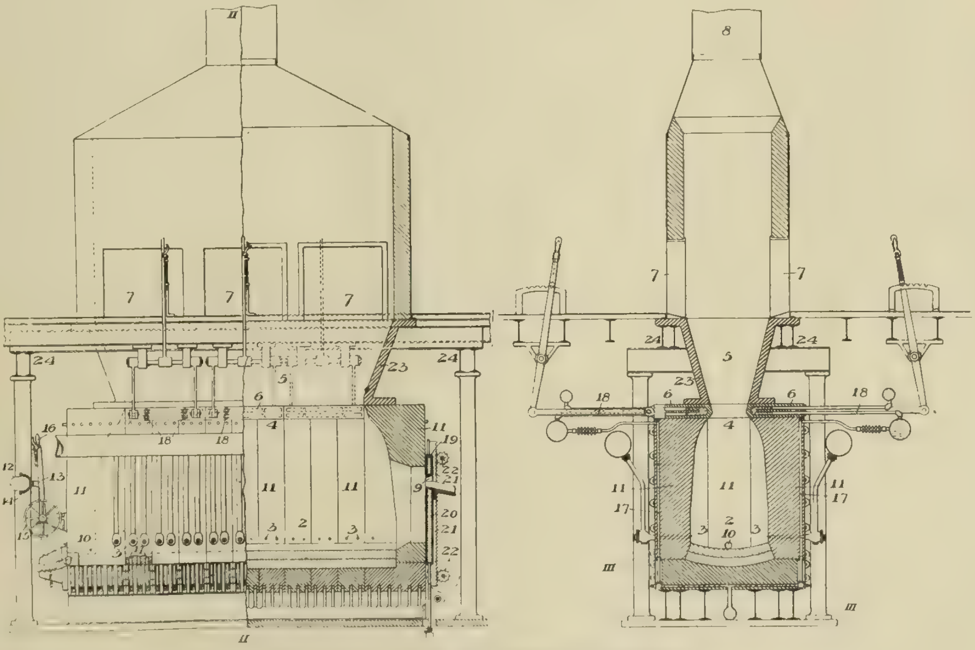

Fig. 1.—Furnace for Pyritic Smelting.

(3) $Cu_2SO_3 . CuSO_3 = 2Cu + CuSO_4 + SO_2$

By adding these three equations, one gets:

$$4CuO + 2SO_2 = 2CU + 2CuSO_4$$

which means that one-half of the copper is obtained in the metallic state and the other half in the form of sulphate. The above reaction (3) requires heating the solution to 180° C. under pressure. In practice the ore is roasted in such a way that all the sulphide of copper is changed into bioxide or sulphate, and all the iron into peroxide. It is then submitted to methodical lixiviation by means of a solution of sulphurous acid. Reactions (1) and (2) take place, while at the same time iron sulphite and sulphate are formed. The saturated solution of the copper and iron salts is collected by means of a pump in a copper boiler, where it is delivered at a temperature of 180°. At this temperature the iron sulphite and sulphate are quite insoluble and precipitate. Simultaneously reaction (3) takes place. The turbid liquid is then injected under its own pressure into a filter press heated by the vapor circulating in the plates

ZINC.

Retorts.—In zinc metallurgy, as now carried out, it is necessary to limit the amount of iron, manganese, garnet, fluorspar, fusible silicates, or the like that may be present in the charge, so that the resultant slag shall contain but a relatively small quantity of these injurious elements. If present in large quantities it is found that the acid walls and bottoms of the retorts are soon destroyed. For this reason the treatment of many kinds of zinciferous ores containing notable percentages of bases, is impracticable. A. L. J. Queneau (789,543, May 9, assigned to New Jersey Zinc Co.) constructs a composite retort. The main body portion is of the usual fire-clay and sand mixture, while the "outer surface" (which may be either the interior or the exterior of the retort) is made up in proper thickness of a mixture of fire-clay and a basic material; here the basic material takes the place of the customary sand, either wholly or in part.

This retort is made in a single operation as follows: The batch of material for the main body portion of the retort is made from a pugged mixture of fire-clay and sand in the usual manner. In preparing the batch for the surface portion of the retort the same fire-clay is used and mixed with an inert or a basic material in granular form (such as chromite, carborundum or the like). This basic material takes the place of all or part of the sand. This mixture of refractory fire-clay and inert or basic material is then pugged in a pug mill to the desired plasticity. It is then stamped in the hammering machine. After the hammering operation it is superimposed upon the preliminarily-stamped clay and sand mixture intended for the main body portion of the retort, and the whole is then thoroughly hammered to a solid wad. This wad is then introduced in the proper position in the chamber of the retort press. The remaining manipulations do not differ from ordinary practice.

IRON AND STEEL.

Liquid Fuel.—A process of smelting iron ore by liquid fuel is described by E. Riveroll (791,577, June 6). The main feature is the deposition of a coating of carbon upon the charge of iron ore and limestone by exposing it to the action of the fuel burning with insufficient oxygen to produce perfect combustion. Fuel thus burned produces a smoky flame, composed of carbon dioxide, plus free carbon, and the latter is deposited in the form of soot. The charge is then subjected to perfect combustion, and the resulting carbon dioxide comes in contact with the carbon deposited on the charge of ore. By this union, at a considerable degree of heat, carbon monoxide gas is formed at the smelting zone, and this gas passes through the charge and robs the ore of its oxygen and leaves the furnace as carbon dioxide. "The ore thus deprived of its oxygen is at the smelting zone reduced, and is from there passed down into the hearth or crucible, where the earthy impurities of the ore unite with the flux, forming a slag, and the iron and slag are separated by gravity, the iron sinking to the bottom and the slag floating on the top of the iron." In a test run of this process, using crude oil only as fuel with a specific gravity of 20.5, the iron ore showed by analysis: Metallic iron, 56.7; silica and alumina, 13.3; sulphur, 1.4. The analysis of the limestone flux was lime, 52.5; silica, 6.3. After smelting the resultant slag consisted of 45 per cent silica and alumina, 50 per cent lime, and 5 per cent iron and other elements.

Open Hearth Operation. H. Knoth has formerly (712,380, Oct. 28, 1902) patented a process of manufacturing steel, wherein the furnace contents are tapped into a ladle in which a portion of the steel is reserved and mixed with the molten pig metal with which the ladle is filled, after its slag and the greater part of its steel contents have been poured off. The ladle, with its mixed charge, is then run back and charged into the same furnace or any other that may be ready to receive it. The object was twofold: first, to reduce the time required to prepare the heats in the furnace for lapping and to avoid the cost of scrap charged with the pig iron. This process still involved waste and delay in the production and maintenance of a highly basic slag capable of taking up the metalloids in the bath. The same inventor now proposes the following process (788,650, May 2):

The unpurified pig metal, constituting the initial charge, when high in silicon, is subjected in a liquid state to an oxygen-yielding blast in an acid-lined vessel and the silicon and part of the carbon content of the metal reduced. (If the pig is sufficiently low in silicon this converter step is omitted.) The desiliconized metal is then introduced into an open-hearth furnace and there treated according to the usual methods, and in the presence of a highly-basic slag to remove the phosphorus and reduce the carbon to the desired point. The slag which results from this treatment is drawn off with the molten steel into the same or a separate ladle, from which it is reintroduced back into the same or any other furnace, as long as it retains its basic qualities. The lower the metals charged are in silicon and phosphorus, the greater is the saving.

Both processes combined are worked as follows: An initial heat of metal and slag is prepared and treated in an open-hearth furnace. The furnace contents are tapped into a ladle. Two-thirds of the purified metal are poured into molds; one-third is recharged into the furnace with new unpurified material (which may have been previously treated in a Bessemer converter). The reserve of slag is then fed back into the furnace.

Purifying Pig.—J. B. Nan (786,048, March 28) pours liquid pig metal upon a mass of broken pieces of solid oxidizing material (iron ore) in such manner that the liquid metal is broken up and descends in small streams through the spaces between and in contact with the pieces of ore. A bath of liquid metal thus accumulates in which the pieces of oxidizing material are maintained immersed until the reaction has proceeded to the desired point. Several arrangements of apparatus are described for this purpose.

Charcoal Iron.—J. J. Hudson (785,002, March 14) makes charcoal iron in a furnace heated by means of oil, gas or hydrocarbon vapor. In this furnace the metal is melted in the presence of charcoal, while at the same time blasts of air are introduced into the metal for the purpose of boiling it. The metal is thereby refined by means of the charcoal, air blast, and other source of heat, eliminating sulphur, phosphorus, silicon, etc. The metal filters through the charcoal, and the latter consumes the impurities as they are eliminated from the molten mass. The mass is decarburized until the analysis of the molten metal shows the quality of iron required. The metal is then drawn directly into a suitable mold, from which it may be taken as soon as it has received a surface chill, given a wash heat in a suitable furnace or oven, and then passed directly to the mills to be rolled into skelp, plates, or other shapes. The inventor proposes to use his process for the refining of mild steel by charcoal, so as to render it available for all purposes for which charcoal iron of the best grade is available.

Iron Sand, Fire Ore, Flue Dust, etc.—M. Moore and T. J. Haskett (791,928, June 6, and 792,440, June 13) treat "magnetic iron sands," or finely crushed ferruginous ores, as follows: They first concentrate the ore and separate from it any silica or earthy materials. The ore is then treated in a tower containing various chambers. While passing through the first chamber the ore is subjected to the action of heat, produced by the admittance of air with waste carbonic, oxide or hydrocarbon gas issuing from a second chamber. The ore is then passed through this second chamber, and is here subjected to the progressive reducing action of those gases alone. The result is that all the oxygen is removed from the ore, which is thereby reduced to the metallic state, but is still in a finely-divided condition. Without coming into contact with an oxidizing atmosphere, the metal is then passed into a Siemens, or other gas furnace, and immersed in a bath of molten metal or slag, which is always maintained therein, and in this furnace it is fused and "balled up" as wrought iron or converted into molten steel. During the time the charge is being delivered to the gas furnace the supply of atmospheric air to the gas furnace is cut off.

D. Baker and W. W. Hearne (788,813, May 21) prepare magnetic concentrates of fine iron ores, such as pyrites residuum, pyrites cinders, etc., as follows for the treatment in the blast furnace: If the material has a large sulphur content it is first desulphurized. For this purpose it is fed into a long rotary kiln, where it is agitated at a temperature low enough to prevent the sulphide of iron from smelting and at the same time maintaining strongly oxidizing conditions. When the desulphurization is complete the material reaches the zone of highest temperature in the kiln where it is agglomerated. No fluxing material is used, but those foreign particles of alumina, silica, lime, magnesia, etc., which are intimately combined with the iron oxide in the ore are utilized. These are fused together with the iron oxide, resulting in an agglomeration of the ore.

C. S. Price (794,152 and 794,153, July 4) prepares flue-dust for the treatment in the blast furnace by mixing 4 to 8 parts of flue dust with 1 part of ground clay in a pug mill. This mixture is moistened with sufficient water to agglomerate it into lumps, which are charged into the blast furnace. The clay is said to be useful in serving to increase the quantity or regulate the quality of cinder, as required in many cases.

Cleaning Furnace Gases.—For the utilization of blast furnace gases it is of utmost importance to thoroughly clean them. Three recent patents refer to this subject. J. S. Oursler (791,160, May 30) employs a gas-washer adapted to cause the gas to pass in a sheet against and into a stream of water, sprays of water being employed to aid in the washing and cooling of the gas. The foreign matter is collected in dust pockets.

W. Schwarz (793,544, June 27) employs an apparatus belonging to that class of purifiers in which water and centrifugal force are used. The special feature is that the purifying effect is confined to the periphery, *i. e.*, to annular spaces contiguous to the internal surface of a stationary casing of cylindro-conical form.

J. Shields (793,745, July 4) conducts the dusty gases beneath or into the midst of a constantly-replenished heat of porous or granular material. The gases pass upward through the heap, and the filtering material collects the dust, and is gradually drawn away from the bottom of the heap, while the top is constantly replenished with fresh material. The gases thus encounter cleaner and cleaner material as they rise. Such a filter is stated to be specially useful in the treatment of the gases issuing from pyrites-burners. Suitable filtering materials are coke, sand, crushed quartz, granulated slag, pumice, etc. The arrangement is shown in Fig. 2. The gases are conducted into the heap by pipe *c* through downwardly-projecting orifices *c'*, situated beneath a horizontal disc *d*, which creates a cavity in the middle of the heap. The gases pass off through *l*.

Metallurgical Furnaces.

Roasting Furnace.—For the treatment of ores which require a comparatively long exposure to heat, it is economical to materially increase the diameter of the ordinary McDougall furnace to secure an increased area for the hearth. With the usual construction this involves a difficulty in so far as the ore tends to accumulate toward the center of the hearth and to choke the furnace. In order to dispose of it as fast as it accumulates, F. Klepetko (793,939, July 4) provides each rabble-arm with a series of rakes whose depths increase as they approach the center of the furnace, so that the radically-effective sweeping surface increases toward the center. The same inventor (792,053, June 13) modifies a detail of construction of the rabble arms described in one of his former patents. For cooling purposes these arms are made hollow, and the inventor now provides them with ribs on the inner surface, following the general longitudinal direction of the arm. C. S. Repath and F. E. Marcy (794,118, July 4) use a central hollow shaft from which radiate the hollow rabble-arms; air for cooling the shaft and arms is taken in at the top of the central shaft, either by natural or forced draft, from where it passes into the arms, and from there it may be discharged into the charge, so that the air answers the twofold purpose of cooling and furnishing oxygen to the charge. The connection between central shaft and rabble arms is as follows: Passed transversely through the walls of the shaft in earth-hearth is a T-bar, eye-beam, or the like, procured to the shaft walls by means of angle-plates. The projecting portions of the T-bars serve as support for the interchangeable hollow rabble arms, which are simply slipped over the support and their ends keyed by a wedge.

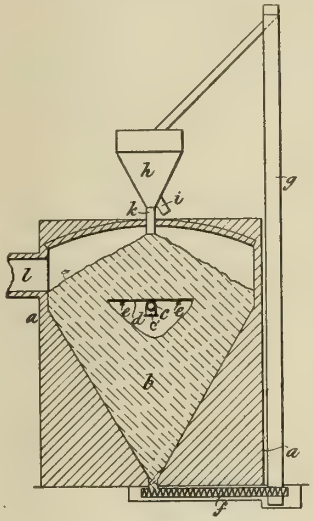

FIG. 2.—APPARATUS FOR CLEANING GASES.

A. M. Beam (793,816, July 4) patents mechanical details of construction of a roasting furnace in which the ore travels through a rotating cylinder, which is subjected to the products of combustion from two fire-boxes. There are three concentric cylinders; the outer and middle one are detachably secured together, so as to rotate in unison, while the central cylinder is connected to the middle one by a series of integral partitions. The space between the middle and the central cylinders forms the ore chamber, while the annular space on the outside and the cylindrical space inside are flues, through which the products of combustion pass from the combustion chambers to the stacks.

Gas Furnace.—J. C. Swindell (787,131, April 11) patents details of a compact arrangement of the combustion chamber of a gas furnace, surrounded on its top and sides by channels in such a way that the incoming air is thoroughly heated by the heat imparted to the walls of the combustion chamber and by the heat of the waste gases. G. L. Fogler (786,746, April 4) heats the gas supplied to the furnace before it is fed into it; for this purpose he uses the waste gases of his furnace, which are, of course, first collected and purified. A. Kurzwernhart (786,770, April 4) utilizes the waste gases in a Siemens regenerative furnace to force the combustible gas into the furnace.

Hydrocarbon Furnace.—C. E. Glafke (790,825, May 23) uses a revolving, inclined, longitudinal smelting chamber into which the ore is fed at the higher end; at the same end the burner is provided, fed with hydrocarbon fuel. The other (lower) end of the chamber is partially closed by a baffle and projects into the stack, in the bottom of which is the molten ore receptacle.

G. L. Bourne (793,174, June 27) endeavors to increase the efficiency of furnaces using crude petroleum or fuel oil as a fuel. For this purpose he introduces oil and air into a primary combustion chamber, and there volatilizes and converts the oil into a complete gaseous state, and then introduces a blast of fresh air to supply the necessary additional oxygen to provide a perfect combustion in a secondary combustion chamber. The heating chamber is located adjacent to the secondary combustion chamber.

Various Furnace Constructions.—J. L. Lansing (792,223, June 13) patents a method of roasting telluride ores. In order to prevent any considerable portion of the values from being lost in the form of tellurium fumes, he roasts the ore in closed furnaces which are connected with a condensing chamber. The latter contains cold water through which the telluric fumes and gases must pass before their discharge into the open air.

W. S. Rockwell (792,169, June 13) patents a combination of two rotary converting chambers, independently rotatable and located end to end in close contact with each other. While the metal in one chamber is being molten, ingots are placed in the other chamber and are subjected there to a preliminary heating, the products of combustion being passed over from the first chamber to the second. After the metal is tapped from the first chamber the metal in the second chamber is melted and new ingots are heated in the first chamber.

In order to avoid tilting, J. Morat (784,956, March 14) mounts the furnace centrally so as to be revoluble on a support, the axial line of the vessel being inclined to the vertical. The vessel is revolved on this angular axis so that certain charging or discharging openings in the side of the vessel are brought either below the level of the molten metal or above it.

W. E. Williams (792,642, June 20) uses an almost spherical melting furnace with a single opening, mounted so as to be revolved in two planes. There is provided a set of fixed concentric flues for supplying air and fuel and discharging the furnace gases, and means for rotating the furnace while so connected with the flues, although otherwise closed. The intention is to expose continuously the contents of the furnace to the freshly-heated portion of the lining that comes underneath it as the furnace revolves, while the melted metal keeps flowing down to the bottom and the unmelted parts are carried up to the sides to be exposed above the melted portion.

C. C. Medbery (788,546, May 2) patents a furnace with a cylindrical revoluble heating chamber supported and controlled by anti-friction rollers. Means are provided for tilting to facilitate charging and pouring, and for adapting the heating chamber to be used as a converter and for reducing and refining operations, also air blast and fuel-supplying pipes and devices, etc.

Miscellaneous Details of Furnace Construction.—P. Healey (786,121, March 21) endeavors to improve the construction of smelting furnaces in which the lower portion of the stack is provided with double walls, forming an annular jacket for water or air by which that portion of the stack around the zone of fusion is kept cool. The feature of the present invention is that neither water nor air alone is used, but a mixture of the two in the form of an atomized spray.

J. W. Seaver (792,735, June 20, assigned to Wellman-Seaver-Morgan Company) patents a furnace-filling apparatus for feeding rectangular elliptical or other elongated furnaces and smelters. He provides two traveling covers connected together and spaced apart and an endless conveyor mounted upon one cover to feed the charge between the two covers, while a traveling lorry runs on the cover carrying the conveyor and discharges on to the latter.

J. B. Ladd and D. Baker (785,311, March 21) patent a bell to secure an even distribution of stock in a blast furnace. The bell comprises essentially a number of concentric sections movable relatively to each other.

W. Kennedy (792,047, June 13) patents apparatus for changing the bell and bell-ring of blast furnaces. The usual bell-operating mechanism is employed for raising the bell and the hopper, which has its lower portion formed by a bell ring, to such a height that the removal of the bell-ring and bell from position over the furnace is possible and new parts may be substituted.

BOOK REVIEWS.

Uber die ferromagnetischen Eigenschaften von Legierungen unmagnetischer Metalle. By Fr Heusler; 64 pages, 13 figures, 8 tables and 3 curve sheets. Marburg; N. G. Elwert.

This little volume on the magnetic properties of alloys of non-magnetic metals gives a brief outline of the principles of magnetism with special reference to the recent important researches of the author upon magnetic alloys.

The principal experimental results with alloys of manganese, aluminium and copper are summed up in two short chapters. It appears that the best magnetic effects are obtainable with about equal molecular weights of manganese and aluminium. This may be called the Mn — Al alloy. The greater the percentage of this alloy in copper, the higher the final resulting magnetic susceptibility. The highest flux density recorded is 6480 gausses.

The pamphlet will have special interest for students of physics and magnetics. It forms a separable part, or section, of a larger treatise, entitled "Schriften der Gesellschaft zur Beförderung der gesammten Naturwissenschaften zu Marburg." "Publications of the Marburg Society for the Advancement of Natural Philosophy."

Diagonal Concentrating Table.

In concentrating mineral particles on a table three forces are employed, viz.: gravity, reciprocating motion and flow of wash water.

Gravity acts in a vertical direction on the mineral particles, forcing them downward, the heavier particles occupying the lower strata.

The reciprocating motion of the table is a horizontal force, and tends to bring the particles of ore forward in the direction of the line of reciprocation. The flow of wash water is also a horizontal force, but transverse to the reciprocating motion. Thus, the reciprocating motion and the flow of wash water act in nearly transverse horizontal lines, so that the resultant of these two forces is a line diagonal to both. Consequently, the material on the table tends to travel in a diagonal direction. In consequence of this general tendency the Overstrom concentrating table, built by the Allis-Chalmers Co., Milwaukee, Wis., is built diagonally to the reciprocating force and to the flow of wash water as indicated in Fig. 1.

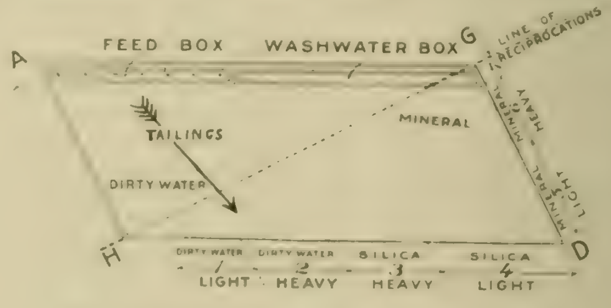

FIG. 1.—DIAGRAM OF ACTION OF DIAGONAL TABLE.

As a matter of fact the whole material would travel in a diagonal direction only if it was uniform in composition, since, in this case, each particle would be acted upon with equal force by the two horizontally acting forces. In reality, however, we have a mixture of mineral particles of higher specific gravity with gangue or silica of lower specific gravity.

The reciprocating motion of the table has a greater action on the particles of mineral, which are closer in proximity to the table surface than on the lighter gangue or silica which remains on top of the mineral. On the other hand, the flow of wash water produces a greater movement of the gangue or silica than of the mineral particles, and washes the particles of the lighter gangue a greater distance in a given time. As the gangue is on the top of the mineral it is acted upon more freely by the wash water until a final separation takes place. Thus, it is that while uniform material of medium specific gravity would tend to travel in a straight line towards D in Fig. 1, the non-uniform material is separated, the heavier mineral being moved by the reciprocating action away from the straight line towards GD, while the lighter gangue is deviated by the wash water to the other side towards HD.

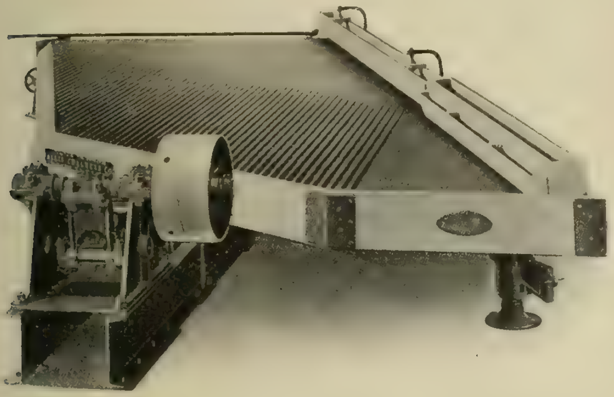

FIG. 2.—OVERSTROM CONCENTRATING TABLE.

The diagonal construction of the table has some very important consequences. First, nearly the whole available surface of the diagonal table is effective surface. The large effective surface diagonal to the reciprocating motion and to the flow of wash water results in a largely increased capacity and cleaner separation, owing to the greater transverse distance the material travels on the table.

Further, this construction enables the use of a long feed-box. Attached to the table, it receives the reciprocating motion of the table, distributes the feed evenly over a large surface, and also, as a result of the shaking, the box becomes an especially efficient sizer in itself. The Overstrom long feed box is placed on the side of the table along which the mineral travels, *i. e.*, along the natural line the mineral takes when acted upon by the reciprocating motion and the wash water. The feed-box is also constructed so as to give a partial concentration. On a mixed feed it will collect the large, round mineral, which has a tendency to roll if left on the table direct, and discharge it through an opening in the forward end of the box, leaving this material high up out of danger. There is also a saving of clean wash water as a result of using a long feed-box.

Fig. 1 shows diagramatically the separation of the material into different zones. At the point D, which is the point of final separation, both the mineral particles and the silica come off light, with practically no mixture of mineral and silica, but with a clean separation, and consequently but little middlings. (This is an important difference from the action of rectangular concentrating tables, where at the point D both the mineral particles and the silica come off heavy and thick with a consequent mixture and an absolute necessity for the rehandling of this mixed material, which is usually called middlings.) The arrow in Fig. 1 shows the direction of the movement of the dirty water across the table. Otherwise this diagram is self explanatory.

Fig. 2 gives an external view of the Overstrom concentrator from the head motion end. In its latest improved design the concentrator embodies several important features. The table is made of structural steel and iron throughout, with the exception of the table top, which is made of California redwood. This construction gives great rigidity of frame with a greatly increased smoothness of table action. The table has been provided with a newly designed head motion, so as to advance the mineral particles more rapidly. To impart the reciprocating motion, rocking arms are employed and a most perfect table action is produced. The table top rests on four long rollers, which extend the whole width of the table and give more than 24 feet of bearing surface. No lubrication is needed, and the action is so that the rollers cannot wear flat. Concerning the details of the construction, as well as erection and operation, the reader may be referred to a fully illustrated pamphlet just issued by the Allis-Chalmers Co. on the Overstrom table (catalogue No. 124).

Fig. 3 shows an installation of Overstrom concentrators in the mill of the Baltic Mining Co., Redridge, Mich.

Exhibits at Boston Electrical Exhibition.

In the Boston Electrical Exhibition, held from July 15 to 22, the General Electric Company had one of the largest spaces in the center of the main floor, covering about 1300 square feet. The principal feature of this exhibit, and probably the newest thing in the entire exhibition, was the new G. E. 2½-watt incandescent lamp. The company's exhibit was brilliantly illuminated with these units. The lamp should be of special interest to our readers in view of the rôle which the electric furnace plays in the manufacture of the filament. (See the paper of J. W. Howell, p. 252 of our last issue.)

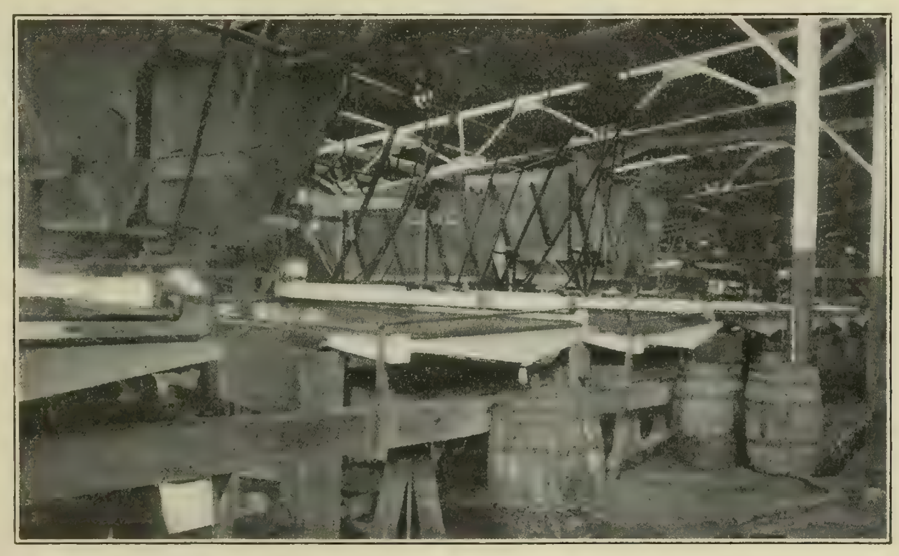

FIG. 3.—OVERSTROM TABLES IN MILL OF BALTIC MINING CO.

This is the first exhibition to the general public where this carbon filament has been shown, and the Electrical Contractors, under whose auspices the exhibition was held, may well feel proud to have their show characterized by such an important event. The new metallized carbon filament lamp is made in three sizes, Nos. 3, 4 and 5, the watt consumption being 125, 187½ and 250 watts per lamp, respectively. These lamps are used with a special line of Holophane pagoda reflectors, which are made in two forms, one which distributes the light below the horizontal plane, and the other which concentrates the candle power in a downward direction. The following table shows the ratings of these lamps:

Type Number	Total Watts per Lamp	Distributed Candle Power in Downward Direction with "D" Form of Reflector	Concentrated Candle Power in Downward Direction with "C" Form of Reflector
3	125	70	120
4	187½	105	180
5	250	140	240

They certainly present an attractive appearance, and as the actual gain is 20 per cent greater efficiency over the best obtained from the highest efficiency incandescent lamps of the past on an equal basis of life, they are sure to meet with a hearty reception. They are particularly suitable for use where the arc lamp is too large and the ordinary incandescent too small.

MERCURY ARC TOWER.

Another exhibit of the company of special interest to our readers was a mercury arc rectifier operated on 220-volt, single-phase, which was converted to 110 volts, direct current, the load being a circuit of the new incandescent lamps. This rectifier has a capacity of 30 amps. and was in constant operation. The application to storage battery charging and for electrolytic purposes in general is obvious.

A collection of small motors of various types was shown. This included alternating and direct-current motors. A new G. E. electric flat-iron also made its appearance here for the first time, and naturally attracted a great deal of attention from the public.

In attendance at the exhibit were Mr. C. B. Davis, Boston manager; Mr. J. P. Felton, supply manager, Boston office; Mr. O. F. Brastow, L. E. Smith and J. A. Wilson, Boston; Mr. E. M. Kinney, Schenectady, N. Y.; Mr. W. M. Wright, New York, and Mr. F. H. Gale, advertising manager. Other callers during the week were M. T. Beran, New York, supply manager, and Mr. D. R. Bullen, Schenectady.

Mercury Arc at the Elks Carnival at Buffalo.

The mercury arc lamp is of special interest to our readers on account of its well-known rectifying effect. It will undoubtedly be used to a large extent in future for electrolytic purposes and in connection with storage batteries, when only an alternating current supply is available. The following note refers to the use of the mercury arc lamp for producing special and very beautiful effects in street lighting.

The entire business part of Buffalo was in gala attire in honor of the nineteenth reunion of Elks in the week of July 10. But if the daylight scene was gorgeous, it was far excelled by the electrical decorations which literally turned the night into day.

Through the energy and resourcefulness of General Manager Charles R. Huntley, of the Buffalo General Electric Co., the city streets presented a night scene of splendor probably never before equaled in similar celebrations. Niagara electric power had been freely drawn upon, and was everywhere in evidence, through incandescent lamps festooned across the fronts of buildings in "welcome" signs, and in elaborate special designs embodying the characteristic Elk emblem.

The unique feature of this veritable festival of light was a "Mercury Arc Tower," located in Shelton Square. This was a fluted column of "Staff," 75 feet high, studded with 100 mercury arc lamps. Standing alone in its splendor, this tower produced a most startling effect, due largely to the characteristic greenish light of the mercury arc lamps. It stood out conspicuously in the mass of many colored incandescents covering the fronts of the buildings on the Square and the radiating streets.

The mercury arc lamps were operated in two series of 50 lights each from 4-amp. Brush arc machines, located in the Wilkinson Street station. As each lamp consumes only 160 watts, the operating expense was extremely low for so brilliant a spectacle. They are of the standard type for outdoor lighting, as developed under the direction of Dr. C. P. Steinmetz, and manufactured by the General Electric Company.

The new use of the mercury arc lamp on a large scale marks a radical step in electric lighting for decorative purposes. The incandescent lamp reached its climax for general illumination and for decorative effects at the Pan American Exposition four years ago in Buffalo, and it is interesting to note that the latest development in this branch of the art should be successfully tried in the same city and using also Niagara power.

This luminous tower was a special contribution to the carnival by the Niagara Power Co., the Cataract Power & Conduit Co., and the Buffalo General Electric Co. Mr. Huntley conceived the idea, and designs were prepared by the illuminating engineers of the General Electric Co., and this company furnished the material and superintended the installation.

Industrial Notes.

Slag Ladles and Cars.—A recent illustrated pamphlet of the Wellman-Seaver-Morgan Co., of Cleveland, deals with Dewhurst patent slag ladles and cars, which have already been extensively adopted in leading plants in Great Britain. They are either built for side tipping or end tipping as required. The ladles are tipped by a pull of the locomotive on the tipping chain, the locomotive pulling away from the ladle while dumping. The ladles are special shape, to facilitate the stripping of the slag and skull. Hence, in dumping, the skull follows the molten slag, requiring no hand work for its removal. The whole of the work is handled by the locomotive and its crew. The chief point is the great simplicity of design, since there is no complicated tipping mechanism, no steam or air cylinders, nor gearing, no tipping poles, no hand work in removing skull. The reduction in the amount of hand labor not only reduces the cost of operation, but also facilitates the work. The cars consequently make quicker trips and fewer cars are required to take care of the work.

Small Direct-Current Motors.—The nicely illustrated folder No. 4039, of the Westinghouse Electric & Manufacturing Co. deals with direct-current motors, type R. This type covers small motors, in sizes from 1-6 to 1¾ hp., and suitable for almost any direct-current service within their capacity. They are specially useful for driving small machine tools, small pumps, mixers, stirrers, labeling machines, etc.

We have received from the *Waterbury Brass Co.*, of Waterbury, Conn., their catalogue on brass, copper, German silver in sheets, rolls, plates, circles, wire, rods, brazed tubing and iron lined tubing. The catalogue gives first concise definitions of various kinds of brass and some other alloys, and notes on tempers, anneals and gauges. Then follow notes and tables on various brasses in sheet, roll and wire form; tables on the electrical properties of pure copper wire, "white metal" wire, and German silver wire; comparative tables of standard sheet and wire gauges (B. & S., Birmingham, inches, mm.); tables of weight and gauges of brass wire, weights per linear foot of brass and copper rods, weights per square foot of copper and brass sheets, weights of Muntz metal and Tobin bronze plates and rods, etc.; tables on B. & S. drill wire gauge; very full tables on weight per foot of seamless brass tubes and seamless copper tubes; tables of iron pipe sizes of seamless drawn brass and copper tubing; brazed brass tubing, and various useful tables.

The New Denver Mint.—Messrs. Geo. D. Feidt & Co., of Philadelphia, have received an order from the new Denver Mint for a general outfit of chemicals and chemical apparatus, including certain electric furnaces and heating apparatus, for use in the melting and refining department of the mint.

The De La Vergne Machine Company, of New York City, have issued a small illustrated folder on Koerting gas engines. It contains an interesting picture of a plant with 2000-hp gas engines driving blast-furnace blowing cylinders. These engines operate with blast-furnace gas of from 80 to 100 B. T. U. per cubic foot.

Cyanide.—In view of the steadily increasing demand for cyanide, especially in gold metallurgy, the United Alkali Co., Ltd., of Liverpool, has perfected processes for the manufacture of high-strength cyanide of sodium, for which as special advantages are claimed high cyanogen strength, great solubility, absolute freedom from sulphides, high extractive value and economy in freight charges. The cyanogen content is equivalent to from 105 to 110 per cent, and a higher grade from 120 to 130 per cent, compared with the ordinary potassium cyanide. It contains a proportion of carbonate of soda, which favors the action of the cyanogen in the solution by assisting to neutralize any acidity. This sodium cyanide is already in successful use in the gold fields of Australia, South Africa and India, and is now introduced in this country by Messrs. James Lee & Co. (founded 1825), 76 William Street, New York.

Curtis Steam Turbine Tests.—In view of the fundamental importance of the power problem in metallurgical and electrochemical operations, the following results of tests of a 2000-kw Curtis steam turbine, made by the General Electric Co., are very important. The turbine is a four-stage machine, designed in 1903, and recently changed in a few particulars as a result of experiments conducted during the past year. The machine as tested conforms as nearly as possible to the standard four-stage machines now produced by the General Electric Co., but is less efficient, since the changes made have been confined to the buckets, while other desirable changes could not be made without rebuilding the machine entirely. The machine operates at 900 r. p. m. The tests were made by Messrs. F. Sargent and L. A. Ferguson, of Chicago, by the most accurate methods available.

	Full Load.	Half Load.	Quarter Load.	Zero Load.
Duration of test in hours	1.25	0.91	1.	1.33
Steam pressure (gauge) in lb	166.3	170.2	155.5	154.3
Back pressure (absolute) in mercury	1.49	1.40	1.45	1.85
Superheat deg. F	207.	120.	204.	156
Load in kilowatts	2037.7	1066.7	555.	—
Steam consumption per KW. hour in lbs.	15.02	16.31	18.09	15.105

Low-Speed Generators for Electrolytic Refinery.—The output of our electrolytic refineries is constantly increasing. This flourishing condition is indicated by an order just placed with the Crocker-Wheeler Co., of Ampere, N. J., through its New York branch. The order calls for two Nr. 898 Crocker-Wheeler engine-type, direct-current generators, with a capacity of 10,000 amps. and 105 volts. These will be special machines with twenty-two poles. The speed will be 100 r. p. m. They will be installed at Carteret, N. J., in a plant where there are at present four Crocker-Wheeler generators of varying capacity up to 750 kw.

The Thermit Process in American Practice.—A suggestive and interesting paper on this subject was presented to the American Society for Testing Materials by Mr. Ernest Stütz, general manager of the Goldschmidt Thermit Company. The paper has recently been issued in pamphlet form, and is especially interesting since it shows how the thermit process of Dr. Hans Goldschmidt has been introduced for many purposes in this country. The first thermit was made in this country a year ago, and the success of its applications for many different purposes is attributed by Mr. Stütz very modestly, but at the same time undoubtedly with much justification, to the fostering care and ingenuity of American engineers. The essential point of the Goldschmidt thermit process is that it enables one to get highly superheated steel of any desired quality anywhere in anything like half a minute. The essential characteristic of thermit is that it welds by fusion, and by reason of this fact calls for the foundrymen's experience more than the blacksmith's. Its success depends on the proper material, shape and condition of the mould. Concerning this some useful hints are given by Mr. Stütz. The important problem of the welding of broken locomotive rails has been investigated by some thirty railroads in this country with more or less success. In all there are records of thirty engines with welded frames that have been in service for three months or longer. Another operation of interest to railroad men is the welding of spokes of drivers. In marine engineering, notable success has been obtained in making repairs with thermit, among others, by Mr. Des Anges, superintendent of the floating department of the Long Island Railroad. Some important repairs of great iron castings are also reported by Mr. Stütz. The process of rail welding for electric traction systems has been investigated in about thirty different American cities in actual operation, and about 5000 joints have been put in up to date. That steel foundries should have been first to recognize the possibilities of liquid steel that can be produced anywhere in half a minute goes without saying. There are already several of the largest with whom thermit is now as much a necessity as foundry sand.

Measuring and Recording Instruments for Chemical and Electrical Work.—We have received the catalogue of accurate measuring and recording instruments for all scientific and industrial purposes, made by M. Jules Richard, of Paris, represented in this country by Mr. Ernest H. Du Vivier, 14 Church Street, New York City. The catalogue deals especially with recording gauges, recording thermometers, pyrometers, ammeters and voltmeters. The apparatus are made for general work as well as for special purposes in different industries. Some of the instruments made for use in chemical works, sugar factories and refineries, etc., etc., are quite interesting. For instance, recording thermometers are made in a great many types for various purposes. Sometimes it is not only necessary to obtain a chart of the temperature, but an attachment is advisable by which, with the aid of an electric bell, the workingman is warned that a dangerous temperature is being reached. This is the case, for instance, in the distillation of ammoniacal products, for which purpose a recording instrument is made having a dial 9 inches in diameter and a tube covered with lead 6½ feet long. It is graduated between 30° and 130° C., or 86° and 266° F., and is fitted with a device to give warning when the danger points are reached.

Orders have been received by the Westinghouse Electric & Mfg. Co., from Nelson Morris & Co., for nineteen type CCL induction motors varying in sizes from 5 to 50 hp, and totaling 410 hp, and for eight motors of the same type from the Decatur Car Wheel & Mfg. Co., Birmingham, Ala. These are the latest design of induction motors which have been built by this company, and have just recently been placed on the market. On the same day an order was entered from the Illinois Steel Co. for eighty direct-current motors having an aggregate capacity of 3920 hp. These motors are all of the series-wound type, and range from 30 to 75 hp. The Rumford Falls Power Co., of Rumford Falls, Me., will install one 550-kw and one 800-kw alternating-current generators, which will be direct connected to waterwheels. The order includes two 540-hp variable-speed induction motors, and 120 type OD transformers for use on the distributing lines.

Personal.

Mr. B. H. Warren, president of the Allis-Chalmers Co., has returned by the Baltic from England, after a prolonged stay of three or four months abroad.

Dr. J. W. Richards, of Lehigh University, sailed on July 22 for Europe, to spend six weeks in Great Britain. He will be back in time for the Bethlehem meeting of the American Electrochemical Society.

We had a pleasant visit from Mr. Adolph Chalas, eldest son of the senior of Chalas & Sons, in London. Mr. Chalas is visiting this country to study electrochemical and metallurgical developments.

Prof. Henry S. Carhart sailed on the Graf. Waldersee to London, upon a special invitation, to join the official party of the British Association for the Advancement of Science, on their trip to South Africa, where the meeting of the association is to be held in Cape Town and Johannesburg. On his return he will attend a conference, called by the German Reichsanstalt, preliminary to the meeting of the International Committee on Units, which will meet in a year or two, according to the plan adopted at the St. Louis Congress. The preliminary meeting will be attended by the official heads of the United States Bureau of Standards, the (British) National Physical Laboratory and the German Reichsanstalt, and by the following four gentlemen: Prof. Carhart, Prof. Kohlrausch, Prof. Mascart and Lord Rayleigh.

Mr. W. M. Probasco has resigned as assistant to the president of the McGraw Publishing Co., to become the vice-president of the Search Light Publishing Co., of New York City. This company owns a most elaborate pile of clippings, articles and pictures on all subjects, classified and kept up to date for the use of all who require prompt information on any subject. As part of its plan the company publishes a weekly called the *Search Light*. Other departments of the company are the publication of books and the handling of the publishing and advertising accounts of important railway, manufacturing and engineering firms. Mr. Probasco was formerly assistant general manager of the Westinghouse Co.'s publishing, advertising and exhibition interests, and in this capacity he designed and organized the Westinghouse exhibits at the St. Louis World's Fair, which received the grand prize.

We received a pleasant visit recently from Dr. Nicholas D. Durdin, professor of metallurgy in the Polytechnic Institute of St. Petersburg. Prof. Durdin is a specialist in iron and steel, having been in charge of the open-hearth department of the Aboukoff Iron Works before being elected to his professorship, and having studied the metallurgy of iron under such lights as Chernoff, Heyn and Ledebur. While visiting America the professor has taken special work in metallography under Prof. Howe at Columbia, and is at present completing a course in the thermochemistry and metallurgy of iron with Prof. Richards at Lehigh University. Prof. Durdin says that the fundamental instruction given by some of our American technical schools in thermometallurgy and metallurgical calculations is the best in the world, and will be of ever increasing assistance in keeping American metallurgists at the head of their profession.

Digest of U. S. Patents.

Compiled by Byrnes & Townsend, Patent Lawyers, National Union Building, Washington, D. C.

Electric Smelting and Reduction Processes.

(*Continued.*)

Nos. 609,466 and 609,467, Aug. 23, 1898, Auguste J. Rossi, New York City.

Produces a ferro-titanium containing more than 5 per cent of titanium, *e. g.:* (1) Mixes an ilmenite ore containing from 35 to 60 per cent of iron and 10 to 40 per cent titanic acid, especially ilmenorutile, 16 parts, with charcoal, 5 parts, and cast iron, 100 parts, and smelts the mixture at a temperature not less than 3500° F., producing an alloy containing iron 90, titanium 8, and carbon 2; (2) smelts a mixture of iron ores containing 20 per cent of titanic acid and 60 per cent of iron, 100 parts, with charcoal, 25 parts, producing an alloy of iron 77.4, titanium 16.5 and silicon 6.10; (3) smelts a mixture of the titaniferous slag produced by the process of his patent 486,941, 130 parts, with powdered carbon, 25 parts, and pig iron, 100 parts, producing an alloy containing titanium 27.53, silicon 2.30, iron 59.15, carbon 10.41; (4) smelts a mixture of the slag, 125 parts, with carbon and titaniferous iron ore, 160 parts. In all cases the charge must afford sufficient iron to maintain a bath of molten iron. The smelting may be effected by an oxyhydrogen flame or in an open hearth furnace. An electric furnace, however, is preferably employed, consisting of magnesia brick. The charge is placed in a graphite crucible, within the furnace chamber, the intermediate space being filled with pieces of charcoal. Adjustable electrodes pass through the side walls into proximity to the crucible. In operation, arcs pass between the electrodes to the crucible and between the pieces of charcoal. After the smelting is effected, the crucible is lifted out and the product is poured into a mold. From 50 to 100 pounds of alloy containing 15 per cent of titanium have been produced from one charge.

No. 624,041, May 2, 1899, Charles B. Jacobs, East Orange, N. J.

Produces soluble barium compounds by mixing barytes with sufficient carbon to reduce a part only of the barium sulphate to sulphide. When the mixture is electrically smelted, a barium sulphide is first produced and then reacts on the remaining sulphate, with the production of barium oxide and sulphur dioxide. The second reaction is incomplete, the product from a charge of barytes 20 parts, and carbon 1 part, containing barium oxide, 60 per cent, and barium sulphide 40 per cent. The production of a ton of the product requires 1636 kw-hours.

(*To be concluded.*)

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Bethlehem Meeting of the American Electrochemical Society.

Bethlehem is a quaint old town, picturesquely located in the Lehigh Valley. In recent years it has come prominently before the general public through the establishment of its Bach Festivals, and is being called by some enthusiasts the Bayreuth of America. Independent of its justification for this claim, it certainly is a delightful little town, and it needs no great power of prophecy to say that the American Electrochemical Society did a wise thing in arranging to hold its autumn meeting, in September, in Bethlehem. It has been the experience of all the other big national engineering societies that those meetings which were held in comparatively small towns were the most successful and enjoyable—we may mention as instances the Great Barrington and Ashville meetings of the Institute of Electrical Engineers. We trust that the experience of the American Electrochemical Society will be the same. In the present case, the easy accessibility of Bethlehem from New York or Philadelphia is a distinct advantage, while an admirable programme of visits to plants, excursions and other social functions has been arranged. To professional metallurgical and chemical engineers Bethlehem and its vicinity offers special attractions in its famous steel, zinc and cement works, to which visits have been arranged, as is noted in the programme published on another page of this issue. Lehigh University will be the host of the society at its professional sessions, and it is unnecessary to tell our readers that Lehigh has been among the first American universities to promote electrochemistry, electrometallurgy and physical metallurgy.

◆

Labor-Saving Appliances in Laboratories.

Too often it is overlooked that with respect to purpose and character there is a very essential difference between the chemical laboratory of a university or college and the works-laboratory of a chemical or metallurgical industrial plant. The purpose of the chemical laboratory of the university is to make the student acquainted with chemical reactions and analytical methods. For this purpose as few labor-saving devices as possible should be placed in the hand of the student in the beginning. He should do all the work himself with his own hands in order to acquire that experimental skill without which purely theoretical knowledge is useless in practical analytical work. It is in the university laboratory that the chemist should acquire the faculty of overcoming difficulties which are liable to confront him at any time. Only by doing all the work himself can he acquire the resourcefulness and the well-balanced head which he needs more than anything else in later years in commercial work. It has been said that a physicist must be his own glass blower and his own mechanic. Nothing less should be required of the student of chemistry. Nor is this requirement unnecessarily exacting. The splendid results obtained with the most simple means in the old classical chemical laboratories like that of Bunsen prove the value of such a course. Naturally, under such conditions the chemist is compelled to devote a very great deal of his time in the laboratory, but the time is not wasted. Even if the work appears to him tedious at times, he is thoroughly recompensed in the end. The writer remembers with pleasure the time when he constructed and wound himself some electrical measuring instruments for his own use in electrochemical work. The time spent in such work is not wasted; its investment brings interest in the form of a most intimate knowledge—not to be acquired in any other way—of the working principles of the instruments which he has to manipulate. The same is true of all apparatus and appliances used in the chemical laboratory. Of course, this principle of letting the student do all the work himself, including the construction of the apparatus which he needs, should not go too far. When once he has acquired the necessary intimacy with methods and apparatus then he is ripe to have at his disposal the labor-saving devices which the ingenuity of modern engineering has placed at the disposal of chemists.

The works-laboratory of a chemical industrial plant has to fulfill quite a different purpose. If the chemist entering a works-laboratory is worth his pay he must be assumed to be familiar with analytical work. The works-laboratory is only one link in the complex mechanism of the industrial chemical organization, and its work should be ruled by the same commercial principles of doing things with maximum efficiency as any other department of the organization. But while in other departments the development has been in the direction of reducing cost of production by substituting automatic machinery for manual labor, many managements of chemical or metallurgical works have reduced the cost of the works-laboratory by substituting, to some extent, boys for chemists. As Dr. Edward Keller points out in the most interesting paper, the first part of which is published in the present issue, this tendency is wrong in principle. Analytical work requires theoretical knowledge, experimental skill and conscientiousness—qualities which cannot be fairly expected from a boy receiving small wages. The salvation lies naturally in following up the same principle which has proved so enormously useful in all engineering work, *i. e.*, in the systematic introduction of labor-saving appliances in the works-laboratory. We believe that this has not been carried out before anywhere with such consequence as in the Baltimore works-laboratory of the Anaconda Copper Mining Co. Its description by Dr. Keller should prove interesting reading to all whom it may concern, and it may be hoped that the introduction of labor-saving devices into the works-laboratory will become more and more general. It may be said that in many works-laboratories this development is already going on. In this way the chemist of the works-laboratory will get spare time which he may apply to useful chemical work. Even if the works-laboratory is always kept separate from the research laboratory—which is undoubtedly preferable in many cases—there will be enough important chemical problems for the works-chemist to solve in spare time. Then we may hope that the works-chemist and his old and useful art of assaying may no longer be (as Dr. Keller says) "too often lightly considered by those having an excessive knowledge of chemistry and metallurgy."

The Influence of Mineral Wealth upon History.

For the past weeks the entire world had focused its attention on the little town of Portsmouth, where the representatives of two countries were playing for a great stake. Like the national game of poker it was a game of bluff, of proposals and counter-proposals, of divining the thoughts of the opponents, of frankness, secrecy, suavity and veiled force, alternating rapidly across the board of play. The diplomats were playing for a great "jackpot" of minerals—a pot not contributed to by the players but made by nature's ever-working forces. In aeons past the country of Northern China was spotted with great basins. In these basins were accumulated large peat bogs in some places, and again ferruginous waters laid down their mineral freight. The metamorphic changes of years have turned these deposits into coal and iron ore, in generous measure as has been done in the United States in Alabama and elsewhere, in England, in the Cleveland district, and in Germany in the "Minette" provinces of Alsace, Lorraine and Luxembourg. Coal and iron ore in proximity make cheap pig iron.

Coal and iron are the foundations on which is built a great commercial structure. Reports, vague and indefinite of the size of these deposits, are known to the spectators. Russia and Japan, however, have some accurate knowledge of the mineral wealth of Northern China. Careful and systematic prospecting has been done by the bureaucracy—benevolent and malevolent—and by the secret service of forehanded Japan. Coal and iron are the essentials of a modern civilization as much as wheat and corn. It is for the possession of these fields that men and treasure were sacrificed on the plains and mountains of Manchuria. For the eating of that "big fat pie" might makes right, and the mightier will become even more mighty by the possession of the prize.

There have been in the past many wars, commercial and actual, for the possession of choice bits of the earth's crust. Chile, by an aggressive war, despoiled Peru of her nitrate beds, which supply an ingredient of fertilizers of plant life and the raw material of the most important explosives. The far-seeing Bismarck brought on the Franco-Prussian war. As victor, Germany took provinces containing iron deposits, the second in size among the proven deposits of the world. And it is a curious fact that the Thomas basic Bessemer process has increased the value of these deposits by the utilization of its phosphatic slags of immense value as fertilizers. Germany won an important source of natural wealth, then unknown, in the phosphate in those oölitic ores. Another war for mineral wealth was the Anglo-Boer. England expended millions of pounds sterling and shed much good British blood to gain the control of the Rand gold deposits, made valuable only because McArthur and Forrest had made practical the selective solubility of gold ores in cyanide of potash. But gold is of secondary importance to coal, iron and agricultural wealth. It really has only a fictitious value, due to human sentiment to its beauty and to its scarcity. Things have a basic commercial value because they satisfy real human needs. In older times Spain, England and France sent adventurous men to take the gold of America. Gold has been the bone of contention for centuries. But still, in the hands of men like Francis Drake, the source of power in those early days was still iron, bronze and wood.

In the Middle Ages the coat of mail, and later the musket, was the controlling factor in war. Nowadays the nickel in nickel-steel armor plate makes the battleship resistant to the shots of the enemy, and thus it becomes a tremendous engine of destruction. It gives a nation with battleships a differential advantage, as did the coat of mail the armed knight. In a prolonged modern war unless a new alloy of iron equal to nickel-steel is found, the possession of the nickel mines of New Caledonia and Ontario would be a similar differential advantage. Smokeless coal is another source of naval advantage. Thus it is that the sources of material wealth will always be fought for to give national supremacy. Mineral wealth is the cause of war, and once won gives the stronger greater strength. Metals are agents of destruction as well as production. Battles have been fought with metallic weapons and armor since the days of Tubal Cain. In times of war the ploughshare and sickle are wrought into sword and shield. Anthropologists tell us that human nature has changed but little in 2000 years. Then it is not remarkable that the primitive stone hatchet and wooden club are seen to be the prototypes of the armored battleship and high-power rifle. The state of prehistoric man and our modern refined civilization have the same fundamental characteristics; energy and courage to wield the weapon of offense are as necessary to-day as they always were and always will be.

That the Russo-Japanese war has been largely affected by mining and metallurgical advances is the logical outcome of natural evolution. Iron and coal are as necessary to the life and growth of a nation as are the iron of the hæmoglobin of the blood and the food of the individual man. The Japanese appreciated this, and desired the control of the iron and coal fields that Russia took from helpless China. They apparently argued that "to the victor belong the spoils." They learned it when Russia took from them the coal beds of their island of Sakhalin thirty years ago. They have not forgotten it since.

Mathematics in Engineering.

In any branch of engineering the ultimate goal is to reduce the principles and methods to exact mathematical form. Their application is then kept well in hand. In order to solve a given problem of practice, it suffices to formulate precisely the requirements, and the solution follows by cut and dried methods of calculation and with the exactness of algebra. The obvious danger of the prevalence of such methods is that the exactness degenerates into rigidity and inflexibility, while engineering practice places before us constantly changing requirements. We would deceive ourselves would we believe that we can treat an engineering problem with the same mathematical precision as, for instance, an ideal case of analytical mechanics. In engineering practice we have not to do with ideal cases, but in order to get a simple method of calculation we change the requirements of our real case, so as to get an ideal case, by introducing certain simplifying assumptions, etc. Experience teaches us within which limits we are justified to do so. When now a case arises not within these limits, we have to revise the starting suppositions as well as the whole method of calculation. The chief trouble is that in such calculations the starting point and the solution have an engineering meaning, while the links connecting the two ends of the chain are pure mathematics. The only effective remedy is that the engineer who makes the calculation keeps always in mind what the physical meaning of each and every formula is. Then the whole mathematical calculation becomes simply a statement in short-hand of the technical reasoning. All technical suppositions are clearly kept in mind, while the precision of mathematical methods prevents any logical error.

That consequent and exact technical reasoning is possible without mathematics, is conclusively proven by Faraday's life work. Yet his way of reasoning seemed obscure until Maxwell devised mechanical models or analogies for Faraday's physical conceptions, and thus brought them within the realm of mathematical calculation. It would seem that the average man needs both faculties: the ability of technical reasoning with the aid of technical (physical or chemical) conceptions (often involving a hidden hypothesis), and the ability of checking the results of such reasoning by mathematical analysis. Technical conceptions may mislead us to see in them more reality than there is in them and to apply them beyond the limits of their usefulness. Mathematical analysis is apt to let us lose sight entirely of the real phenomena which our analysis covers. A thorough blending of the methods of technical conceptions and of mathematical analysis is needed for the average man. The genius who revolutionizes sciences and arts is not thus handicapped; he devises and works out his own methods. Faraday, who revolutionized electromagnetic science and laid the cornerstone for electric engineering, worked with physical conceptions of his own. Willard Gibbs, whose revolutionary work in the thermodynamics of chemistry is being recognized more and more from day to day, gave the results of his researches in the form of pure mathematics. To make his work the common property of chemists, Gibbs must be supplemented by the chemical experimenter. Gibbs' phase rule has become the basis of a new classification of chemistry through the work of the school of Roozeboom. Gibbs' exposition of the conditions which govern chemical equilibrium has already shed a flood of light on many phenomena, but far more experimental work will be needed to appreciate in full measure the real grandeur of Gibbs' life work. When and how far his work will affect chemical and metallurgical engineering is impossible to say at present. Under present conditions the chemical and metallurgical engineer needs only a small amount of mathematical apparatus. The fundamental principles which govern all his reactions are contained in molecular equations for the calcution of the masses and energy equations for the calculation of the physical conditions under which his reactions take place. What he needs more than anything else at present, are empirical constants (specific heats, heats of fusion, etc.) for use in his thermochemical calculations.

Some Impressions at the Annual Meeting of the Society of Chemical Industry Held in England in July, 1905.

By Maximilian Toch

In Sept., 1904, the English section visited the United States, and for a year prior to their coming a committee met at the hall of the Chemists' club on the last Friday of each month to perfect the arrangements, and the reception the English branch received, when they arrived in the United States and were taken on a tour to St. Louis and back, is now history.

When we arrived in England on the 8th of July, some thirty-five of us had been passengers on the "Campania," and the recollections of that trip are certainly not to be forgotten. The sea voyage was perfect—sunshine and smooth water every day. On the 4th of July, the inimitable Dr. Wiley conducted the Independence Celebration. When we arrived at Liverpool we felt as if we were one huge family. Our London quarters were at the Hotel Russell, which is a new hotel, situated in Russell Square, that reminds one very much of Gramercy Square, in New York, it being situated the same as Gramercy Square is, in the old residential portion of the city. The University College of London is in Gower Street, only a few minutes walk from the hotel, and the opening meeting took place at 11 o'clock Monday morning, July 10, in the lecture hall, replete with scientific history.

Let it be remembered for all time that Dr. William H. Nichols, our president, unquestionably carried off the honors of the day, and we felt justly proud of him, not only at the opening meeting but throughout the whole trip. His ability to speak extemporaneously is marvelous, but his ability to say the right thing at the right time is much more remarkable.

To give a detailed history of the pleasant trip and of all the good times that we had from London to Glasgow would form a volume of many hundred pages, so it can only be said that wherever we went the hospitality that was shown us was simply overwhelming. In London, the Lord Mayor gave us a reception, and wherever we went the Lord Mayor of that city received us, and the freedom of the city was granted us. There was one trip in particular of which we all spoke, and it was the journey in a private train from Newcastle to Edinburgh. The scenery, the train itself, the two dining cars, the excellent dinner and the condition of the weather were all of the best, and when we arrived at Edinburgh every man and woman felt in superlative condition.

The trip through Scotland calls for some comment, and as one of our party said, the only trouble with Scotch hospitality was that there was insufficient Apollinaris served with it.

On the last day, during the boat trip around Glasgow, four silver loving cups were presented to Dr. J. Lewkowitsch, Mr. Julian Baker, Mr. Gordon Salamon and Mr. Thomas Tyrer. At night the final reception was given us by the Lord Mayor, and when we parted from our many friends we felt that the British are our friends and brothers, and that the Society of Chemical Industry is stronger than ever.

Bethlehem Meeting of the American Electrochemical Society.

The eighth general meeting of the American Electrochemical Society will be held in Bethlehem, from Sept. 18 to 20. The first session will be held on Monday, Sept. 18, afternoon. Members leaving New York with the Black Diamond Express (Lehigh Valley) at 11.55, or Philadelphia at 12.30, will arrive in time for the first session. The following papers will be presented at this meeting:

R. Threlfall—"A Standard Method of Measuring the Specific Resistance of Electrolytes."
Oliver P. Watts—"Notes on the Use of Aluminum as a Reducing Agent."
G. Gin—"New Furnace for the Electrical Manufacture of Steel."
G. Gin—"Note on the Electrical Resistivity of Iron and Steels at High Temperature."
A. G. Betts—"An Electrolytic Process for Refining Silver."
A. G. Betts—"Notes on the Electrometallurgy of Antimony."
C. F. Burgess—"Report on the Electrochemical Exhibits at the Louisiana Purchase Exposition, 1904."
Charles Baskerville—"Artificial Willemite."
W. D. Bancroft—"The Chemistry of Electrochemistry."
L. Addicks—"Ammeters for Electrolytic Work."
F. D. Easterbrooks—"Electrolytic *vs.* Sulphuric Acid Parting of Bullion."
M. Toch—"Insulating Paints."
R. Von Foregger—"The Utilization of Active Oxygen, Chemically and Electrically Produced."
N. M. Hopkins—Title to be announced later.
E. F. Roeber—"Thermodynamics of the Electric Incandescent Lamp."
H. Schlundt and R. B. Moore—"The Chemical Separation of the Excited Activity of Thorium."
R. B. Moore and H. Schlundt—"Radioactivity of Some Natural Waters of Missouri."
C. F. Burgess—"Some Observations on the Influences of Arsenic in Pickling Solutions."
O. W. Brown and W. F. Oesterle—"The Electric Smelting of Zinc."

Exhibit of Voege's System of Index Collection of Swiss Electrochemical Patents, Wednesday, p. m., by C. F. Carrier, Jr.

On Tuesday afternoon the plant of the Lehigh Zinc Co. and the Bethlehem Steel Co's. world-renowned works will be visited. For Wednesday afternoon an option of either of two excursions is offered. The first is by trolley to one of the big cement works in the vicinity, the other an excursion to Mauch Chunk, including a trip around the Switch-Back. Any who remain in Bethlehem Wednesday afternoon are cordially invited to attend the opening exercises of Lehigh University.

A smoker will be held at one of the commodious club houses in Bethlehem on Monday evening. A lunch will be tendered to the visiting members by the authorities of Lehigh University, in the gymnasium, on Tuesday, Sept. 19, at 12.30 p. m.

A subscription dinner will be served at the Eagle Hotel, Tuesday evening, Sept. 19. The Eagle Hotel is to be the convention headquarters.

Besides the attractions which the beautifully situated historic town of Bethlehem offers in itself, a special programme of entertainment is provided for the visiting ladies. For Monday afternoon a carriage drive, starting from the Eagle Hotel at 3.00 p. m., has been arranged through the Bethlehems to South Mountain, with a visit to the grounds and buildings of Lehigh University. In the evening there will be an informal reception by the ladies' committee to the visiting ladies and their escorts at the residence of Professor and Mrs. J. W. Richards. On Tuesday morning an excursion will be made to Easton, with a visit to the large works of the Standard Silk Co.; lunch will be taken at Huntington, followed by a visit to Wygatt Mountain and grounds and buildings of Lafayette College, while in the evening a subscription dinner will follow, as already mentioned. For Wednesday morning an excursion is planned to Nazareth, with visits to the quaint Moravjan buildings, Nazareth Hall, school, Indian burying ground and the Whitefield house. For the afternoon the option of either an excursion to Mauch Chunk or the attendance of the opening exercises of Lehigh University is offered.

Any inquiries are to be addressed to the secretary of the local committee, Mr. Walter S. Landis, Met. Eng., Bethlehem, Pa.

Report of the Karlsruhe Meeting of the German Bunsen Society.

By H. Danneel, Ph. D., and J. K. Clement, Ph. D.

The following is a concise report of the papers presented at the general meeting of the Bunsen Society for Applied Physical Chemistry, held in Karlsruhe from June 1 to 4.

Electrolysis With Alternating Current.

A paper on this subject was presented by Prof. Max LeBlanc. When an alternating current is passed through a solution of $CuSO_4$, with copper electrodes, the metal is dissolved by one-half wave of the current (during one half of the period) and is immediately redeposited by the next half wave. The total effect is that the electrodes remain undissolved. If, however, there is an opportunity for the metal to enter when dissolved, into a compound from which it cannot be deposited, even by direct current (*e. g.*, in a KCN solution of Cu), H_2 and not Cu is liberated in one-half- period, while Cu is dissolved in the next half period, *i. e.*, the total effect observed is the solution of the metal in accordance with Faraday's law. This holds only with alternating currents of low frequency. When the frequency is so high that the Cu has not time to enter into the complex cyanic compound, the action is the same as with copper electrodes in $CuSO_4$: the total effect is no solution of Cu. With high frequencies Cu does not dissolve; with low frequencies it dissolves with 100 per cent efficiency. Between these limits the amount dissolved decreases with increasing frequency. This is true of the majority of the metals.

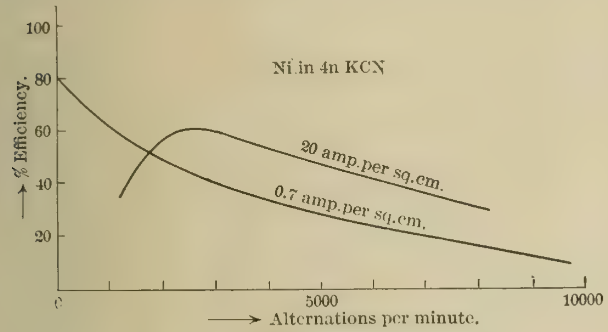


FIG. 1.—ELECTROLYSIS WITH ALTERNATING CURRENTS.

From the results of experiments, inferences may be drawn concerning the velocity with which the complex Cu cyanide compound is formed. The velocities with which substances produced by electrolysis enter into unsoluble compounds may be determined in the same way. In the Luckow process, for example, with lead electrodes in a carbonate solution, with low frequency current, white lead is formed; the lead has sufficient time to form the insoluble compound during one-half of the period of the current. With increasing frequency, the percentage output of white lead diminishes. With high frequencies no white lead is formed.

These remarks apply to symmetric alternating currents only. With an unsymmetric current waveform—for example, if the positive portion of the current wave is three times as long as the negative—and with Cu electrodes in $CuSO_4$, the total effect is the solution of Cu to the extent of 50 per cent of the theoretical amount required by Faraday's law, and this is independent of the frequency. This result is confirmed by experiment, but is true only of the electrodes with crystalline surfaces. If the Cu is heated above its melting point and suddenly cooled in alcohol, it is converted into a more stable modification which dissolves less readily. Some further deductions are drawn from this by the author, who also cites some observations as proof that the passivity of metals cannot always be explained by the assumption that the surface is charged with oxygen. In $CuSO_4$ the two modifications of copper have a difference of potential of 0.013 volts. This difference does not disappear when the electrodes are connected over night through a resistance of 170 ohms.

Brochet and Petit found that the curves whose abscissas and ordinates represent, respectively, frequency and output are abnormal for some metals. While the curve for Cu in KCN falls gradually from zero frequency and 100 per cent output, to infinitely great frequency and zero output, the curves for Ni, Fe and other metals beginning at zero frequency, rise to a maximum, and then fall with increasing frequency. Danneel has shown (*Zeitschrift für Electrochemie* 10, 741) that the cause of this phenomenon lies in the velocity with which these metals change over to the passive state. LeBlanc shows that this is the correct explanation. When a four-normal solution of KCN is electrolyzed between Ni electrodes with alternating current of current density, 0.7 amp. per square centimeter, the output curve is normal as is shown in Fig. 1. Under these conditions Ni as anode does not become passive, even with direct current. With a current density of 20 amps. per square centimeter, the curve shows the characteristic maximum. With this current density Ni becomes passive as anode with direct current.

Painting.

Prof. W. Ostwald lectured on this subject. He possesses an unusual talent for painting, and has written a book on the application of physical chemistry to the technique of painting. In this he has shown that many artists are unscrupulous in that they do not apply themselves to the physical side of their art. The result is that expensive pictures are often ruined in a short time. In this paper Ostwald gives some particularly good examples of the application of physical chemistry.

It is well known that oil paintings become full of rents very rapidly when the paint has been put on too thick. The cause of this is that the canvass on which the pictures are painted stretches in the course of years, whereas the oil paint shrinks. This defect may be avoided, first by taking some other material, *e. g.*, aluminium instead of canvas; secondly, by applying the paint as thin as possible, as in the case with all old paintings, which are well preserved, *e. g.*, the Sistine Madonna. The paint on the Sistine Madonna is so thin that the threads of the canvass can be seen through it.

The shrinkage of the paint is caused by the oxidation of linseed oil which is used in mixing the colors. Ostwold shows that the process of oxidation is autocatalytic, *i. e.*, the products of oxidation themselves accelerate the process catalytically. This process is of importance for the technique of drying. The much used white lead is very poisonous and in time turns black—due to the sulphur which is contained in many colors. As a substitute for white lead, Ostwald suggests lithopon, which is not sensitive to light. He illustrated his remarks by some of his own pictures which were exhibited.

Geological Thermometer.

Prof. van't Hoff in a paper on this subject pointed out that from the constitution of the natural salt beds north of the Harz Mountains in Germany, inferences may be drawn concerning the prevailing temperature on the earth at the time the salt beds were formed. For example, anhydrous Na_2SO_4 (Thenardite) is formed from sea water at temperatures above 29° C. At lower temperatures the hydrated Glauber's salt, $Na_2SO_4 + 10H_2O$ is formed. Anhydrous salts, gypsum for example, form with water a hard mass. Glauberite melts at 29° C., so this must be the temperature at which Glauber's salt is formed. But this melting-point is lowered by the presence of foreign substances. By determining analytically the amounts of the various substances contained in Glauber's salt, one may obtain an indication of the temperatures at which the natural salt beds were formed.

The following examples are given: Astrakanite, 5°; Glauberite, 10°; Thenardite, 14°; Kieserite, 18°; Leonite, 18°;

Langbeinite, 37°; Löweite, 43°; Vanthoffite, 46°. Inferences regarding the temperature may also be made from a study of the conditions under which minerals occur together (paragenesis): Löweite and Glaserite can be deposited out together only above 57°; Kieserite and KCl above 72°. According to experiment 72° is the highest temperature which could have prevailed at the time the salt beds were formed. A salt sea evaporating by ordinary summer heat assumes this temperature.

Hydrogen Peroxide.

Prof. **Walter Nernst** spoke on the formation of hydrogen peroxide at high temperatures. The question is, whether H_2O_2 is formed from steam + O_2 at high temperatures. No trace of H_2O_2 can be found in a mixture of the gases through which electric sparks have been passed. The reason for this is probably that the H_2O_2 if really formed by the sparks, is too rapidly decomposed in the moment in which it leaves the hot spark gap. If this is correct it must be possible to obtain H_2O_2 by cooling the products of the reaction very suddenly.

Nernst showed that H_2O_2 can be obtained by directing an oxyhydrogen flame against the surface of a vessel of water, or by producing electric sparks under water. It remained to be shown that the velocity of decomposition of H_2O_2 is very great. To determine the decomposition velocity of H_2O_2, Nernst passed a current of air through a solution of H_2O_2, and then heated the mixture of air + H_2O_2, at a fixed temperature, for a definite length of time in a glass tube of definite dimensions. The velocity of decomposition is about as great as that of ozone. The reaction is bimolecular, the decomposition takes place in accordance with the equation: $2H_2O_2 = 2H_2O + O_2$.

From the quality of the decomposition velocities of H_2O_2 and O_3, and from the fact that, in the gas, electric sparks produce the latter, but not the former, it must be concluded that O_3 is formed by ultraviolet light, which is radiated from the electric sparks, and that ultraviolet light does not form H_2O_2. Both gases, H_2O_2 and O_3 are formed under the influence of heat. The gas which is formed by the spark decomposes on leaving the vicinity of the spark.

Radioactivity.

Prof. C. **Engler** spoke on the radioactivity of the thermal springs in Baden-Baden. The speaker portrayed the geological origin of the radio-active springs of Baden-Baden, and described a method and a transportable apparatus which he used in his determination. The mud from the springs contains radioactivity in much greater mass than the water. The radio-active substances seem to be deposited with manganese peroxide, for the amount of manganese peroxide varies parallel with the amount of activity, though not proportional to it. The mud was analyzed. No thorium was found, although the decay curve is of the type characteristic of thorium emanation. It must therefore be assumed that a very strongly active new element is present. Radium also was found.

Tantalum.

Dr. **Werner von Bolton** presented a paper on tantalum and the Siemens & Halske tantalum lamp. Since we have already reported at some length on the author's work on pages 52 and 118 of our present volume, only new information shall be mentioned here. The speaker exhibited a regulus weighing 64 grams, in the smelting of which 75 hp. was required. This great amount of energy is the cause of the high price of the metal: the ore occurs quite frequently on the earth. The metal can be easily worked into very thin sheets and into fine wire (down to 0.03 diameter).

Interesting from an electrochemical standpoint is the ease with which it becomes passive. If 120 volts be applied to a cell of two tantalum electrodes in H_2SO_4 almost no current flows. But if one electrode be replaced by Pt only one half-wave of an alternating current can pass, *i. e.*, it can be used as a rectifier to convert alternating current into direct current. Glowing tantalum attacks water with the evolution of H_2 and violent combustion. Fine tantalum wires can be ignited with a match and burn in the air.

The speaker exhibited some tantalum lamps of 25 c.p. fed with an alternating current of 0.35 amp. at 110 volts. He heated his lamp to over 2000°, and obtained 350 c.p. with 0.3 watts per c.p. On raising the temperature the filament melted.

Physical Chemistry of Wine.

Prof. **Th. Paul** presented a paper on this subject. Up to the present time the volumetric method has been used for determining the amount of acid in wine. It is well known that the sour taste of wine does not depend on the total amount of acid contained, but on the concentration of hydrogen ions; a solution of weak acid is often not at all sour to the taste when a strong acid of the same concentration is decidedly unpleasant.

Wine is an alcoholic-aqueous solution of electrolytes and non-electrolytes, sulphuric acid, phosphoric acid, acetic, lactic, succinic and malic acid, glycerin, sugar, ester, tannin, dyes, etc. There are four methods for determining the concentration of H-ions: 1. Measurement of conductivity (on account of the great number of substances present this method does not give unequivocal results). 2. The application of hydrogen electrodes (it gives reliable results, but is too difficult for the wine dealer to manipulate). 3. The inversion of cane-sugar. 4. Catalysis of ethyl acetates. The last two methods, especially the inversion method, can be used very satisfactorily.

The speaker described an apparatus for measuring inversion. The determination was made at 76°, because at high temperatures it can be made more rapidly—the inversion constant is 0.0000097 at 25° and 0.00469 at 76°. Furthermore, one source of error is avoided, viz.: the ferments, which are always present in wine, also produce inversion. This ferment is killed at a temperature of 76°.

(To be Concluded.)

CORRESPONDENCE.

Genesis of Chemical Elements.

To the Editor of Electrochemical and Metallurgical Industry:

Sir: The editorial note on the subject of Genesis of Chemical Elements in your July issue, leads me to call attention to a brief article by Robert T. Hill, of United States Geological Survey, entitled "Source of Volcanic Water," which appeared in *Engineering and Mining Journal* of July 6 last.

It seems to me that the theories of Arrhenius and J. J. Thomson are related to a degree calling for a careful study of these theories, each as related to and affected by the other.

If the views of Arrhenius are found to be tenable and the earth's interior has a temperature more nearly the solar temperature than has hitherto been assumed, there are better reasons than ever for believing, not only that uranium, radium, polonium, helium and other elements are factors in atomic disintegration, but that a similar genetic relationship exists between all of the chemical elements, and that while marked radioactivity is shown by relatively few elements at present, an equal or greater radioactivity may have been shown in earlier stages of the earth's development by all of the elements of lower atomic weights, though they now appear to be entirely stable.

In conclusion, I add a copy of a note penciled on the margin of a work on chemistry nearly three years ago, after an apparently fruitless study of a table of "The Periodic System of the Elements":

"All 'elements' alike (*no elements*); when matter is in a condition, so to speak, of *a solid gas*, reached by sufficient simultaneous heat and pressure. Gravity is the one problem."

Roanoke, Va. **Horace M. Engle.**

Labor-Saving Appliances in the Baltimore Laboratory of the Anaconda Copper Mining Company.

In a most interesting paper recently presented before the American Institute of Mining Engineers, Dr. Edward Keller points out that the ruling principle in shop and factory, *i. e.*, to secure the greatest output at least cost by the substitution of machinery for manual labor, has had little effect on the chemical laboratory in our industrial establishments. There the methods of analytical work have been carried on along individual traditional lines. Long series of filtrations and stirring are still performed singly by hand in a slow and laborious manner. As a rule, the chemist wastes much time by such manual labor, which, in a modern establishment and under up-to-date management, could be more advantageously applied to useful chemical work; and such higher chemical work is gradually and surely becoming recognized as an essential extension of old-time methods of assaying. While it may be said that in some laboratories, boys and not chemists, do the manual labor, it should be borne in mind that all chemical work requires care and accuracy, and in these respects boys are naturally a source of concern to the responsible chemist. Moreover, the cheapness of child-labor is largely offset by the increase in the expense account of laboratory materials, caused chiefly by breakages resulting from carelessness.

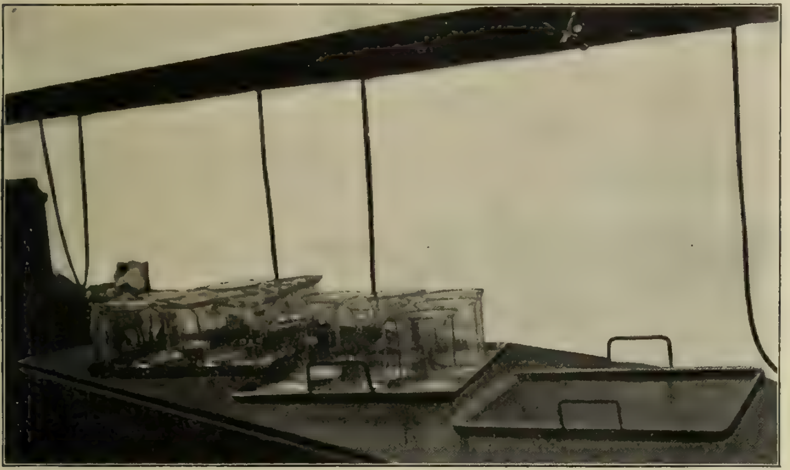


FIG. 1.—METALLIC TRAYS HOLDING NINE NO. 5 GRIFFIN BEAKERS.

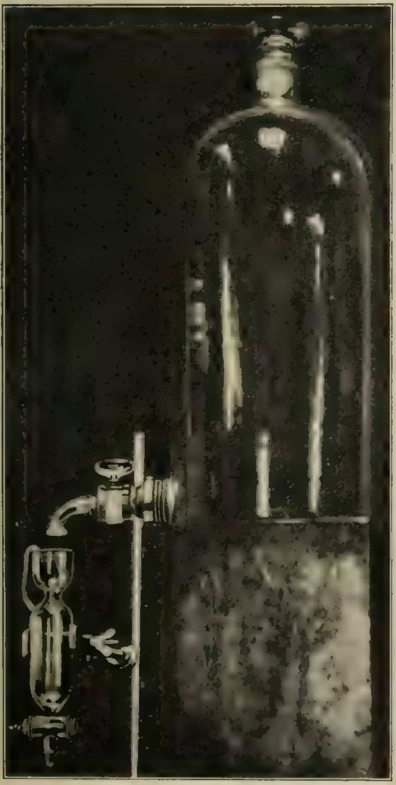


FIG. 2.—CYLINDER FOR MEASURING ACID.

On the basis of such considerations, Dr. Edward Keller, when authorized by the Anaconda Copper Mining Company to build a new laboratory in Baltimore, Md., decided to make its equipment superior to any that exists in this country or abroad. For this purpose he devised a great many labor-saving appliances. Their introduction has resulted in economy in several ways. Much labor has been saved; breakage of expensive glassware has been very largely eliminated, and the time of the furnace work, and, consequently, the consumption of gas, have been much reduced. Furthermore, the gain has been a moral one, and work formerly regarded as tedious has become more of a pleasure; especially has the sojourn in the furnace room during the hot summer months been rendered cooler by being greatly shortened.

These labor-saving appliances of Keller may eliminate the laboratory boy, but if not, they will make him more reliable. They also increase enormously the quantity of chemical work that one man can do in a day. In the domain of his own laboratory the change from the old systems to the new is considered to bear about the same relation as the change from ancient horse-car to the modern rapid transit.

A fully illustrated description of these labor-saving devices will undoubtedly be of great interest to our readers. For the permission to publish these extracts of Dr. Keller's paper we are obliged to the Council of the American Institute of Mining Engineers.

General Arrangement.

In such a carefully planned laboratory as this the general arrangement is, of course, a most systematic one; the ruling principle was to design it so that the number of movements required from the beginning to the end of every operation is reduced to a minimum.

The hygienic arrangements of the room were given proper attention, especially the ventilation, which, as a rule, is only made satisfactory by the proper construction of the hoods. At the Baltimore works-laboratory the two-compartment hood, built of brick, is plastered inside, has a tile floor, the openings into the laboratory being supplied with vertical sliding glass doors. The stack, 21 feet high, is directly over the slate roof of the hood, which tapers from two sides toward the openings of the stack. The cross-section of the openings from the hood to the stack equals that of the free space in the stack. The hood is built against the wall of the building, and parallel to this wall is a second one, leaving a free space of 4 inches between them, which communicates with the hood by a number of 2 x 4-inch holes along the floor, and with the stack by a larger opening at the highest point of the hood. This construction gives a good draught along the floor, as well as at the apex of the

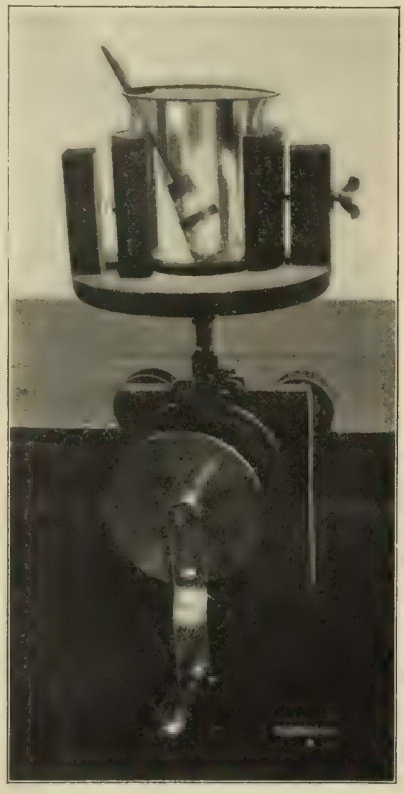
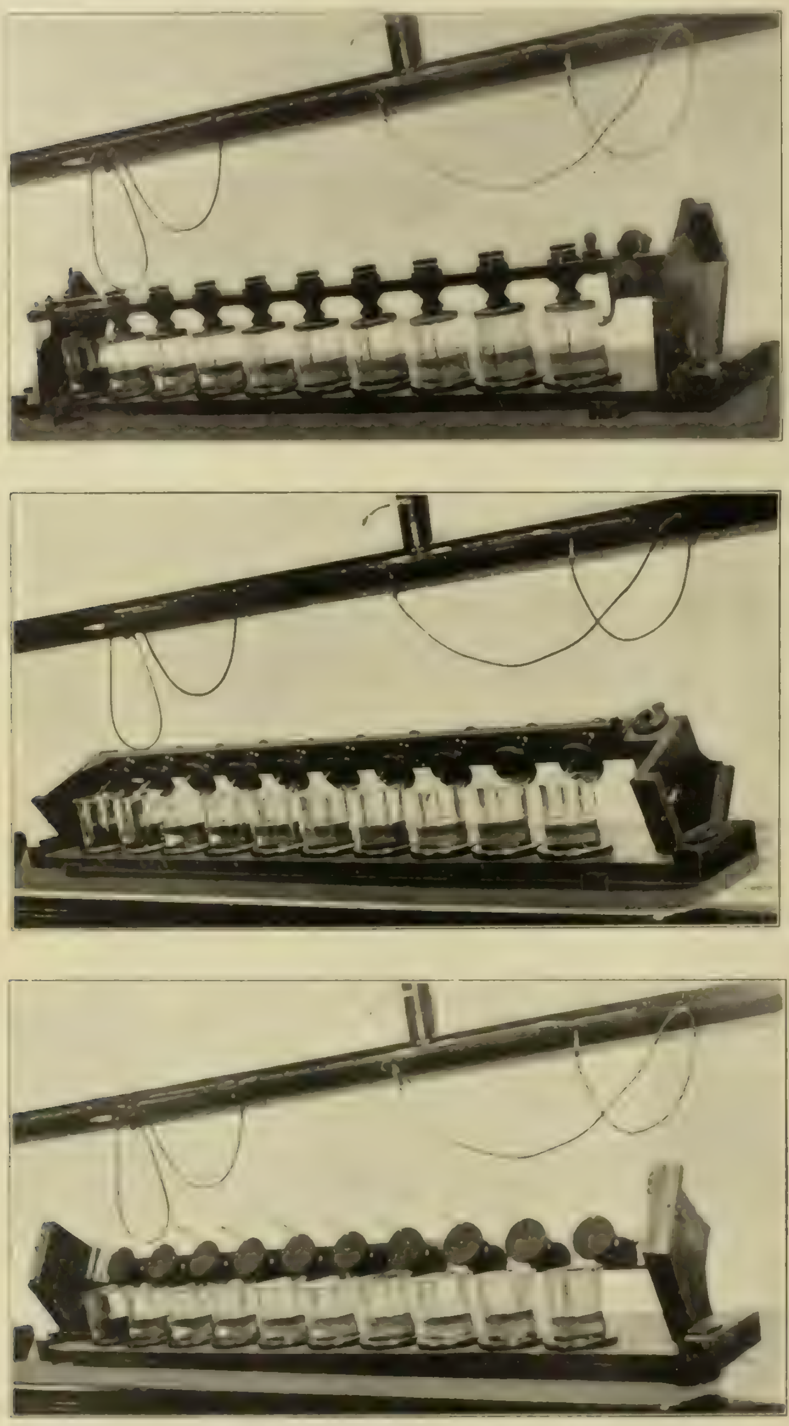


FIG. 9.—THE "POLICEMAN" IN BEAKER IN ROTATING MACHINE.

[illegible] acid, regardless of the amount of copper that is being dissolved with nitric acid, there is never any noxious odor of "red fumes" in the work room of our laboratory.

With regard to cleanliness of work in the laboratory, Dr. Keller quotes his first teacher in practical chemistry, Hermann von Fehling, who used to tell his students that *Der Chemiker muss im Frack-Anzug arbeiten koennen.* "This remark made a lasting impression upon me, and I have always disliked the sight of black or yellow hands, or of work-clothes having many spots and holes. It is not given to everyone, however, to be as tidy as Fehling was, but it is possible to aid natural defects by artificial means."



FIGS. 3, 4, 5.—THE STIRRING MACHINE.

[illegible] for Handling Beakers and Acids.

Assaying forms the bulk of the work at the Baltimore laboratory. Scores of silver-determinations and hundreds of scorifications and cupellations for gold are often made. Silver, and sometimes gold, is determined in the metallic copper material by what is commonly known as the combination method, which consists in dissolving both copper and the contained silver in nitric acid, leaving the gold as a metallic residue. The silver is reprecipitated in the form of chloride and, with the gold, is separated from the copper solution by filtration. The incineration of the filter, scorification with metallic lead, and cupellation with a subsequent parting of the two precious metals, completes the assay.

Keller has simplified this operation first by the introduction of metallic trays to hold the No. 5 Griffin beakers, thus avoiding handling them either singly or in pairs. The trays, shown in Fig. 1, hold nine beakers, and as soon as the copper has been dissolved the tray may be placed directly on the fire in order to expel the nitrous fumes by boiling the solutions.

The acid used to dissolve the copper is kept in a 5-gallon bottle, having a glass cock, or spigot, with an 0.25-inch orifice, through which the outflow is very rapid. From this bottle the acid is tapped into a specially designed measuring cylinder, shown in Fig. 2, provided with a stop-cock similar to that of the bottle, which allows the acid to be tapped into the beakers, without dripping, in a neater and cleaner manner than can be done by pouring from a bottle, beaker or cylinder.

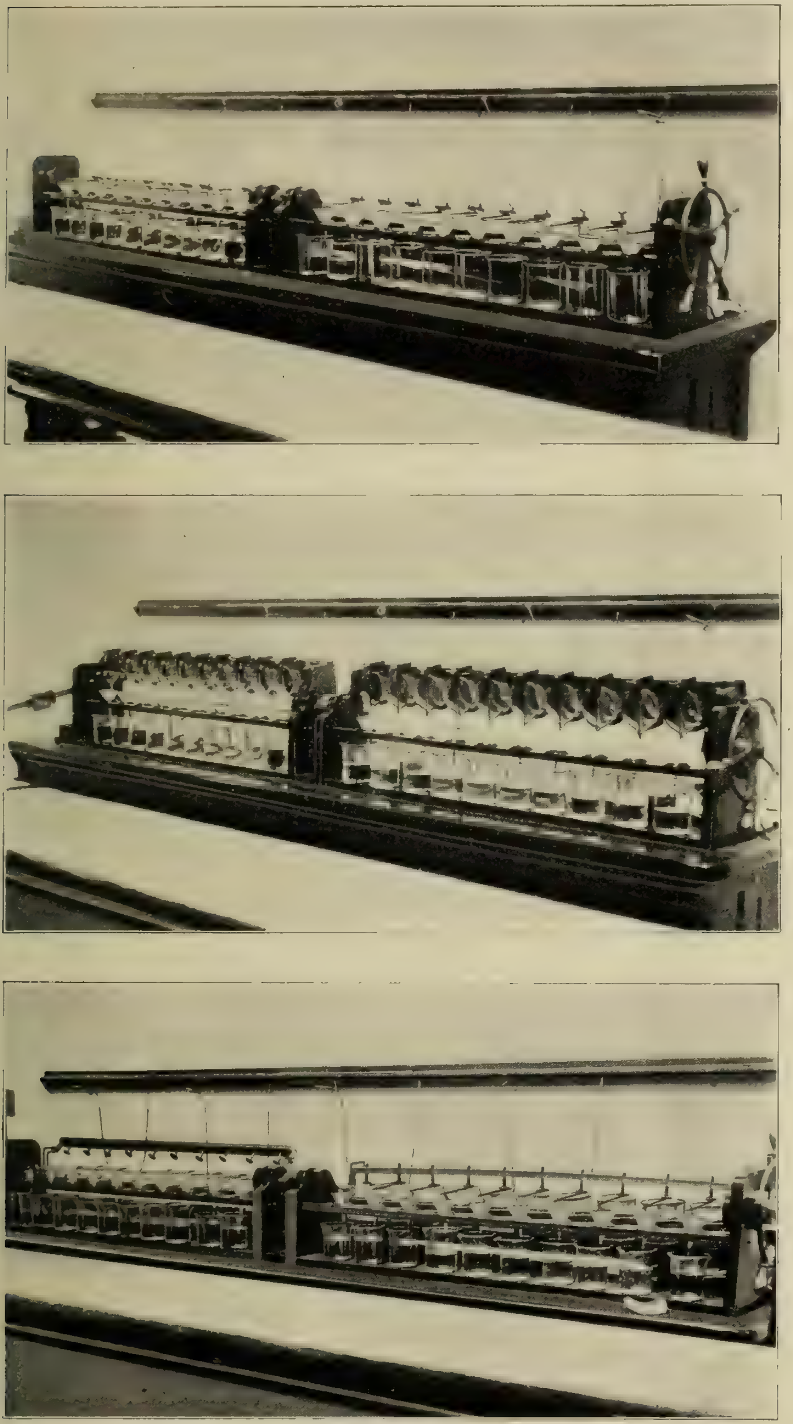
After the copper solutions have become cold and the proper quantity of sodium chloride requisite to precipitate all the silver has been added, it is absolutely necessary to stir the mixture in order that the reaction shall take place throughout, and that, after adequate settling, none of the silver chloride will run through the paper during the subsequent filtration. The stirring of a large number of solutions by hand occupies much time and is very tedious, but omitted, the determination becomes faulty. Figs. 3, 4 and 5 illustrate a stirring machine which obviates all the difficulties incident to the manual operation of stirring, and, by its use ten solutions can be stirred in the same length of time occupied in stirring one by hand, and there is no splashing of solution or breakage of beakers.

Fig. 3 shows the machine ready for use; Fig. 4 shows the position after the stirring-rods have been lifted from the solutions and are ready to be rinsed; and Fig. 5, the position assumed after the rods have been rubbed off, the discs holding them and forming a cover to the beakers, now being ready to be washed. This latter operation, however, is generally superfluous, for very seldom does any of the solution reach the covers.

The characteristics of this stirring-machine are the three-based feet, the convenient driving-gear on the right which admits of the ready application of the mechanical powers; the slack take-up for the belt on the left, and the rubber discs which hold the rods and form covers to the beakers. The action of the machine is made universal by having several holes in each disc so that the position of the rods will conform to any size of beaker. The separate construction of both beaker-stand and stirring-stand permits a ready change of a set of beakers and allows the rods to remain a permanent part of the whole.

Filtering Apparatus.

The construction and operation of Keller's filtering or decanting apparatus are illustrated in Figs. 6, 7 and 8; and by its use twenty filtrations can be performed with perfect ease, and without the least danger of loss by splashing or breakage. The beaker-rack is tilted by means of a hand-wheel on the right, retrograde motion being prevented by a ratchet. The point of rotation of the whole series of beakers lies some distance from their lips, about at the end of the glass rods that guide the stream of liquid to a definite point in the filter, an arrangement which is essential for steady pouring. The lifting of the load is aided by a counterpoise on the left.



FIGS. 6, 7, 8.—THE FILTERING OR DECANTING APPARATUS.

By the use of this apparatus the filtering can be done leisurely with one hand, so steadily that the precipitate remains undisturbed until all the clear liquid is poured off, and the time of the whole operation is greatly shortened.

Figs. 6 and 7 show the machine as originally constructed, with a locking-device for each beaker and rod, consisting of an arm pressed downward by a spring, and free to rotate vertically within an angle sufficient to clamp, as well as to allow it to sweep horizontally over the rod and beaker, the horizontal rotation also being entirely free. Fig. 8 presents a locking device on the same machine by means of which ten beakers and rods are locked simultaneously. It is a matter of taste which arrangement is the more satisfactory. The beaker-rack and the filter-stand are adjustable to several sizes of beakers and funnels, and the machine can be made for any number of beakers. Light wooden trays are provided to carry the beakers, in sets of ten, to or from the apparatus.

Table I, containing the results of a test recorded by one of Keller's assistants, shows the difference in time between the new method of mechanical filtration and the old method by hand. The quantity of solution contained in each beaker varied between 275 and 300 c. c.

Table I.—Comparison of Results of Machine Filtration vs. Hand Filtration.

	Machine Filtration.	Time Consumed. Minutes.
10.21 a. m.	Started.	
10.22 a. m.	Placed rods	1
10.30 a. m.	Finished decanting	8
10.41 a. m.	Washed beakers and rods	11
10.45 a. m.	Washed filters	4
10.55 a. m.	Rubbed out beakers	10
10.59 a. m.	Folded filters and placed in scorifiers	4
	Total	38

	Hand Filtration.	Time Consumed. Minutes.
11.13 a. m.	Started.	
11.14 a. m.	Placed rods	1
11.38 a. m.	Finished decanting	24
11.58 a. m.	Washed beakers and rods	20
12.04 p. m.	Washed filters	6
12.13 p. m.	Rubbed out beakers	9
12.17 p. m.	Folded filters and placed in scorifiers	4
	Total	64

Difference in favor of machine 26 minutes.

Keller decanted ten beakers, each containing from 275 to 300 c. c. of solution, in 4 minutes 30 seconds, the solutions being run through a double S. & S., No. 597, 12.5 cm. filter.

In connection with the work as just described, Keller entirely dispensed with the well-known wash-bottle, on account of the unnecessary physical strain involved in its use, as well as its unsanitary character in many instances. The distilled water contained in carboys, placed about 4.5 feet above the floor level, is siphoned into a system of glass tubing, extending throughout the laboratory wherever the water may be needed. This glass tubing is provided with numerous T connections, to which are attached rubber tubes, having pinch-cocks and glass nozzles at their ends, which permit the direction of a stream of water to any desired point. The water in the overhead carboys is replenished by forcing a new supply from another carboy in which it has been condensed, placed on a carriage on the floor, up through a glass siphon by means of compressed air.

When the filtration is complete and the precipitates have been washed out of the beakers, the latter must be rubbed out with paper in order to be sure that neither silver chloride not gold is retained on the walls of the vessels, perhaps by a little oil often contained in the samples, or by the drying and hardening of small particles of chloride above the surface of the solution. In order to facilitate this cleaning, Keller has designed a machine in which the beakers are rotated.

This machine is on rollers, and runs on rails fastened to the table in front of the filtering apparatus, and may be locked at any desired point. The rubbing is done by means of a "policeman," Fig. 9, which consists of a rod having a cork fastened at one end, over which is clamped a piece of filter paper, held in place by a conically-cut ring. The paper thus fastened is run along the bottom and side of the rotating beaker. For each beaker a fresh piece of paper is quickly clamped to the cork.

New assay furnace tools designed by Dr. Keller for his laboratory will be described and illustrated in our next issue.

Action of Acids on Iron and the Use of the Acid Pickle.

By Charles F. Burgess.

Definite knowledge, both quantitative and qualitative, on the subject of the corrosion of iron by the common acids, is of great usefulness, alike to the scientist and to the technical man. Yet it must be acknowledged that information relative to the action of acids on iron and its alloys is notably deficient, notwithstanding the fact that for a century or more it has been a subject of constant study, and that many investigators have given, and are still giving, their attention to it. To the investigator, who as an incentive needs a hope of practical utility rewarding his research, a study of this subject offers much that is attractive. That results of scientific as well as of practical value may arise from it is shown by the insight which has been given recently into the constitution and properties of iron and its alloys, through a knowledge of the action of acids upon the various constituents of commercial iron and steel.

To one who has never given the subject any especial thought it might appear that to determine the rate of the corrosion of iron by hydrochloric, sulphuric or nitric acid, and to note the effect that each exerts upon the iron, would be a simple matter enough, and he might feel some surprise that the subject has not been thoroughly investigated long before this. Study will show him, first of all, that scattered throughout scientific and technical literature there is an abundance of material coming from many valuable sources, which might possibly be so collated and classified as to place all needed information in a useful form. He would soon find, however, that such literature abounds in the most conflicting statements—that, for instance, one authority will claim for hydrochloric acid a more rapid action on iron than he is willing to accord to sulphuric acid, while another will ascribe a greater activity to sulphuric acid. He will see, too, that while many qualitative results are given, comparatively little quantitative work is recorded, while between the quantitative results that are given by different investigators there is but little agreement.

This condition of affairs may be ascribed to several causes. Differences in the degree of care exercised by various observers can account only in small part for such glaring inconsistencies of statement. It has been commonly assumed that hydrochloric acid is hydrochloric acid the world over, and that wrought iron is the same material wherever it is found. It is to such assumptions that many of these discrepancies are due. The assertion that a 10 per cent solution of hydrochloric acid will dissolve ten grams per hour from a square foot of wrought iron actually means little more than that from wrought iron in general a certain amount of corrosion takes place which can be readily detected. To make such a statement of any real value we should be given the temperature, the number and the amount of impurities in the acid solution and the chemical and physical constitution of the iron which has been thus treated.

When from available literature we exclude such data as this on the ground of its incompleteness there remains comparatively little that is suitable for use. On the other hand, to furnish all the desired data involves an expensive and laborious investigation that few are able or willing to undertake, while even when the results are given, so many variable factors are involved that to draw general conclusions and to deduce general laws from a number of investigations becomes a most complicated matter. This might lead to a question as to whether the importance of the matter would justify such research, but reference need only be made to the various industries in which the corrosion of iron is an economic factor to allay any doubts upon that head.

Prevention of Corrosion.

As it is the most important structural material in use at the present time, iron is necessarily frequently used under conditions giving opportunity for acids and other corrosive liquids to make contact with it. This is only an exaggerated example of the corrosion to which steam boilers, structural iron work, underground piping and the like are subjected. Iron is employed in the chemical industries for piping, storing and conveying acids, iron tank cars being used extensively for the last-named purpose. In the design of such structures, consequently, the corrosive factors must be taken into consideration, this being necessary for the sake of both safety and economy. Knowledge of these factors will direct that iron shall be used for certain acids, but rejected for others. Also, since it is known that some grades of iron are more resistant to the corrosive effects of certain acids than are other grades, it is important to ascertain which are the most durable grades. It also is often possible to introduce certain impurities into the acid, which will still farther enhance the durability of the iron containing-vessel. Thus it will be seen that definite knowledge as to the influence exerted by such impurities is highly desirable.

Acid Pickles.

While the prevention of corrosion is a matter of economic importance in certain lines of work, for other purposes it is equally important that the action of the acid should be facilitated.

The term "pickle," as used in the metal-working industries, is applied to acid solutions into which a metal article is dipped to free its surface of oxide, scale or crust, or to give it a rough or matt appearance. While such dip is used in connection with the treatment of copper, brass, steel, iron and many other metals, it finds its most important application in the iron and steel industry, and it is with reference to this branch of the work that the following will have principally to deal.

Pickling constitutes an essential step in the well-known galvanizing process. Prior to immersing the sheet metal in the molten zinc it must be freed from scale and other oxides, and this is most satisfactorily accomplished by the use of the acid dip. To show something of the magnitude of this one operation alone, it may be estimated that since 60,000 tons of zinc per annum are used in this country for galvanizing, the area of surface which is thus protected is approximately equivalent to 70 square miles, or 40,000 acres. The area which must be pickled in the manufacture of wire and sheet metal probably largely exceeds even this. Statistics given in *The Mineral Industry* for 1903 show that about 1,500,000 tons of iron wire were produced during that year. Such an amount of iron in the form of wire possesses over 100 square miles of area, and since the process of pickling must be repeated several times during the operation of drawing, the pickled area is unquestionably larger than this amount. From figures taken from the United States Census Report for 1900 about 20,000,000 pounds of tin were used that year for the production of tinned sheets, and this again represents a second area of about 140 square miles which must be pickled, not once, but twice.

In the process of wire drawing, rods about two-tenths of an inch in diameter are drawn through a series of round, tapering holes in a die, each hole being smaller in diameter than the preceding one. Prior to passing the rod received from the rolling mill through these dies, it must be pickled, to remove any scale which would otherwise rapidly cut the die. After passing through a number of dies, varying from two to six times, according to the quality of the iron being drawn, and the amount of reduction required, the metal becomes hard and brittle and annealing is necessary. This is accomplished by raising the iron to a red heat, and allowing the temperature to fall slowly. In this process another scale forms on the iron, and a second pickling operation becomes necessary.

The manufacture of iron stampings and cold-rolled ware likewise involves repeated annealing and pickling of the product. In the production of enameled ware, enormous quantities of which are manufactured annually in the United States, the acid corrosion constitutes an important step in the operation, since the scale must be removed before the article is stamped into shape and again after it has been annealed, that it may present a perfectly clean surface for reception of the enamel.

Pickling is also used by the electroplater in preparing his work to receive its nickel or copper coating, and it is likewise necessary for structural iron prior to painting, lacquering or varnishing of the metal surface. Many other applications of the process could be cited, but these given are sufficient to show that the action of acids on iron surfaces ranks among the most important problems with which the modern engineer has to deal.

In spite of the fact that this operation is by no means a modern one, and that those who use it form a numerous class of workers, no uniformity of practice seems to have been evolved. Each operator has a method of treatment peculiarly his own, the outgrowth of his personal experience and rule-of-thumb trials rather than of any general principles that have gained recognition. One operator will use hydrochloric acid, another will prefer sulphuric, and a third will cling to certain mixtures. Not infrequently do metal-plating establishments receive visits from an "expert," whose garb testifies to the material prosperity of the wearer, and who offers for a certain bonus to divulge a secret formula by which the pickling department may double its output through the elimination of certain difficulties which thitherto had proved unsurmountable. Strange and wonderful are many of the recipes thus procured. One such which has been brought to the attention of the writer produced a pickle, that when first set up, undeniably proved to be most active, but for what purpose the majority of its dozen or more ingredients had been added was difficult to explain. Especially did the requirement of 2 per cent of whiskey add to the mystery!

An ideal pickle for removing scale and rust would be one which would remove the coating without attacking the underlying metal. To this end the acids must be so chosen as to exert a maximum solvent action on the oxide and a minimum solvent action on the iron.

According to Comey (Dictionary of Chemical Solubilities) iron is readily soluble in most acids, particularly so in sulphuric and hydrochloric; nitric acid also exerts great solvent action, unless in such degree of concentration as to render the iron passive. The rate of corrosion of acids varies with the concentration, increasing as the concentration increases up to a certain limit, after which the rate of corrosion decreases. This variation is sometimes very irregular. The temperature also has a marked effect, the activity of the acid increasing with the higher temperatures. This furnishes a reason for the common practice of heating pickling solutions. The increase of acid efficiency produced by heating is not uniform for all grades of iron, it being claimed that for charcoal pig and case-hardened cast iron the increased rapidity of attack by boiling acids is less than for certain other grades of iron. The addition of soluble salts of copper, platinum and certain other metals greatly increases the action of an acid, while the addition of arsenic, it is claimed, almost entirely prevents the action of sulphuric and hydrochloric acid on iron.

According to the same authority, the principal iron oxides to be considered are ferrous oxide, FeO; ferric oxide, Fe_2O_3, and ferroferric oxide, to which is assigned the formula Fe_3O_4. Ferrous oxide is easily soluble in hydrochloric and nitric acids, but nearly insoluble in sulphuric acid, even when heated. This oxide is, however, of minor import. Ferric oxide, an example of which is the ordinary brown rust, is attacked by acids with difficulty, the more so the higher it has been heated. Aqueous hydrochloric acid solutions when heated are given by Fresenius as the best solvents for this oxide, while Mitscherlich claims it is most easily soluble in eight parts of sulphuric acid to three parts of water. By far the most important oxide with which the pickler has to deal is the black, or magnetic oxide, to which the definite formula Fe_3O_4 is commonly ascribed, but which is sometimes considered as being a mixture of FeO and Fe_2O_3. The only solvent which Comey gives for this oxide is hydrochloric acid.

It will be seen that the oxides of iron are much more resistant to the action of acids than is the iron itself, so that the principal desideratum of an ideal pickle cannot be attained through their use. To remove the oxide we must depend, to a large extent, upon the action of the acid on the metal which underlies the oxide, thereby loosening it so that it falls off or is left readily removable by physical means.

The black oxide, which is always formed when iron is heated in the presence of air or other oxidizing agents, presents various physical characteristics determined by the degree of heating, the length of time and the properties of the underlying metal. It may form, under some circumstances, a thin, film-like and strongly-adhering coating, and in other cases it may create thick scales which can be detached by hammering, rolling or other mechanical means. While this coating usually presents a continuous appearance to the naked eye, when carefully examined under the microscope it will be found to be perforated by cracks or holes through which the acid can enter. The iron in contact with such scale is more readily attacked by the acid than when the surface is entirely clean, since the black oxide is electronegative to the iron and sets up active electrochemical couples.

Practice in Pickling.

It must not be inferred from what has been stated previously that iron and steel manufacturers have not given extensive and systematic study to the matter of utilizing acids to their highest efficiency. In this country the leading producers of wire, sheet metal and structural iron have worked out economies in this detail of manufacture as well as in other steps of their processes, and have thereby led the world in cheapness of production. There has been continual and careful comparison of acid consumption and methods, and through modifications of the operation, necessitated by changes in the treatment of the steel previous to its pickling, the quantity of acid required has been reduced materially, and this, in turn, has led to a saving in the metal itself, since the consumption of an acid means the consumption of a chemically equivalent amount of iron. This detail in manufacture has not, however, been the subject of wide discussion before engineering societies, nor has it been given the same degree of publicity accorded other features of the manufacturing processes.

Choosing at random information to be found in various technical books and publications, we can readily see the differences in practice which prevail among the various users of the pickling operation.

In describing the Cowper-Cowles process of electrogalvanizing (Electrochemical Industry, March, 1903, p. 263), it is stated that the removal of mill scale from forgings and plates has always been a matter attended with much difficulty. The pickle recommended is one part sulphuric or hydrochloric acid to ten parts of water, the period of immersion ranging from one-half hour to twenty hours. It is asserted that the British admirality specifies that all steam pipes, boiler and collector tubes and all plates for boilers shall be pickled in a liquid containing one part hydrochloric acid to nineteen parts of water until all flack oxide is removed.

According to Marks (*The Manufacture of Iron and Steel Tubes*) the process to be used involves the employment of a weak acid solution, consisting of one part hydrochloric acid to thirty-nine parts of water. Brannt (*Metal Workers' Handybook*) asserts that for cast iron and wrought iron articles a mixture of one part sulphuric or hydrochloric acid and ten parts of water, to which some tar has been added, should be used.

Randau in his work on "Enamels and Enameling" says that the usual strength of acid pickle is one part of commercial sulphuric acid to twenty to twenty-two parts of water, the time of immersion depending upon the strength of the pickle. If the exposure is not to be more than from ten to twelve hours a stronger pickle is used, and, as a rule, the ware should not be allowed to remain in the pickle for more than twenty-four hours. When the ware is left too long in a pickling liquid that has been in use for some time, and has, consequently, taken a relatively large amount of iron into solution, the liquor forms, in the presence of iron, certain insoluble basic salts which adhere firmly to the metal in the form of a soft powder, thus defeating the very object of the pickle. Such "over-pickled" goods must then be scoured with sharp sand and [illegible]. The solution should be operated at a temperature of from 30° to 40° C.

TABLE SHOWING VARIETY OF PRACTICE IN THE OPERATION OF PICKLING SOLUTIONS FOR IRON.

MATERIAL TREATED.	PURPOSE.	COMPOSITION OF PICKLE.	TIME.	REFERENCE.
Drawn iron and steel tubes.	Removal of mill scale after annealing.	1 part HCl–39 parts water.	....	Marks.—"The Manufacture of Steel and Steel Tubes."
Cast or wrought iron articles.	Removal of scale.	1 part HCl or H_2SO_4 to 10 of water, to which tar is also added.	....	Brannt.—"Metal Worker's Handibook."
Iron ware.	Preparatory to enamelling.	1 part of H_2SO_4 and 20 to 22 parts of water. 30° to 40° C.	Under 24 hours.	Randau.—"Enamels and Enamelling."
Structural iron.	Removal of scale and rust.	25% to 28% H_2SO_4 hot, or 9% to 10% H_2SO_4 cold.	6 to 12 minutes. 5 hours.	Sabin.—"Technology of Paint and Varnish."
Sheet iron.	For galvanizing.	HCl preferable to H_2SO_4. 1 part acid to 1 part water.	....	Harbord.—"Metallurgy of Steel."
Sheet iron.	For tinning (preliminary operation).	Commercial H_2SO_4, 1.6 to 1.74 s.g. diluted by equal amount of water.	....	Harbord.—"Metallurgy of Steel."
	(After rolling and annealing.)	Weak solution of H_2SO_4 free from arsenic.	....	Harbord.—"Metallurgy of Steel."
Structural iron.	Removal of mill scale for electro galvanizing.	1 part H_2SO_4 to 10 water, or 1 part HCl to 10 water.	5 to 20 hours.	Cowper-Coles.—"Electrochemical Industry" March, 1903, p. 263.
Boiler plates, tubes and steam pipes.	Removal of scale for inspection.	1 part HCl to 19 water.		British Admiralty specification. Ibid.
Rough castings and wrought iron.	For electroplating.	1 part H_2SO_4 to 4 water.	....	Hawkins.—"The Polishing and Plating of Metals."
Smooth castings and stampings.	For electroplating.	1 part H_2SO_4 to 10 water.	....	Hawkins.—"The Polishing and Plating of Metals."
Castings with adhering sand.	For electroplating.	1 part HF to 15 water.	....	Hawkins.—"The Polishing and Plating of Metals."
Iron and steel.	For removing rust.	Pure concentrated HCl.	4 to 20 minutes.	Hawkins.—"The Polishing and Plating of Metals."
Sheet iron.	For galvanizing.	1% H_2SO_4 solution.	6 to 8 hours.	Molesworth.—"Pocket Book for Engineers."
Wrought iron.	For electro-galvanizing.	HCl preferable to H_2SO_4 except for cost. H_2SO_4 of 1.85 s.g. diluted by 5 parts water. HCl of 35° Twaddell diluted by 1 water.	.... 20 minutes. 12 to 15 minutes.	Davies.—"Galvanized Iron."
Wrought iron.	For electroplating.	2% HNO_3 or 8% H_2SO_4 or 2% HCl.	....	Foster.—"Electrical Engineers Pocket Book."
Cast iron.	For electroplating.	3% HNO_3 or 12% H_2SO_4 or 3% HCl.		

Sabin (*Technology of Paint and Varnish*) discusses the practice followed both in the United States and abroad relative to the removal of mill scale from structural iron prior to painting. The pieces are immersed in hot dilute sulphuric acid of a strength of from 25 to 28 per cent, some factories employing a 20 per cent solution. To completely remove rust and scale will take from six to twelve minutes. Combinations of rust and scale are much more readily removed than the adherent blue or irridescent rolled scale alone. Where only the hydrated oxide is to be removed this is readily dissolved in a 10 to 12 per cent acid. He states that in Germany it is customary to use a 9 or 10 per cent acid, cold, leaving the metal in solution for five hours, but that this requires a larger plant and offers no especial advantages. In subsequently washing the article by dipping it into water the iron becomes coated with a gummy or collodial substance, the exact nature of which is not known but which is very difficult to remove. To prevent the adherence of this substance it is recommended that the washing should be effected by using a jet of water, which removes the material as rapidly as it forms. It has been suggested that this trouble may be due to the presence of arsenic in the solution, and for this reason many picklers in purchasing acid specify that it shall be free from arsenic.

He also states that it is often difficult, and sometimes quite impracticable, to pickle steel high in carbon or cast iron containing graphitic carbon, on account of the deposition of a film of carbon, like stove blacking, on the surface. Hydrochloric acid has been used instead of sulphuric, but it is not well suited to the purpose, being more expensive to procure and more difficult to remove. It also forms a gummy substance on the iron, which is worse than that produced by the use of sulphuric acid, and in the subsequent alkaline treatment it must be neutralized with caustic soda instead of with lime. After the iron has been freed from sulphuric acid by washing it with a jet of water, it is put into a bath of lime water or milk of lime made boiling hot—a matter of great importance—and left there long enough for the metal to reach the temperature of the surrounding liquid. It is then removed to an oven and dried, after which the lime is brushed off. If desired the lime may be removed before putting the iron into the oven. In this case it will be found that the surface, which is perfectly clean and bright, rusts easily and quickly, whereas if the lime is removed by drying and brushing the surface it is less likely to rust. Even then, however, it rusts easily and should be painted immediately.

Sabin says that some of the largest work done recently has been treated in the following manner: The steel from the mill was put into a hot 10 per cent caustic soda solution until all the grease and oil came off, together with dirt and a certain amount of scale, making a bulky sludge. Next, the steel was washed in boiling water and put into a hot solution of sulphuric acid of 10 per cent strength, and left until the metal surface was completely exposed, after which it was dipped into boiling water, then into a hot 10 per cent solution of carbonated soda, well washed and finally dried in the oven.

Harbord and Hall (*Metallurgy of Steel*) say that where readily obtainable hydrochloric acid is preferable to sulphuric in the proportion of one part of acid to one part of water, since it does not require warming, and the sheets previous to galvanizing do not require to be soaked in water but may be passed at once to the molten zinc bath after only a moment's drainage. The cost of pickling by either acid is stated to be about the same, the difference in the quantity required being equalized by the difference in price, but hydrochloric acid has the advantage of being quicker in its action and generating fumes which are less objectionable to the workmen.

It appears from this same work that the practice followed in the preparation of sheet iron for tinning is different from that used for galvanizing. The black iron is first subjected to what is known as the black pickle, in which sulphuric acid made from pyrites is used, the acid having a specific gravity of from 1.6 to 1.74 and being diluted with an equal amount of water. The sheets coming from this bath are washed in water and annealed and then rolled to give a smooth surface. They are then pickled again in a weak solution of sulphuric acid, made from burning sulphur to render it free from arsenic, which would leave black streaks on the plates.

In the electroplating industry, even a larger number of problems arise in connection with the pickling process than in the industries heretofore cited, this being due to the greater variety in the kinds of iron which must be treated. A distinction is made ordinarily between dips and pickles, the first named usually consisting of undiluted acids or mixtures of acids, the purpose being to dissolve the coating substances while leaving the underlying metal smooth and unattacked. Pickles, on the other hand, are diluted acid solutions designed to corrode the metal. Hawkins, in his work entitled "The Polishing and Plating of Metals," which is a thoroughly practical account of the electroplating processes used in this country, recommends the following solutions:

Dip for Iron.—For removing rust from iron or steel goods of any kind a dip of pure muriatic acid is used. Where the goods are not very rusty four or five minutes' immersion will be sufficient; if badly corroded, from ten to twelve minutes will be required; but this dip will remove the rust, no matter how thick, without hurting the metal if the work is removed and thoroughly rinsed, first, in cold then in hot water, and dried in sawdust as soon as the rust disappears.

Pickle for Iron.—For a quick-working pickle on rough iron castings use sulphuric acid in the proportion of one part acid to four parts water; for a slower acting pickle intended for smoother castings and stampings use one part sulphuric acid to ten parts of water. The amount of water may be increased still more in pickling very smooth pieces where little sand or scale is present. The pickle in every-day use for castings and wrought iron goods consists of one part acid to four parts of water. Generally it is used cold, but where large quantities of work are to be pickled and the saving of time is an object, the operation will be hastened if the pickle is kept at about 150° F.

Where castings contain silica and adhering sand, a hydrofluoric acid pickle is used to considerable advantage. For a strong and quick-acting solution, one part acid to fifteen parts of water is used, and for a slower-acting solution the quantity of water may be doubled.

(*To be Concluded.*)

A Furnace for Metallurgical Research.

By Woolsey McA. Johnson.

A furnace designed for use in a metallurgical laboratory should be capable of producing on a small scale as many as possible of the conditions of large scale work as regards temperature and character of flame. The operator should be able to control these conditions with ease and certainty. Moreover, the furnace should be durable. Any parts not durable, should be easily removable. Besides, the furnace must not take up too much valuable room in the laboratory.

An experimental furnace of mine located at Hartford, Conn., was described in the *Engineering and Mining Journal* of July 18, 1903. Shortly after that date I designed for the metallurgical research laboratory of the Lanyon Zinc Co., which department I organized, a furnace which, like the first furnace, was capable of being used for a variety of reactions. In actual practice this furnace was found to be durable, and easy to handle. The total cost was $175. The control of temperature can be seen by referring to the Electrochemical Industry, May, 1904, p. 185. Here it will be seen that we could hold temperature constant, or raise it about a degree a minute for a long time.

The fuel used was Kansas natural gas, consisting largely of methane. Consequently, unless special precautions are used, the gas has a short cutting flame. The ignition point is high, but combustion once started proceeds usually with great violence. Accordingly to conserve heat and carry the flame for some distance, the furnace was designed with 13½-inch walls, and with air spaces between these walls wherever possible. These air spaces were filled with mineral wool, whose melting point, acording to statements of manufacturers, was higher than temperatures expected, but in reality this wool was made from a very fusible slag. Probably the wool in these spaces was melted during the first day the furnace was really heated up.

But at all events, the thick walls with air spaces brought about the desired effect,—the flame when once the furnace was in operation could be carried a long distance without cooling the luminous particles of carbon. For, usually heating was effected by radiation of a luminous flame (the same principle as used in an open-hearth furnace) and not by "direct contact."

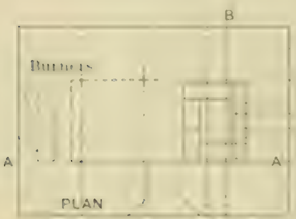
The air used in all cases for burning the gas was preheated, by passing through three retorts in series in the highest chamber, and finally through a flue 4½ inches by 22½ inches to the gas supply. These retorts were put in green, that is, dried, but not burnt in the kiln. The butts were knocked out so that they were practically fireclay pipes. They could easily be replaced by knocking out loose brick work at both ends of the furnace. Small "jack arches" in the wall above the retort permitted this. When slowly heated up and always kept warm, (say above 300° C.), these retorts will last a very long time. But any sudden cooling down develops innumerable internal strains and cracks from these strains. Thus the whole body of the fireclay becomes rotten.

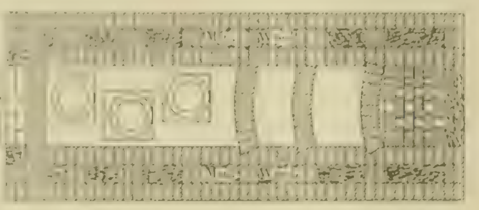
In about a year we put in two new sets of retorts in the preheater. The cost of a new retort being not over 50 cents, the repairs on preheater amounted to an insignificant sum.

At first the draft of the furnace was sufficient to give the furnace all the air necessary. But in time, the sum of all tiny leaks developed the reducted draft so that we put a 12-inch electric desk fan on a shelf outside the furnace and directed it at the opening of the preheater. By varying the voltage of the pan it was possible to regulate wind on furnace to a nicety. Pyrometer tests showed that it was possible to deliver air to the combustion chamber at a temperature of 1100° C., if desired. The gas amounting to about one-tenth the volume of air was not preheated, for it would have helped but little.

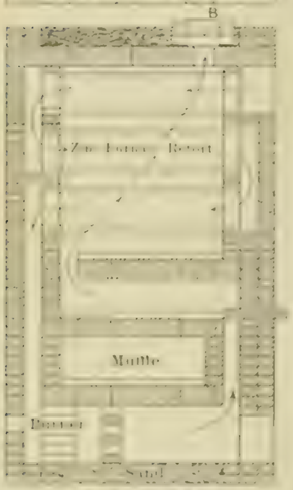
The gas came into the building at a pressure of 1½ pounds, and was fairly constant in pressure.

One large gate valve with a 10-inch sheet zinc dial, graduated in fractions of a circle on handle allowed accurate control of flow of gas. By use of two valves, one on each burner, it was possible to cut gas off without disturbing the position of regulating valve.





CROSS-SECTION END ELEVATION AT B B



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The burners were of the type called "jet-burners." They consisted of a T, fitted with two 1½-inch capped nipples. This all formed a large T, which was drilled with numerous 3-32-inch holes. These burners were set in the furnace with plastic fireclay, indicated in drawings. They could be easily replaced, but as set in a recess of brick and kept cooled by the flow of gas, they lasted for a year, and should last a much longer time.

The heated air was then mixed with these jets of gas. The air being hot, and the combustion chamber extremely hot, active combustion ensued and was helped by the mixing which the gases received as they passsed through checker work. The flame was then carried to the working chambers and used for the desired purpose.

The rate of combustion of coal and gas, and the character of flames are important and neglected parts of metallurgy. The checker work was first composed of firebrick made from good St. Louis clay. These were soon melted to a paste and were replaced by silica brick. The silica brick lasted until some sudden change of temperature "rotted" them.

The lower working chamber was about 30 inches by 22½ inches by 15 inches high to bottom of arch. The floor of the furnace was silica sand. This was an extremely wise provision, for it permitted any fusible material, such as charge spilled from the crucible to be pulled out and fresh dry sand put in.

From the lower working chamber the flame passed up a flue 22½ inches by 4 inches into the upper working chamber. By a proper manipulation of air and gas it was possible to produce any heat from 300° C. in this working chamber. Here we did all our work in reduction temperatures, volatility of silver in roasting, decomposition, temperature, etc. The lower working temperature was used almost exclusively for fusions.

In this, and in two small electric furnaces I repeated with much extension nearly all Percy's experiments on the metallurgy of zinc as well as many other experiments to test in the "try-it-and-see" way, many ideas that occurred to me while metallurgist of the No. 2 works of the Lanyon Zinc Co.—ideas that seemed likely to help us in our large scale work. These small experiments, if promising, would be carried out on 100 or 200 pounds of ore at the No. 2 works. I have never had a more perfect union between theory and practice. This happy result can be attributed in large measure to the fact that we had in this experimental furnace a first-class metallurgical instrument which allowed the experimenter to control conditions accurately.

Between the upper and lower chamber was a muffle chamber of about the same size. This was used for drying firebrick, crucibles, etc., etc. The door to every chamber consisted of a large block of fireclay. In several places were clamps to hold a Le Chatelier pyrometer tube, and circuits leading to the galvanometer placed on a brick pier near furnace.

Above the upper working chamber were set the three zinc retorts which served as preheater, as described before. The top of the furnace consisted of large firebrick plates covered with mineral wool. The stack rested on heavy iron bars exposed to air for cooling purposes. It was about 12 feet high above the furnace, for first half firebrick, then red brick. Inside dimensions, 8 inches by 8 inches. At each corner of stack were 2-inch angle-bars, tied every 4 feet in the usual manner of tieing chimneys. The furnace rested on a concrete foundation 20 inches wide, following the line of walls. Firebrick plates were used as dampers between upper and lower working chambers.

The "ironing" of the furnace was substantial and satisfactory. At each corner a heavy 4-inch angle-bar was set in brick work. Horizontal courses of 5 inch by ¾-inch flat steel bars were set in flush with face in brick work, one at the top of the furnace and one to take thrust of each of the three arches. There were thus four courses of these steel bars extending clear around the furnace. Four 40-pound old rails were set as buckstaves on the longer sides. The buckstaves were bedded in concrete at the bottom. Four sets of 1-inch round bars, each with eye, threaded end, nut and washers tied the furnaces horizontally. These are set just like tieing of a square chimney. They included the buckstaves. Upper expansion was neglected. The furnace developed few cracks after a year of many temperature changes.

The chief disadvantages of this furnace were inaccessibility of the lower chamber, for the operator was obliged to stoop to put in the crucibles, and the fact that it was hard to clean the upper chamber. But lack of space in the laboratory caused us to build vertically rather than extend horizontally.

In a later issue of this journal, a description of a serviceable electric crucible furnace will be given.

Huntington-Heberlein Process.—It is reported that the American Smelting & Refining Co. has decided to adopt this process, and that it has been put in operation successfully by the Sullivan Group Mining Co., Marysville, B. C. The process divides the roasting of PbS into two steps. First, in a reverberatory furnace a mixture of PbS with CaO is heated in a strongly oxidizing atmosphere to 700° C. Second, the product is cooled in a strongly oxidizing atmosphere to 500° C., and subjected to an air blast in a converter. This reaction develops sufficient heat to maintain the temperature; SO_2 escapes, and the product is PbO, $CaSO_4$ and gangue

Paul L. T. Héroult.

Of European electrochemical and electrometallurgical engineers none is better known all over the world than Dr. Paul Héroult, the famous pioneer of the electrometallurgy of aluminium and steel. While France is proud to claim him as a Frenchman, the results of his work belong to the world.

Paul Louis Toussaint Héroult was born at Thury-Harcourt (Calvados, Normandy), on April 10, 1863. Since his grandfather lived in England he spent his youth's happy days partly in France and partly in England. He was educated at the Lyceum of Caën, Normandy, and later at St. Barbe, in Paris, where he graduated in 1882, to enter the Paris School of Mines (Ecole des Mines). From 1883 to the beginning of 1884 he served in the French army and then returned to the School of Mines, but on account of the death of his father he was soon forced to quit his studies in order to conduct the business of his father's tanning works.

However, Héroult continued there his metallurgical researches. He was specially interested in the electrochemical production of metals. He bought a small Gramme dynamo of 1.5 hp., which he replaced in 1886 by another small machine giving 800 amps. at 15 volts. This machine was specially built for his aluminium process, for which he had applied for patents in the same year. Héroult's revolutionary work and success in the aluminium industry is so well known to our readers that nothing further need be said here. In 1887 he became the technical manager of the Neuhausen aluminium works of the Swiss Metallurgical Co., where his process is in use up to the present. He then started an aluminium plant in France, at Froges, Isère, for the Societé Electrometallurgique Française. About the same time he made two trips to this country, where, meanwhile, independently of, and simultaneously with him, Charles M. Hall had worked out his aluminium process, and had started the works of the Pittsburgh Reduction Co.

After his return to France, Héroult assumed the management of the Froges works, but after the erection of a larger water-power plant in La Praz, Savoy, he became the general manager of these new works at La Praz, where he still resides. Dr. Héoult married Mademoiselle Chateau, and has five children.

To show what a broad man Héroult is, it may be mentioned that in the erection of the La Praz works he proved his ingenuity as a hydraulic and mechanical engineer, since he modified the original designs of the water pipe line over the Arc River, so as to diminish the first cost very materially. The designers wanted to erect a special bridge for the support of the pipe line, while Héroult found by mathematical calculations that it would be possible to construct the pipe line in form of an arc without any further support than the two pillars at both sides of the river. In 1892 the power developed at La Praz was 3000 hp., which was increased to 13,000 hp. in 1895.

Héroult's electric aluminium furnace, which by the addition of a tap hole was rendered a continuous apparatus, became in his hands the instrument for numerous further researches. In it he made the first commercial calcium carbide produced in France. In it he undertook, for the Societé Electrometallurgique de Froges, the manufacture of artificial corundum, ferrochrome and ferrosilicon, and thus he was led to his researches on the use of the electric furnace in the iron and steel industry, which is now in the foreground of interest. Héroult's work in this field has been so often described in these columns that it must suffice here to give references to these descriptions (our Vol. I., pp. 63, 287, 449, 467; Vol. II., pp. 408, 481; Vol. III., pp. 45, 155), and to state the fundamental principle, which is the production of an artificial slag above the fused bath, the carbon electrodes being suspended into the slag without reaching into the bath itself. In this refining furnace Héroult states that at a cost not exceeding 50 cents a ton, he can make out of any steel delivered in the molten state (*i. e.*, as tapped out of a Siemens furnace or a Bessemer converter, a metal of any desired composition containing less than 0.01 sulphur and 0.01 phosphorus. By operating this electric furnace in connection (not in competition) with existing open-hearth or Bessemer converter plants, Dr. Héroult predicts a revolution in the quality of the steel used for many industrial purposes.

The main point is that he has made steel on a commercial scale in his electric furnace, and has sold it in competition with crucible steel for nearly three years. His company's exhibit at the St. Louis World's Fair received the grand prize.

As another example of Dr. Héroult's versatility it may be mentioned that he is now also engaged in researches on a new system of navigation. The main idea is that at a very high speed the water will behave like a solid, and that it will require less power to skip, so to say, on the surface of the water than to plough through it. While this idea is already used to some extent in automobile boats, Dr. Héroult has devised special apparatus which very materially reduces the friction of the hull on the water.

But after all it is the man as a man that makes Dr. Héroult most attractive. His massive body gives an impression of immense latent energy, while in his direct and cordial manner he represents a most happy mixture of the best traits of the different nationalities among which he has lived. He speaks English and German as fluently as French and without any accent. In his courtesy and vivacity he is a typical Frenchman, while in his broad and well founded common sense in dealing with engineering problems he is like an American captain of industry.

There is one more point to be emphasized about Dr. Héroult as an engineer—that is his firm belief in the importance of theory. He once told the writer that wherever he was successful as an engineer, his success was based on previous theoretical analysis and calculation.

For ourselves we are glad to use this opportunity of acknowledging Dr. Héroult's friendship toward this journal from the very start. To our September issue, 1903, he contributed an article on his electric steel process—the only article he has ever written for an engineering journal; and in one of his characteristic letters he confessed that this was the first time that he had ever made a penny with his pen.

What Dr. Héroult has done has been cheerfully recognized, irrespective of national sentiment. When the Technische Hochschule of Aachen, Germany, opened in 1902 Prof. W. Borchers' new Institute for Metallurgy, Héroult received the honorary degree of doctor ingenieur. Last year France showed that republics are not always ungrateful to their own children, when Dr. Héroult was awarded the grand Lavoisier medal, which is granted every seven years to that French inventor whose work has exercised the greatest influence on the progress of French industries in the field of chemical arts.

In the metallurgy of aluminium, Dr. Héroult has made history. We have every reason to hope that he will be equally successful in the electrometallurgy of steel, and we may expect the earliest large developments in this country. While, for obvious reasons, it is impossible to state much that is going on in plants owned by corporations, we may say that the Canadian government has appropriated the necessary funds for a large scale test at Sault Ste Marie, while the Lake Superior Power Co. is to provide a building and free power for a limited period. These tests will be under the supervision of Dr. Eugene Haanel, Superintendent of Mines, Ottawa, Ont. Since these tests have to do not only with steel refining, but with the reduction of iron from ores in the electric furnace, their outcome is of very great interest, especially as Dr. Héroult will introduce several entirely novel schemes of his own, in order to utilize the electrical energy to the utmost possible limit.

However, industrial developments of apparently greater importance are soon to be expected in this country with respect to steel refining. In this field Dr. Héroult has with him the best wishes of all American electrochemists and electrometallurgists, besides the practical help of the Halcomb Steel Co., of Syracuse, N. Y., who have put up a plant for the production of 80 tons per day. This plant is to be started in about six weeks. Outside of the two old electric steel plants at La Praz and in Sweden, another large one, using the Héroult furnace, is being completed at Remscheid, in Germany, by the Electrostahl Co., formerly Richard Lindenberg Soehne, while the same process has now also been adopted by the Sault du Tarn Co., in France.

Metallurgical Calculations.—VII.

By J. W. Richards, Ph. D.

Professor of Metallurgy in Lehigh University.

THERMOPHYSICS OF CHEMICAL COMPOUNDS.

The metallic oxides are the most important compounds occurring in metallurgy, together with the oxides of hydrogen, carbon and of sulphur, arsenic and antimony, which are formed during combustion and in roasting operations. We do not include as oxides those combinations of two or more oxides in chemical proportions, such as of silica with metallic oxides, which form oxygen salts or ternary oxygen compounds. They will be separately discussed. In the following lists Sm, as before, means the mean specific heat per kilogram, in large Calories (or per pound in lb. Cal.) in the range of temperature given; S means actual specific heat at the temperature given, and Q is the quantity of heat absorbed in fusion or volatilization or in passing through any designated range of temperature. Temperatures are given only in centigrade degrees, excepting that *t* signifies Fahrenheit temperature; volumes of gases are understood as being at 0° C. and 760 m. m. pressure, unless distinctly stated otherwise.

OXIDES

Hydrogen Oxide, H^2O:

Ice Sm (—78° — 0°) = 0.463 (Regnault)
(—30° — 0°) = 0.505 (Person)
Water Sm (0° — 100°) = 1 + 0.00015t (Pfaundler)
S (at 15°) = 1.0045 (Pfaundler)
S (at 0°) = 1.0000 (assumed)

In all metallurgical calculations it will be sufficiently exact to assume the specific heat of water at ordinary temperatures as unity. In heating to the boiling point, from zero, 101.5 Calories.

[illegible]

Sm (0° to t°)
1 m³ up to 2000° = 0.34 + 0.00015t (Mallard and Le Chatelier)
1 m³ 2000° — 4000° = 0.48 + 0.00017t (Vielle)
1 ft^3 up to 2000° = 0.021 + 0.000009t — lb. Cal.
1 ft^3 2000° — 4000° = 0.030 + 0.000011t — lb. Cal.
1 kilo. up to 2000° = 0.42 + 0.000185t — Cal.
1 kilo. 2000° — 4000° = 0.59 + 0.00021t Cal.
1 lb. up to 2000° = 0.42 + 0.000185t — lb. Cal.
1 lb. 2000° — 4000° = 0.59 + 0.00021t — lb. Cal.
1 ft^3 up to 3600° *F* = 0.038 + 0.000009*t* B. T. U.
1 ft^3 3600° — 7200° *F* = 0.054 + 0.000011*t* B. T. U.
1 lb. up to 3600° *F* = 0.76 + 0.000185*t* B. T. U.
1 lb. 3600° — 7200° *F* = 1.06 + 0.00021*t* B. T. U.

Above figures are for water vapor as it exists in almost all metallurgical problems, as true gas far removed from its maximum tension at any given temperature. When the water vapor exists as saturated steam under its maximum tension, as in a steam boiler, it is recommended to consult tables giving the total heat in steam at any temperature condensing to liquid water at 0°, or to use Regnault's formula for the same:

$$Q = 606.5 + 0.305t$$

which expresses the amount of heat required to convert 1 kilogram of water, liquid at 0°, into steam at its maximum pressure at the temperature *t*. A little reflection will show that the above formula amounts to taking the specific heat of a kilogram of saturated steam, under a constantly *increasing pressure*, as Q = 0.305t
or under increasing pressure Sm = 0.305
or per cubic meter (0.81 kilos.) Sm = 0.247

These figures apply to boiler practice only. In cases where water is evaporated at or below 100° C., and converted afterwards into steam at a high temperature, I recommend first calculating the heat required to convert the water into vapor at 0° (606.5 Cal.), and then treating the vapor afterwards as true gas, by the formulæ of Mallard and Le Chatelier. This puts water vapor at once, from the beginning, on the same footing as all the other gases, and greatly simplifies all subsequent calculations, without introducing any unnecessary errors.

Latent heat of fusion = 80 Cal. (Bunsen)
Latent heat of vaporization = 606.5 Cal. at 0° (Regnault)
= 600.0 Cal. at 10°
= 537.0 Cal. at 100° (to vapor at 760 m. m.)
Total heat in saturated steam = 606.5 + 0.305t (Regnault)

Most metallurgical problems which have to do with converting moisture in fuel, ores, etc., into vapor, are concerned with the evaporation of the water far below 100° by means of dry gases, into which the water vapor enters at a partial pressure, which is only a small fraction of atmospheric pressure. The problem is, in these cases, to find how much heat is necessary to convert the water into vapor at the low temperature corresponding to such low pressure. By finding the temperature corresponding to said partial pressure, as a maximum tension, applying Regnault's formula, we get the heat absorbed in producing the vapor at that temperature. We can then add to this the heat required to raise the vapor up to the higher temperature at constant pressure, using Mallard and Le Chatelier's formulæ.

Example: Wet peat in a kiln is dried by hot air, the issuing air being at 50° C. and the tension of the moisture in it 25 millimeters. How much heat is absorbed in latent heat, and how much as sensible heat per 1 kilogram of water evaporated?

Solution: The tension, 25 millimeters, is the maximum tension of aqueous vapor at 26° (Tables). To change 1 kilogram of water at zero to saturated vapor at 26° requires, according to Regnault's formula, 606.5 + 0.305 (26) = 614 Calories. (The real latent heat of vaporization at 26° is, therefore, 614 — 26 = 588 Calories.) The heat required to evaporate 1 kilogram of water under the conditions of the peat kiln is therefore:

Heat to raise to vapor at 25 m. m. (26°) = 614.0 Calories
Heat to raise vapor from 26° to 50° (constant pressure) = [0.42 + 0.000185 (50 + 26)] × (50 — 26) = 10.4 "
Total = 624.4 "

In most cases of drying or evaporation the results will be sufficiently accurate by assuming the water first evaporated at 0°, with latent heat of evaporation 606.5 Calories, and then raised as gas to the end temperature. The difference between this method and the above will be small, where the temperature of the issuing gas is below 100° and the proportion of vapor in it small; where the temperature of issuing gas is high and the amount of vapor in it large, the more exact method should be used. To facilitate this, the maximum tension of aqueous vapor for temperatures up to 100° is here given:

Temperature C°	*Max. Tension, m. m. of Merc.*	*Temperature C°*	*Max. Tension, m. m. of Merc.*
0°	4.6	50°	92.0
5°	6.5	55°	117.5
10°	9.1	60°	148.9
15°	12.7	65°	187.1
20°	17.4	70°	233.3
25°	23.5	75°	288.8
30°	31.5	80°	354.9
35°	41.8	85°	433.2
40°	54.9	90°	525.5
45°	71.4	95°	633.7
50°	92.0	100°	760.0

Beryllium Oxide, Be^2O^3:

Sm (0° — 100°) = 0.2471 (Nilson and Pettersson)

Boric Oxide, B^2O^3:

Sm (16° — 98°) = 0.2374 (Regnault)

Carbonous Oxide, CO:

Sm (0° to t°) 1 m^3 up to 2000° = 0.303 + 0.000027t (Mallard and Le Chatelier)
1 m^3 2000° — 4000° = 0.2575 + 0.000072t (Berthelot)
1 ft^3 up to 2000° = 0.0189 + 0.0000017t lb. Cal.
1 ft^3 2000° — 4000° = 0.0161 + 0.0000045t lb. Cal.
1 kilo. up to 2000° = 0.2405 + 0.0000214t Cal.
1 kilo 2000° — 4000° = 0.2044 + 0.000057t Cal.
1 lb. up to 2000° = 0.2405 + 0.0000214t lb. Cal.
1 lb. 2000° — 4000° = 0.2044 + 0.000057t lb. Cal.
1 ft^3 up to 3600° *F* = 0.0341 + 0.0000017*t* B. T. U.
1 ft^3 3600° — 7200° *F* = 0.0290 + 0.0000045*t* B. T. U.
1 lb. up to 3600° *F* = 0.4329 + 0.0000214*t* B. T. U.
1 lb. 3600° — 7200° *F* = 0.3679 + 0.000057*t* B. T. U.

Carbonic Oxide, CO^2:

There has been much doubt about the specific heat of this gas, and several experimenters have given it particular attention. Direct combustion of CO to CO^2 in air has recently given Mallard and Le Chatelier a directly-observed temperature of 2050°, while the formula, which has been so far considered by us as the most reliable (Sm = 0.37 + 0.00027t), leads to 1947°. After renewed consideration of the whole subject the writer considers the best values those given below, because by accepting these they will agree with the observed temperature of combustion, 2050°. We will hereafter use these values instead of the one just above. Above 2000° the values of Berthelot and Vielle are the only ones to be used.

Sm (0° to t°) 1 m^3 up to 2050° = 0.37 + 0.00022t (calculated by Richards)
1 m^3 2000° — 4000° = 0.815 + 0.0000675t (Vielle)
1 ft^3 up to 2050° = 0.023 + 0.000014t lb. Cal.
1 ft^3 2000° — 4000° = 0.051 + 0.0000042t lb. Cal.
1 kilo. up to 2050° = 0.19 + 0.00011t Cal.
1 kilo. 2000° — 4000° = 0.42 + 0.000034t Cal.
1 lb. up to 2050° = 0.19 + 0.00011t lb. Cal.
1 lb. 2000° — 4000° = 0.41 + 0.000034t lb. Cal.
1 ft^3 up to 3700° *F* = 0.041 + 0.000014*t* B. T. U.
1 ft^3 3600° — 7200° *F* = 0.092 + 0.0000042*t* B. T. U.
1 lb. up to 3700° *F* = 0.34 + 0.00011*t* B. T. U.
1 lb. 3600° — 7200° *F* = 0.72 + 0.000034*t* B. T. U.

In all the above formulæ, Q from t° to 0° is equal to Sm multiplied by t; *e. g.*, Q. (0° to t°) 1 m^3 to 2050° = 0.37t + 0.00022t^2.

Nitric Oxide, NO^2:

Sm (27° — 280°) 1 m^3 = 1.35 (Berthelot and Ogier)
1 kilo. = 0.65 Cal.

Magnesium Oxide, MgO:

Sm (24° — 100°) = 0.2440 (Regnault)
Sm (0° — t°) = 0.2420 + 0.000016t (assumed)
$Mg(OH)^2$ Sm (19° — 50°) = 0.312 (Kopp)

Aluminium Oxide, Al^2O^3 (alumina, corundum):

Sm (0° to t°) = 0.2081 + 0.0000876t (constant by Regnault, coefficient of t by Richards, on a corundum crystal tested up to 1200°)
Melting point = about 2200°
Latent heat of fusion = 2.1 T = 51,933 Cal. for Al^2O^3
$= \frac{51,933}{102} = 509$ Cal. per kilo.
Heat in solid at 2200° = 881 Cal. by formula
Total heat in liquid to 0° = 1,390 Cal.
Specific heat, liquid = 0.593 = specific heat of solid at M. P.

Silicon Oxide, SiO^2 (silica, quartz):

Sm (0° to t°) = 0.1833 + 0.000077t (constant by Regnault, coefficient of t by Richards, on clear quartz up to 1200°)
Latent heat of fusion (at 1750°) = 135 Cal. (Voigt)
Melting point = 1900° (Boudouard)
Latent heat of fusion = 2.1 T = 6,563 Cal. for SiO^2
$= \frac{6,563}{60} = 109.4$ Cal. per kilo.
= 135 Cal. (Voigt)
Heat in solid at 1900° = 626 Cal. by formula
Total heat in liquid to 0° = 761 Cal.
Specific heat, liquid = 0.476 = specific heat of solid at M. P.

Sulphurous Oxide, SO^2:

Sm (16° — 202°) 1 m^3 = 0.4447 (Regnault)
1 kilo. = 0.1544

If we assume that the molecular specific heat of SO^2 is 8 Calories (from the rule that the molecular specific heat of a gas at constant pressure is 5 + n, where n is the number of atoms in the molecule) we would have

$$S\ (\text{at } 0^\circ)\ 1\ \text{m}^3 = \frac{8.}{22.22} = 0.36\ \text{Cal.}$$

Combining this with Regnault's value of Sm, we would get the formula

Sm (0° to t°) 1 m^3 = 0.36 + 0.0003t
1 kilo. = 0.125 + 0.0001t

These values are probably accurate enough for furnace calculations, and are very useful in pyritic smelting and the roasting of sulphide ores. While it is always desirable to have direct determinations of such important quantities, yet when they have never been made it is allowable to work out and use the most probable values.

Sulphuric Oxide, SO^3:

The specific heat of this important gas has not been measured. As an approximation we may assume

S (at 0°) per molecule = 9.0 Cal. (assumed)
per m³ = 0.405 Cal.
per kilogram = 0.10 Cal.

For getting an approximation to Sm we may assume the coefficient of t the same as in NH^3, which contains the same number of atoms and is of analogous formula, and we then have

Sm (0° to t°) per molecule = 9.0 + 0.0036t
per m³ = 0.405 + 0.00017t
per kilogram = 0.100 + 0.00004t

Calcium Oxide, CaO (lime):

This important datum has not been determined, but since it is so closely analogous to MgO, an approximation may be obtained by assuming that the molecular specific heats of the two compounds are alike. That for MgO is 0.2440 × 40 = 9.76 Calories. That for CaO would therefore be

$$Sm\ (24° - 100°) = \frac{9.76}{56} = 0.1743$$

and assuming a usual coefficient of t; for compounds of this type

Sm (0° — t°) = 0.1715 + 0.000071t

Titanic Oxide, TiO^2:
S (up to 200°) = 0.1790 (Nilson and Pettersson)
Q (0° to 200°) = 35.8 Cal.
Q (0° to t°) for t over 200° = 35.8 + 0.1790 (t — 200) + .000055 (t — 200)²

Chromium Oxide, Cr^2O^3:
Sm (10° — 99°) = 0.1796 (Regnault)

Manganese Oxides:
MnO Sm (13° — 98°) = 0.1570 (Regnault)
Mn^2O^3 Sm (15° — 99°) = 0.1620 (Oeberg)
(Manganite) $Mn^2O^3H^2O$ Sm (21° — 52°) = 0.1760 (Kopp)
(Pyrolusite) MnO^2 Sm (17° — 48°) = 0.1590 (Kopp)

Iron Oxides, Fe^2O^3:
Sm (0° — t°) = 0.1456 + 0.000188t (Regnault and Richards)
Fe^3O^4 Sm (0° — t°) = 0.1447 + 0.000188t (Regnault and Richards)

Nickel Oxide, NiO:
Sm (13° — 98°) = 0.1588 (Regnault)

Copper Oxides:
Cu^2O Sm (19° — 51°) = 0.1110 (Oeberg)
CuO Sm (12° — 98°) = 0.1420 (Regnault)

Zinc Oxide, ZnO:
Sm (0° — t°) (to t = 1000°) = 0.1212 + 0.0000315t (Richards)

Arsenious Oxide, As^2O^3:
Sm (13° — 97°) = 0.1276 (Regnault)

Zirconium Oxide, ZrO^2:
Sm (0° — 100°) = 0.1076 (Nilson and Pettersson)

Columbic Oxide, Cb^2O^5:
Sm (0° — t°) = 0.1037 + 0.000070t (Kruss and Nilson)

Molybdic Oxide, MoO^3:
Sm (21° — 52°) = 0.1540 (Kopp)

Tin Oxide, SnO^2:
Sm (16° — 98°) = 0.0936 (Regnault)

Antimonious Oxide, Sb^2O^3:
Sm (18° — 100°) = 0.0927 (Neumann)

Tungstic Oxide, WO^3:
Sm (18° — 98°) = 0.0798 (Regnault)

Mercuric Oxide, HgO:
Sm (5° — 98°) = 0.0518 (Regnault)

Lead Oxide, PbO:
Sm (22° — 98°) = 0.0512 (Regnault)

Bismuth Oxide, Bi^2O^3:
Sm (20° — 98°) = 0.0605 (Regnault)

Thoric Oxide, ThO^2:
Sm (0° — 100°) = 0.0548 (Nilson and Pettersson)

For those metallic oxides whose specific heat have not been determined, an approximation to the specific heat can be found by assuming that the metallic atoms in a molecule of oxide have each a specific heat of 6.4 Calories, and each atom of oxygen in an oxide molecule has an atomic specific heat of 3.6. As a rough approximation, we can further assume that the mean specific heat from 0° to t° increases 0.004 per cent for each degree of temperature, so that the above calculated specific heat being from 0° to 100°, we can figure out S at zero and the rate of increase of S or Sm with t.

Example: What is the most probable value of the specific heat of CaO? Since molecule weight is 56, and the molecule may be assumed to have a molecular specific heat of 6.4 + 3.6 = 10.0 Calories, the mean specific heat per kilogram (0° — 100°) is 10.0 ÷ 56 = 0.1786. S at zero will then be 0.1786 ÷ 1.004 = 0.1779, and we would have the formulæ:

$$Sm = 0.1779 + 0.000071t$$
$$S = 0.1779 + 0.000142t$$
$$Q = 0.1779t + 0.000071t^2$$

Problem 11.

The Jacobs process of producing artificial emery consists in melting down impure bauxite in an electric furnace, and letting the mass cool. Assume the bauxite to be calcined before use, and to contain

	Per Cent.
Alumina (Al^2O^3)	88
Ferric oxide (Fe^2O^3)	5
Silica (SiO^2)	5
Titanic oxide (TiO^2)	2

Required:

(1) The heat necessary to just melt a metric ton of this material at 2000°.

(2) The net electric power theoretically required.

(3) The total electric power actually required.

Solution:

(1) Because of the presence of the impurities the alumina melts easier than it otherwise would. We can, therefore, calculate the heat in the alumina melted at 2000°, and the heat in the others melted at their proper melting points, because they absorb their latent heats of fusion at the lower temperatures. Take 1000 kilos. of material.

Heat in 1 kilo. of alumina:
Q (0° — 2000°) = 0.2081 (2000) + 0.0000876 (2000)² = 767 Cal.
L. H. F. at 2000° = 2.1 (2000 + 273) ÷ 102 (mol. wt. Al^2O^3) = 47 "
Total = 814 "

Heat in 1 kilo of ferric oxide:
Q (0° — 1600°) = 0.1447 (1600) + (assumed) 0.000188 (1660)² = 713 "
L. H. F. at 1600° = 2.1 (1600 + 273) ÷ 160 = 25 "
Heat in liquid = (2000 — 1600) × 0.59 = 236 "
Total = 974 "

Heat in 1 kilo. of silica:
Q (0° — 1900°) = 0.1833 (1900) + 0.000077 (1900)² = 626 "
L. H. F. (Voigt) = 135 "
Q (1900° — 2000°) liquid = 0.476 × 100 = 48 "
Total = 809 "

Heat in 1 kilo. of titanic oxide:

$Q\ (0^\circ - 2000^\circ) = 35.8 \times 0.1790\ (1800) + 0.000055\ (1800)^2 = 536$ "

L. H. F. $= 2.1\ T = 2.1\ (2000 + 273) \div 80 = 59$ "

Total = 595 "

Total heat to fuse 1000 kilos.:

Heat in alumina	880 × 814 =	716,320 Cal.
Heat in ferric oxide	50 × 974 =	48,700 "
Heat in silica	50 × 809 =	40,450 "
Heat in titanic oxide	20 × 595 =	11,900 "
	Total =	817,370 "

(2) One kilowatt equals 859 Calories per hour.

Kilowatt-hours absorbed by melt $= \dfrac{817{,}370}{859} =$ **951.5** kw-hours.

(3) Assume an efficiency of 50 per cent, requires **1903** kw-hours. If the dynamos work at 95 per cent efficiency, the working power of the water turbines, gas engines or steam engines would be $\dfrac{1903}{0.95} =$ **2000** kw-hours.

Problem 12.

The ferric oxide charged into a blast furnace is reduced at 900° C. by carbonous oxide gas, CO, according to the reaction:

$$Fe^2O^3 + 9CO + (13.15N^2) = Fe^2 + 3CO^2 + 6CO + (13.15N^2)$$

Required:

(1) The thermal value of the reaction at 900°.

(2) What would be the temperature of the resulting products after the reaction had occurred?

Solution:

(1) The 6CO and $13.15N^2$ occurring on both sides of the equation need not be considered. The heat value of the equation at *ordinary temperatures* would therefore be:

Decomposition of Fe^2O^3	= —	195,600 Cal.
Decomposition of 3CO	= —	87,480 "
Formation of $3CO^2$	= +	291,600 "
Thermal value of reaction	= +	8,520 "

At 900°, the value is changed as follows:

Added:

Heat in Fe^2O^3 (160 kilos) at 900° = $160\ [0.1447\ (900) + 0.000188\ (900)^2] = +\ 45{,}200$ "

Heat in 3CO (= $66.66m^3$) at 900° = $66.66\ [0.303\ (900) + 0.000027\ (900)^2] = +\ 19{,}640$ "

Total to be added = + 64,840 "

Subtracted:

Heat in Fe^2 (112 kilos.) at 900° = $112\ (0.218t - 39) = 17{,}585$ "

Heat in $3CO^2$ ($66.66m^3$) at 900° = $66.66\ [0.37\ (900) + 0.00022\ (900)^2] = 34{,}080$ "

Total to be subtracted = 51,665 "

Heat of reaction at 900° = 8,520 + 64,840 — 51,665 = + **21,695** "

(2) The above heat would serve to raise the temperature of all the products, viz.: Fe^2, $3CO^2$, 6CO and $13.15N^2$ (from blast).

Their thermal capacity, 900° to t, is:

$Fe^2 = 112\ [0.218\ (t - 900)]$

$3CO^2 = 66.66\ [0.37\ (t - 900) + 0.00022\ (t^2 - 900^2)]$

$6CO = 133.33\ [0.303\ (t - 900) + 0.000027\ (t^2 - 900^2)]$

$13.15N^2 = 13.15\ [\quad " \qquad " \quad]$

Sum $= 0.018625t^2 + 93.47t - 99.205 = 21.695$ Calories

Whence $t = 1046^\circ$

It follows from the preceding discussion that the reduction of iron oxide by CO, at about 900°, is an exothermic reaction, heat being evolved, and that the reaction proceeds to a finish when once started. The relations for FeO would be probably similar, but we do not have the specific heat of FeO to use in making the exact calculation.

Problem 13.

If Fe^2O^3 charged into a blast furnace were reduced by solid carbon, at 900° C., by the reaction:

$$4Fe^2O^3 + 9C = 3CO^2 + 6CO + 4Fe^2$$

Required:

(1) The thermal value of the reaction at 900° C.

(2) The temperature of the resultant products after the reaction.

Solution:

(1) Value of the reaction at zero = — 4 (195,600) Cal.
+ 3 (97,200) "
+ 6 (29,160) "
= — 315,840 "

Add:

Heat in $4Fe^2O^3$ at 900° = + 180,800 "

Heat in 9C at 900° = $9(12)\ [0.2142\ (900) + 0.000166\ (900)^2] = +\ 35{,}340$ "

Subtract:

Heat in $4Fe^2$ at 900° = — 70,340 "

Heat in $3CO^2$ at 900° = — 34,080 "

Heat in 6CO at 900° = $6(22.22)\ [0.303\ (900) + 0.000027\ (900)^2] = -\ 39{,}275$ "

Algebraic sum = — 243,395 "

Absorbed per molecule of Fe^2O^3 = — 60,850 "

This reaction is therefore strongly endothermic, and therefore tends to check itself constantly by the resultant cooling effect.

(2) If the Fe^2O^3 and C were at 900° to start with, the products would be below 900° after the reaction, since they would have to furnish the deficit of heat.

The products would give out in cooling from 900° to t°.

$4Fe^2 = 4(112)\ [0.218\ (900 - t)]$

$3CO^2 = 66.66\ [0.37\ (900 - t) + 0.00022\ (900^2 - t^2)]$

$6CO = 133.33\ [0.303\ (900 - t) + 0.000027\ (900^2 - t^2)]$

Sum $= -\ 0.0183t^2 - 162.73t + 161{,}250 = 243{,}395$ Cal.

Whence $t = -\ 537^\circ$

This result is, of course, due to the hypothetical assumption that this endothermic reaction would go on until completed without checking itself. The impossible result obtained means that if this single reaction really goes on, it will soon check itself on account of the decrease of temperature.

Colby Kjellin Furnace.

The legend under Fig. 3, on page 299 of our August issue, contains an unfortunate error. It should read "Diagram of Colby Furnace" instead of "Diagram of Kjellin Furnace." This error will probably have been evident to our readers from the accompanying remark that the diagram was reproduced from patent 428,552, granted on May 20, 1890—this is one of Mr. Edward A. Colby's patents. The diagram is most interesting since it shows how much of the latest designs of the induction furnace was anticipated by Mr. Colby as early as 1890.

As already mentioned on page 134 of our April issue there were granted three fundamental patents to Mr. Colby in 1890. They are Nos. 428,378, 428,379 and 428,552, May 20, 1890. The first footnote on page 299 of our last issue should be corrected accordingly.

The Electrochemistry of the Metallic Arc.

By Isador Ladoff

(Continued from page 303.)

Child determined the following data of comparative resistance of the metallic electrodes.

Metal.	Resistance of Anode Volts.	Resistance of Cathode Volts.
Zinc	12	14
Iron	13	15
Copper	11	14

V. v. Lang (Wied. Ann. 31, p. 384, 1887) determined the tension of different metallic electrodes and expressed them by means of Froelich's formula. The constant "a" in this formula, called by V. v. Lang the "counter electromotive force" of the arc, represents the tensions below which no arc can be maintained.

The results secured by V. v. Lang we put here side by side with the data secured by Monasch (Wied. Ann. 58, p. 73, 1896). The first data for "a" relate to arcs in the usual atmosphere, the second in an atmosphere of nitrogen:

Metals.	Constant "a" in Air. Volts.	Constant "a" in Nitrogen. Volts.
Platinum	27.41 ± 1.16	29 — 31
Aluminium		26 — 29
Nickel	26.18 ± 2.95	
Magnesium		21 — 23
Iron	25.03 ± 2.16	19 — 22
Copper	23.86 ± 1.33	29 — 32
Zinc	19.86 ± 2.27	21 — 22
Silver	15.23 ± 0.45	
Cadmium	10.28 ± 3.38	21 — 22
Lead		18

It appears obvious that each metal has its own constant "a," the value of which is smaller for nitrogen than for air. V. v. Lang noticed that the constant "a" is higher for metals with a higher melting point. For carbon, having the highest melting point, the constant "a" is the highest, namely, 40 volts. Silver, which ought to possess a higher value of "a," taking in consideration its melting point, represents the only exception. In 1897, V. v. Lang (Wied. Ann. 62, p. 569, 1897) found another exception in the constant "a" of aluminium, namely, "a" = 30 volts.

Feussner claims that there is a fixed relation between the constant "a" and the boiling point of metals.

Guye and Monasch (Arch. des Sciences Physiques et Naturelles [Geneve] [14] 15, 15, III, 1903) found that when the length of the arc and the current of a metallic arc are kept constant the electric tension is higher in proportion as the atomic weight of the respective metal is higher. The following table represents the data procured when a current of 0.04 amps. (alternating of 47 periods per second) and an arc of 5 mm. length was maintained:

METALS.

	C	Mg	Fe	Ni	Cu
Atomic weight	12	24	55.9	58.6	63.2
Tension	640	700	850	850	870
	Ag	Cd	Pt	Au	
Atomic weight	107.7	115.5	194.3	196.7	
Tension	900	725	1000	1040	

Cadmium appears to be the only exception to the rule.

Schulze (Dissertation, Hannover 27, XI., 02, pp. 25-31) demonstrated that the groups of the periodic system possessing higher atomic weight and higher melting points are distinguished by a comparatively higher drop of potential in the arc, and especially a larger resistance of the anode. Schulze claims that within the same group the loss of tension in the arc is decreasing with the increase in atomic weight (in earthy alkalies). Otherwise there is a certain similarity between the drop of tension in the gaseous arc and the conductance of electricity in the solid body.

Arons (Drud. Ann. 1, p. 700, 1900) investigated the decrease of tension with decrease of pressure below one atmosphere in nitrogen gas in various metallic arcs. For cadmium, for instance, he found the following data: When the current was kept at 1.6 amps. and the length of the arc 1.5 mm. the voltage varied with the pressure as follows: for cadmium,

Pressure in Hg. mm.	10	60	100	220	380	600	750
Volts	12	16	17	21	22	23	23

Magnesium electrodes were tried. The arc was kept at 1.4 mm. length and the current at 4.5 amps., with the following results:

Pressure in Hg. mm.	90	220	360	490	660
Volts	17	17	20	20	22

The tension of the electrodes fell in proportion as the pressure dropped to 20 mm. of mercury.

Guye and Monasch investigated the behavior of the tension of copper electrodes during a diminution of pressure below 1 atmosphere. They used an alternating-constant current of 0.058 amps. of high tension at various constant lengths of the arc, and found that the tension drops together with the pressure. However, the critical point could not be reached. The arc changed entirely its aspect when the pressure fell very low, and turned into a series of spark discharges, as it is observed in rarified gases (Geissler tubes). The arc finally disappeared entirely, and the electrodes were surrounded with violet light, which covered a larger portion of the electrodes the lower the pressure dropped. The space between the electrodes, occupied at the usual pressure by the arc, was filled with a crimson and violet-colored glow, interrupted by a dark zone in the middle. When the current is kept constant and the pressure fluctuates periodically it causes a periodical fluctuation in the tension. When the tension is kept constant the current fluctuates periodically with the pressure.

Very interesting are the changes produced by the temperature in the tension of metallic electrodes.

Becquerel (Ann. Chim. Phys. [31], p. 355, 1853) already proved that electric discharges pass through longer distances and at lower tension in a warmer than in a colder electrode. De La Rive (Archives de l'Electricité 1, p. 262, 1841) heated various electrodes by the means of an alcohol lamp and found that the duration of the arc increased with the temperature of the electrodes.

Tommasi (Comptes Rendus Hebd. de Seances de l'Accad. de Sciences Paris 93, p. 716, 1881) passed an arc between horizontally arranged U-formed copper tubes. He watched the difference between the behavior of the arc when the tubes were artificially cooled by a current of water and then when they were at normal temperature. The luminosity of the arc dropped with the drop of temperature. At the lowest temperature attained the arc approached in its nature spark discharges, and could be easily extinguished. At all times, however, the sparks were colored green, indicating the vaporization of the copper.

Lecher (Sitz. d. K. Acc. d. Wiss. Wien 95, II., p. 992, 1887) experimented with platinum electrodes of 5 mm. diameter arranged horizontally. At a distance of about 2 mm. the electrodes showed about 35 volts tension. After he surrounded the electrodes with a spiral of copper wire, in order to lower their temperature, and repeated the experiment he found the tension lowered to 26 volts. In copper pencils of 4.4 mm. diameter the temperature proved to be so low that only the influence of heating was investigated. The tension of the pencils at 2 mm. distance was about 26 volts. After the heating of one electrode the tension rose to 28 volts, and this rise was greater when the negative was heated than the positive. Silver electrodes of 4.9 mm. diameter at a distance of 2 mm. showed a tension of about 20 volts. The tension increased to 28 volts when the negative pencil was heated, and to 23 volts when the positive pencil was heated.

The investigation of Arons (Wied. Ann. 58, p. 81, 1896) shows different results from those just presented, namely, a loss of tension when the temperature surrounding the pencils drops. R. Herzfeld doubts (Wied. Ann. 62, p. 442, 1897) if the cooling effect of the copper wire used by Lecher in his experiments was sufficient to produce results. Herzfeld threw a stream of fluid carbonic acid, kept under 38 atmospheres pressure, on the tips of metallic electrodes in order to notice the effect of cooling on the tension. The current was lowered while the tension increased. Herzfeld admits, however, that this result may be due to some chemical causes. The resistance of metallic electrodes increases with the temperature, while the opposite of it is true of carbon electrodes.

Guenther Schulze (Dissertation Hannover 27, XI., 02, p. 33) found that the resistances of both electrodes, cathode and anode, increases when they are cooled down, and that the resistances drop when the artificial abstraction of heat from the electrodes is diminished. G. Schulze could measure the dependence of the resistances of the electrodes from temperature only in the best conductors of heat, copper and silver.

When an arc of 10 mm. length is formed between copper pencils by means of a high-tension alternating current of 0.04 amps., the arc is apparently very quiet and surrounds the tips of both electrodes uniformly. However, when an alternating current of the same amperage is forced across the same electrodes, at a distance of 12 mm., the arc is very unsteady. (Guye and Monasch, Ecl. El. 34, p. 305, 1903.) The one end of the arc raises above the edge of the pencil and then leaps around, climbing on the cylindrical part of the electrode. Frequently the flame of the arc divides itself into several branches. The location of the arc changes every moment when it is about 11 mm. long. There seems to be no regularity in the wandering of the arc from place to place under such conditions. The zone in which the arc is unsteady the investigators call the "unquiet zone" (unruhige zone). In this zone the distance between the pencils is not equal to the length of the arc, but the arc is longer than this distance. As each variation of the length of the arc at constant amperage results in a variation of the tension, the needle of the voltmeter inserted in the circuit while the arc is in the unquiet zone is moving to and fro around a certain value of tension, and it is impossible to take a reading of the voltage. The zone in which the arc is quiet in a high-tension alternating current, and behaves similar to an alternating-current arc between carbons at low tension in respect to the mutual relation of their variable values, is called by the investigators the "normal zone." Any "normal" arc turns unsteady when the distance between the pencils is increased beyond a certain length. The distance between electrodes, causing unquietness of the arc, generally depends on the nature of the electrode. In the experiments conducted by Guye and Monasch the length of the arc at a constant current of 0.04 amps., above which the arc turned unsteady, proved to be as follows:

	mm.
Magnesium	15.0
Cadmium	13.0
Silver	11.0
Copper	10.5
Nickel	9.5
Iron (chemically pure)	8.0
Platinum	6.0
Aluminium	5.0
Gold	4.5

In the "normal" zone it is possible to take careful readings of the electric values. The regularity prevailing in the carbon arc is characteristic of a metallic arc at high tension. The effective tension of metallic electrodes producing an arc in a high-tension alternating current increases with the distance between the pencils occupied by the arc if the efficient current remains constant. When the length of the arc is kept constant a lower tension corresponds to a higher current. Guye and Monasch investigated the high-tension alternating-current arc of the following metals: Silver, copper, platinum, gold, nickel, iron, aluminium, magnesium and cadmium. In all metals the relation between the length of the arc and tension of the pencils could be expressed graphically in a straight line if the current is kept constant. Only the aluminium arc was unsteady under the same conditions under which other metallic arcs remained normal, on account of the formation of a thick crust of earthy, insulating oxides, over and above which the arc climbs to the unoxidized parts of the pencils. The application of Froehlich's formula to high-tension alternating current arcs has little value, as the constants "a" and "b" vary with each current.

(To be Concluded.)

Notes on Electrochemistry and Metallurgy in Great Britain.

(From Our Special Correspondent.)

THE FARADAY SOCIETY.

Of the papers discussed at the July meeting of the Faraday Society, first place was given to that by Mr. Sherard Cowper-Coles, on the rapid deposition of copper. The first speaker was Mr. W. R. Milburne, who stated that a plant put up on the Continent had fulfilled all Mr. Cowper-Coles' claims, demonstrating the value of the centrifugal process on a practical scale. Indeed, the larger the sheets required, the better had been the results. There had, of course, been a few minor initial difficulties, such as driving the mandril and securing the circulation of the electrolyte, but these had all been overcome. Mr. Prebble referred to the value of the centrifugal process in repelling the air bubbles, and stated that circulation began at the right place, the electrolyte acting as a liquid burnisher. Dr. Steinhart referred to the importance of the industry, particularly in the United States, where 250,000 to 300,000 tons of copper were refined electrolytically in each year. He regarded the process as eminently suited to the duties described in the paper. He wants, however, to know whether impurities were present in the anode, and, if so, with what percentage of impurities was it possible to work. He congratulated the author on having prepared a paper which, as a whole, was the most interesting that the society had had for a long time. Mr. Cloud spoke as one having no practical experience of rapidly rotating cathodes. He disagreed with the author's view that the formation of nodules on plates was due to a small particle of impurity. He (the speaker) had found that the use of glue gave smooth plates, but was unable to explain the reason. The smoothness obtained by Mr. Cowper-Coles' method was not due so much to a burnishing action of the electrolyte, but to a dragging out of the crystal. Two communications were then read, the one from Mr. McKenzie asking for particulars of the voltage and current density employed, and the other from Mr. A. S. Elmore, which virtually amounted to an abstract of a paper on the Elmore process. He claimed that the Elmore process had yielded more than Mr. Cowper-Coles claimed for his centrifugal process, and asked whether Mr. Cowper-Coles could make tubes of less than 2½ inches diameter. In reply Mr. Cowper-Coles referred those who had asked for details of voltage and current density to a diagram which gave these. In reply to Dr. Steinhart's query, Mr. Cowper-Coles stated that he had used Chili bars for his anodes, and the lowest grades with which they had worked contained 93 per cent of copper. As regards nodules, he believed that the seat of each of these was a speck of dirt in mechanical suspension, but not in solution. Sir Joseph Swan held this view. As to Mr. Elmore's communication, it was easiest in all electrolytic processes to obtain relatively large tubes and draw these down.

Prof. Haldane Gee's paper on "The Use of Balanced Electrodes, was introduced by Mr Hutton, who spoke of the trouble involved in weighing the electrodes in a voltameter and the manifest value of a continuous balance for the purpose. The discussion on the paper was practically confined to a few observations by Dr. F. Mollwo Perkin, who remarked that the subject was one of considerable interest to him, as he had endeavored to produce a coulomb-meter which should act as a substitute for the voltameter. He had found that large anode surfaces were necessary when using a mercury coulomb meter, which should be worked across the terminals of a shunt resistance placed in the main circuit. The best coulomb-meter was undoubtedly a gravity balance with the cathode hung on one arm.

Dr. Perkin then gave abstracts of the two papers by himself and Dr. Law, the one on the "Electrolytic Analysis of Antimony," and the other on the "Electrolytic Oxidation of Hydrocarbons of the Benzene Series. Part II., Ethyl Benzene, Cumene and Cymene." Partly owing to the lateness of the hour, and perhaps in part to an unwillingness to challenge Dr. Perkin on questions in organic chemistry, the discussion on the second of these papers was confined to the propounding of two questions and their answers. Replying to Dr. Borns' query as to whether, if acetic acid were used in place of acetone, the substances would not go into carbon dioxide and a certain amount of tar, Dr. Perkin said that his experiments had yielded mainly alcohol and very little aldehyde. Acetic acid had not yet been obtained. Replying to Mr. W. R. Cooper no work had yet been done on the chain compounds—they had tried to saponify petroleum, but with no satisfactory result.

Mr. R. S. Hutton's synopsis of the paper on "Heat Insulation, Particularly with Regard to Materials Used in Furnace Construction," of which he was joint author with Mr. J. R. Beard, was received with considerable interest. A written communication criticised the use of the water jacket, and referred to some experiments recently carried out at Finsbury. In these no jacket was used, but the substance was raised to a uniform heat, and the amount of heat required to keep this constant was measured. The writer praised Canadian mica, saying it was as good as any of the media Mr. Hutton had tested. In reply, Mr. Hutton stated that he was aware of the experience referred to, and thought their method good enough for applying to steam pipe coverings. For furnace linings he defended the method described in his paper as better and of more practical utility.

Before actually adjourning for the holidays—the formal adjournment having begun when the vote of thanks to Dr Hutton and Mr. Beard was passed by acclamation—those present gathered around Mr. Cowper-Coles' specimens of centrifugally deposited copper, which were both numerous and interesting.

The Galbraith-Stewart Electrothermic Process of Producing Steel from Iron Sands.

In company with the representatives of the leading English technical papers, a number of members of the Faraday Society and some iron and steel manufacturers, I went to the Brush Works at Loughborough to view the Galbraith Stewart experimental furnace. This is of the resistance type, and consists of graphite bars. No meters were fitted for measuring the current supplied, but we were informed that this was supplied from a 100-kw. alternator, and was transformed down to about 18 volts. The graphite bars were arranged in parallel, the current entering at the front and leaving at the back. Contact is made with the bars by means of substantial iron blocks, pressed against the graphite grids by means of a weighted lever. The bars become incandescent, and fine granular iron sand is fed continuously from the top of the furnace, dropping into a receptacle at the bottom in a molten condition.

The invention is a New Zealand one, and is the latest attempt to treat the huge beach deposits of almost pure iron sand which have hitherto defied the metallurgist. An analysis by Mr. W. H. S. Shakell, which is quoted as typical of these sands, gives the following percentage analysis: Peroxide of iron, 67.04; protoxide of iron, 30.17; manganese peroxide, 0.22; aluminium oxide, 0.16; silica, 0.50; titanium, 1.60; calcium and titanium, traces, and 0.31 undetermined.

At the outset it was manifest that the furnace was still in the laboratory stage. For instance, the heat radiation was very large, the infeeding of cold sand slow (about 1 pound per minute), and the resultant material merely consisted of nodules of the metal. To dwell too much on these would be ungenerous, as Mr. Stewart, who, in a lucid and honest speech, modestly declared that the experiments were not on a commercial scale; the only change which could be shown was from the granular to the globular form. Ultimately, Mr. Stewart said, a molten bath would be provided whence the molten metal required for ingots could be run off. As exhibited the sand was mixed with a small quantity of carbon and fed cold into the furnace. In the future it was proposed to heat the sand with producer gas, bringing it to a red heat and feeding into the furnace in a reduced state, the conversion into steel being effected during its passage over and between the incandescent electrodes, thence falling into a hot bath for tapping. Strictly speaking, there would be no intermediate pig iron stage, the oxygen required being present in the form of peroxide and protoxide.

To those who, like your correspondent, went in the hope of seeing more than globules of iron or steel, the demonstration was disappointing. Some small ingots were exhibited, but these were the product of globules of the iron smelted in crucibles heated in an ordinary coke foundry furnace. Many obvious questions call for answer, such as the energy actually required, the life of the electrodes, losses in radiation and contact resistance, and analyses of the ultimate product. Several English metallurgists hold the view that induction furnaces of Kjellin type are better fitted than resistance furnaces for dealing with minute particles of iron. The Galbraith Stewart process has yet to vindicate itself; if it can do this it becomes at once evident that its field is not restricted to deposits of iron sand, but that magnetically separated particles of iron ore may be treated directly by it without the need for preliminary briquetting, concerning which one hears so many adverse comments, based on the fragile character of the briquettes.

Papers on Iron and Steel at the Mining and Metallurgical Congress at Liege.

I was unable to leave London for the meeting of this Congress, and, therefore, did not secure immediate copies of the papers read. My own copies, as a subscriber to the Congress, are, unaccountably, not yet to hand, so I am therefore constrained to send abstracts of the reprints of the papers read which have been published in contemporary English journals. Many of the papers which were presented partook of the nature of reviews of the progress along certain lines. Among these may be mentioned Mr. Hadfield's paper on "The Effect of Liquid Air Temperatures on the Properties of Iron and Steel," which was virtually a summary of the paper read last May at the Iron and Steel Institute. The paper by M. O. Ackers is a résumé of recent open-hearth steel processes. It contains, however, some particulars of the practice followed at the Jurgewka Works, Donetz, where the available ores are too high in phosphorus for the Bessemer process, while too low for the basic process. Here the furnaces are first charged with ore and heated in the furnace until the ladles of liquid pig are brought up and their contents poured into the furnace. The decarburization of the pig iron in contact with the mineral oxides takes place, and produces a violent reaction, which hastens the end of the operation; in five hours from the time of charging, 25 to 30 tons of steel may be run out. At the Jurgewka Works there are five 25 to 30-ton furnaces supplied with metal from a 150-ton mixer. The metal shows the following average composition:

Silicon.	Manganese.	Sulphur.	Phosphorus.
1.0 to 1.5	2.25 to 3.0	0.0 to 0.3	0.15 to 0.25

Krivoi-Rog ore, containing 65 per cent of iron and 0.10 per cent of phosphorus, is used.

The results obtained at these works are remarkable. During July, 1904, 13;754 tons of good ingots were made on a working equal to 140 days and 570 charges, or 98 tons per twenty-four hours per furnace. A 20 per cent saving of coal was effected. The average composition of a charge was 25,724 kilos. of liquid iron and 176 kilos of ferromanganese, 6240 kilos. of ore and 2400 kilos of fluxes. The working of the charge occupied five and one-half hours from charging to tapping. The composition of the pig iron used and of the steel obtained were as follows:

	C.	Si.	Mn.	S.	Ph.
Pig iron........	4.25	1.17	2.80	0.03	0.20
Steel	0.07	0.01	0.50	0.03	0.04

Five thousand kilos. of slag were produced containing:

Si.	Mn.	S.	Ph.	Fe.	CaO.	AlO.	MgO.
24.8	14.40	0.14	0.46	1.10	35.55	1.80	9.7

After describing the Bertrand-Thiel and Talbot processes, together with the Suzycki modification of the latter, the author concludes with the following summarized conclusions:

"Charging open-hearth furnaces with liquid pig iron is so convenient and economical that there is no doubt it should be adopted, and that with this view new open-hearth plants should always be erected in the vicinity of blast furnaces. The advantages arising from this selection of site are:

"1. Economy of labor in handling the output of the blast furnaces and in charging the steel furnace.

"2. The time of working a charge being diminished by using liquid pig iron, production per furnace can be increased.

"3. Manufacture of steel is independent of the quality or cost of scrap, of which the percentage proportion per charge can be adjusted to suit the supply.

"4. The steel works can reap the advantage accruing from the use of the waste blast furnace gases for power purposes.

"5. The fuel economy is considerable.

"When ores containing not too much phosphorus are available, as at Jurgewka, the ordinary process of charging liquid pig iron may be followed; with ores higher in phosphorus, but containing less than 1.8 per cent, the Bertrand-Thiel process may be tried; when a mild steel is required, the Talbot and Surzycki processes give good results. As long, however, as ores—and, consequently, pig iron—high in phosphorus are found abundantly, open-hearth processes in general will have a worthy rival in the basic Bessemer process."

Magnetic Alloys of Non-Magnetic Materials.

In my July letter I thought I had, for a while at any rate, exhausted my tributes to Mr. Hadfield's versatility. A paper on "The Magnetic Qualities of Some Alloys Not Containing Iron," of which Mr. Hadfield appears as joint author with Prof. Fleming, read on June 8 before the Royal Society, has disillusioned me. In connection with this research it is not the present commercial value of the products which counts, for that is poor, but the potentialities opened up, which are very great indeed. The alloys which formed the subject of the paper contained in the one case, manganese, 22.42 per cent; copper, 60.49 per cent; aluminium, 11.65 per cent. There was a certain amount of intermingled slag, probably 2 or 3 per cent, mostly consisting of MnO and SiO_2, and slight traces of other metals. Aanalysis showed that there was also present: Carbon, 1.5 per cent; silicon, 0.37 per cent, and iron, 0.21 per cent. Hence, it may be said that nothing but a trace of iron occurs in this sample of alloy. The other ring, No. 1,888/7, had an approximate composition: Manganese, 18 per cent; copper, 68 per cent; aluminium, 10 per cent; lead, 4 per cent. These alloys, unfortunately, have poor mechanical properties and are brittle and cannot be forged. Rings were cast from the material and turned in the lathe to the desired form. The rings having been carefully shaped, their dimensions were then measured. The ballistic method of testing was employed for the purpose of obtaining the magnetic data. Omitting the ten tables which appear in the paper, the following conclusions are reached respecting the behavior of the ring, whose analysis is first given: (1) the alloy exhibits magnetic properties which are identical with those of a feebly ferromagnetic material. (2) The magnetization curve is of the same general form as that of a ferromagnetic metal such as cast iron, and indicates that with a sufficient force a state of magnetic saturation would most probably be attained. (3) The alloy exhibits the phenomenon of magnetic hysteresis. It requires work to reverse the magnetization of the material and to carry it through a magnetic cycle. (4) The material has a maximum permeability of 28 to 30, which is not greatly inferior to that of the values reached for cobalt or a low grade of cast iron for small magnetic forces, and occupies a position intermediate between permeability of the ferromagnetic and the merely paramagnetic bodies, such as liquid oxygen and ferric chloride. (5) The material exhibits, therefore, the phenomenon of magnetic retentivity and coercivity. It is not merely magnetic, but can be permanently magnetized. The alloy composing the second ring, when tested, was found to have a flatter magnetization curve than the alloy No. 1871. Its hysteretic properties closely approximated to that of the first tested alloys. Both of these have far greater hysteresis than pure iron-nickel or cobalt for corresponding cycles of magnetization.

This, of course, is only a detailed investigation of Dr. Heusler's discovery in regard to which Mr. Hadfield exhibited a sample at the British Association Meeting at Cornbridge in 1904. But what is of importance is the hypothesis Mr. Hadfield and Prof. Fleming advanced that ferromagnetism is not a property *per se* of the chemical atom, but of certain molecular groupings. "In spite of the fact that ferromagnetism has been hitherto regarded as the peculiar characteristic of certain chemical compounds—iron-nickel and cobalt—it may, in fact, depend essentially on molecular groupings composed of a comparatively large number of molecules, and hence it may be possible to construct alloys which are as magnetic, or even more magnetic, than iron itself." In this sphere, with such a goal, a half decade is not too long a period to give to the thorough investigation of the question.

The Minimum Scientific Training.

Some remarks of Prof. Sir Alexander Kennedy's on "The Academic Side of Technical Training," made before the Union Society of University College, London, while neither electrochemical nor electrometallurgical in their context, are of such importance that I feel constrained to quote them here. Sir Alexander Kennedy, who is responsible for the design of the conduit system of electric traction adopted in South London by the London County Council, said: "The idea of giving a young man just as much mathematics, just as much physics, or just as much chemistry as the minimum that he can professionally require, is not only pernicious, but absolutely fallacious. I am sure that the only way of knowing a subject up to a certain point, in such a fashion that, up to that point, it can be thoroughly utilized, is to study the subject up to a point very much further advanced. It is not at all a valid objection to the teaching of any particular point in mathematics or physics, that it is more complicated or more advanced than anything which the engineer will be likely to require. That, in itself, is not an objection at all, because, as I have said, it is impossible really to master a scientific subject up to a certain, often very elementary, point without having at least a superficial knowledge of a much greater extent of the subject. But it is desirable, indeed necessary, from our point of view, that the advanced work in purely scientific subjects should be specially chosen, so as best to deepen and make certain the knowledge of the earlier work." All who have had to deal in any manner with the technical training of engineers or chemists, or have had to depute duties to the technically trained product of various institutions, will at once recognize the importance of these truisms, and impress them on their subordinates. The reason, perhaps, why so many fail to think intelligently is because their mental horizon is bounded by their daily occupations, which they fail to appreciate in an adequate manner, through being only partially trained, and which they do not extend by independent study.

Market Quotations

With regard to the alkaline products, prices are stationary, although the fear of labor troubles on the one hand and advanced prices for raw cotton on the other, will probably depress the Lancashire demand and tend to lower prices. In sympathy with the hardening in the price of copper, copper sulphate has risen to £21 per ton. The price of shellac continues its see-saw course, having risen during the month to £9 per cwt.

Cleveland pig iron remained steady at £2.5.6 per ton until July 12, then receding a fraction. The expected recovery began about the 21st, the closing price on July 31 being £2.6.6, and further rises are now expected. Copper rose steadily throughout the month to £68.10.0 per ton, and the indications to-day point to further rises. Tin has furnished the sensation of the month, touching the phenomenal price of £151 per ton on July 26, a rise of over £12 in twenty-three days. A slight fall to £149 took place by the 31st, but prices are hardening again, owing to shrinkage of supplies and the well-maintained demand. Lead has been firm in price, closing at £14.5.0 for ingots. Quicksilver is fetching from £7.5.6 to £7.7.6 per 75-pound flask.

London, Aug. 5, 1905.

ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS.

Electric Furnaces.

Electric Furnace.—F. A. J. FitzGerald and P. McN. Bennie, 792,255, June 13. Application filed Dec. 3, 1904.

The furnace chamber, shown in Fig. 1, is divided into any desired number of independent compartments 9, 10, separated by refractory walls. These walls comprise resistors 11 and refractory facings 12 which may be of the same character as the lining 6. The several resistors 11 are connected electrically in series by resistors 13 placed at opposite ends of adjacent compartments. The principal terminals are at 14, while 16 is an auxiliary terminal.

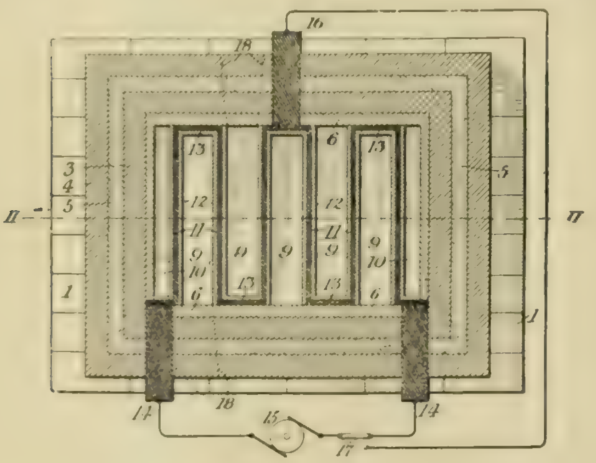


FIG. 1.—ELECTRIC RESISTANCE FURNACE.

Producing Tungsten Steel.—E. D. Kendall, 795,517, July 25. Application filed May 31, 1905.

Concentrates of tungsten ore are granulated and mixed with metallic zinc dust and fragments of iron, free from carbon. The mixture is treated in an electric furnace. The escaping zinc oxide fumes are collected in an outside chamber.

Electric Furnace.—H. N. Potter (assigned to George Westinghouse), 797,747, Aug. 22. Application filed July 23, 1902.

Claim 3 is for "the method of reducing the terminal voltage of a conducting gap, which consists in subjecting the said gap to the influence of great heat." The inventor uses the arc gap within a resistance furnace, so that his furnace is a combination of a resistance and an arc furnace, whereby the arc may be operated at a lower voltage than would be possible without the high temperature produced by the resistance heating. "In this manner it is possible to produce a range of temperatures beginning with that reached by the unaided [resistance] furnace and ending at temperatures at least as high as those of the electric arc." A great many different applications of this principle are dealt with, especially in connection with tube furnaces, as shown, for instance, in Fig. 2, where the tube furnace 4 is supplied with electrical energy from the alternator 21 through the intermediary transformer 22, 23, while a direct-current discharge is produced over the gap 4, 5 from the battery 24. The use of a granular resistance material is brought by the inventor under the same principle: two grains make contact at one point only; the current passing through this point heats the surrounding gap of gas with a resulting leakage current across the gap. Claim 11 is for "the method of producing heat inside an electric furnace, which consists in arranging a conducting resistance and a vapor resistance, one within the other, and heating both resistances simultaneously by the passage of electric current."

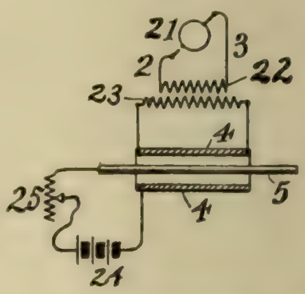


FIG. 2.—COMBINATION OF RESISTANCE AND ARC HEATING.

Electric Furnace.—C. L. Saunders, 794,255, July 11. Application filed Feb. 15, 1901.

A horizontal continuous annular trough forms a hearth which is rotated. The current is supplied through electrodes on the hearth; stationary contact pieces outside are connected with the supply circuit, and are engaged in moving contact by contact pieces on the hearth. The charge is fed at one place into the hearth, passes through the electrically heated zone and is then discharged. The gases escape through a pipe in the cover.

Apparatus for the Reduction of Iron Sand, Iron Oxide and other Suitable Substances.—D. R. S. Galbraith, 796,312, Aug. 1. Application filed Oct. 27, 1903.

"Iron sand" is first freed from all silicious matter (evidently by magnetic concentration) and then treated in the electric furnace *a* shown in Fig. 3. The charge consists of the iron sand (iron oxide) and the necessary quantity of powdered charcoal or other reducing agent (together with a suitable flux if there is some silicious matter, etc., still mixed with the iron sand). The charge falls down upon and between tiers of carbon rods *c*, which are made incandescent by the passage of an electric current, and may be covered at the upper surface with refractory roofs. A reducing gas is introduced through *g*, and electric resistances in form of rods *i* are inserted in this passage. The reduced and molten metal accumulates in the chamber *h*. Tap holes are provided *q* for the metal and *p* for the slag.

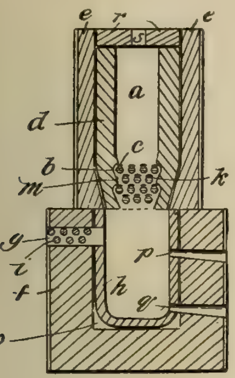


FIG. 3.—STEEL FURNACE.

Electric Oven.—L. E. Custer, 793,424, June 27. Application filed Sept. 14, 1904.

Details of construction of an electric muffle furnace for dentists. The floor and walls of the heating chamber are made of porous material, and provided with perforations or grooves in which crimped or looped wires are arranged so as to evenly distribute the heat.

Electrolytic Processes.

Making Aluminium.—A. G. Betts, 795,886, Aug. 1. Application filed April 1, 1905.

The intention is to prepare pure aluminium cheaply from ores without necessitating the purification of the ore before the metal is reduced. Aluminium oxide occurs in nature for the most part associated with other oxides, notably with those of iron and silicon in clay, and with those of iron, silicon and titanium in bauxite. All these oxides are more easily reducible than alumina, so that it is not possible to prepare pure aluminium by first reducing the ore to an alloy and then slag off the impurities by oxidation analogously to the purification of pig iron and most other metals. Betts' process consists in first forming such an alloy and then refining it electrolytically.

The crude ore is reduced directly to a metallic state with or without diluting it further by the addition of another metal, as copper, zinc or tin. The product is placed in the fused state at the bottom of an electrolyzing cell and forms the anode; above it comes a specifically lighter aluminium-depositing electrolyte (*f. i.*, cryolite saturated with alumina); a layer of fused pure aluminium at the top forms the cathode. Since aluminium is most easily oxidized of the metals present in the anode, it alone dissolves and is deposited on the cathode. One of the various applications of the process given by the inventor is as follows: Clay is melted with iron ore or with metallic iron or copper to alloys of copper or iron with silicon and aluminium, the aluminium is electrolytically extracted and ferrosilicon or silicon bronze are obtained as by-products.

Another result is that in the electrolytic reduction of aluminium oxide, pure or native, a cathode of zinc, tin or aluminium bronze may be used to receive the deposited aluminium, so that a liquid alloy at a temperature below the melting point of pure aluminium is obtained. Therefore, other baths than the fluoride bath of the Hall process may be used, for instance, fused sodium-aluminium chloride. The inventor also states that aluminium can be extracted by his process from such materials as aluminium silicide and carbide and similar non-oxygen-containing compounds of aluminium.

Stopper for Electrolytic Pots Containing Fused Baths.—C. M. Hall, 796,325, Aug. 1. Application filed April 30, 1902.

The tap-hole is filled with a plastic material, composed of powdered carbon and tar, and before this becomes hard a tapering metal plug is driven inward until its end nearly reaches the interior of the pot-lining. Although the plastic material hardens and becomes baked around the plug, the latter may be readily withdrawn when it is desired to open the hole.

Electrolytically Refining Silver.—A. G. Betts, 795,887, Aug. 1. Application filed March 8, 1905.

The silver alloy forms the anode. The electrolyte contains a free, non-oxidizing, monobasic strong acid, forming a soluble silver salt, the silver salt of this acid, and a small amount of gelatin or gum-arabic, to prevent the formation of trees. The proportion of the gelatin should be between 1 to 12,000 and 1 to 15,000 of solution. A suitable solution contains 4 per cent silver, as silver methyl sulphate, and 4 per cent methyl sulphuric acid. No necessity exists for wrapping the silver anodes in cloth. The anode slime is mostly gold. The bismuth from the anode accumulates in the solution and is precipitated with hydrofluoric acid, as bismuth fluoride, or after precipitating silver—for instance, with metallic bismuth—by precipitation with metallic lead. If the latter method is adopted the resulting lead solution can be treated with finely divided silver sulphate, precipitating lead sulphate and regenerating silver solution to be used over again.

Electrolytic Decomposition of Saline Solutions.—A. B. Larchar, 793,138, June 27. Application filed Oct. 9, 1900; Oct. 9, 1902 (assigned to Penobscot Chemical Fiber Co.).

This patent is an interesting indication of the tendency of paper and pulp mills in the Eastern States to erect their own electrolytic plants for making bleach and to develop their own processes, in which diaphragm cells are generally used. The special feature of the present patent is the construction of the cathode, which is a grating of vertical slats, extending all the way around the cell along the four side walls. The steel slats are placed near together, so that the channels leading from the inside to the outside are long and thin, so as to divide the liquid passing through into thin films. On the inside a diaphragm of asbestos paper is stretched on the grating. The whole inside of the cell forms the anode compartment, and contains graphite anodes. The portion of the cell outside of the diaphragm forms the cathode compartment. Fresh brine solution is introduced into the inside compartment, and is held there at a level slightly above the outside level. The electrolyte passing through the diaphragm is divided up into a great number of films between the slats of the grating; these films are in intimate contact with the cathode surface. The caustic soda which is formed outside of the inner cell flows off through an overflow pipe, the mouth of which may be raised or lowered at will, so as to adjust the difference of the levels of the solution in the anode and cathode compartments. By adjusting this properly, it is stated to be possible to prevent the caustic soda from flowing back into the inner cell, where it would form non-desirable compounds with the chlorine. The chlorine developed in the anode compartment passes off through a pipe.

Electroplating Device.—H. Schuessler, 796,872, Aug. 8. Application filed Sept. 17, 1904.

Mechanical details of a tank for electroplating sheet-metal plates. The tank has a horizontally inclined bottom with a longitudinally movable carrier for the plate to be coated. The anode is provided above this carrier and parallel with it.

Batteries.

Secondary Battery.—I. Kitsee, 793,881, July 4. Application filed March 17, 1893.

According to the inventor, the change of the capacity of a battery plate for storing energy bears a certain relation to the over-oxidation of the active material. In order to restore the "vitality" of a battery plate after a certain number of charges and discharges, the inventor reduces the higher oxides to lower states of oxidation. This may be done by a very slow process of heating in dry heat in a hermetically sealed vessel, or by the application of sodium chloride to the electrode moist with sulphuric acid; or by the action of sulphurous gases, etc. Claim 1 is for "the method of restoring the usefulness of overcharged active material in secondary cells, which consists in bringing said active material into contact with a solution adapted to in part reduce said active material to a lower oxide, and in part change the same to a salt adapted to act as binder for the particles of said lower oxide."

Battery Grid.—H. C. Porter, Waukegan, Ill. Patent 792,611, June 20, 1905. Application filed Aug. 19, 1904.

The object of the invention is the production of a plate in which the active material, or material to become active, is firmly secured against dislodgement and which is prevented from buckling. As shown in elevation and vertical section along line 22 in Fig. 4, the grid consists of a supporting body A, constructed with solid edges B. It is made up of a series of interior horizontal bars C, which are shown V-shaped on their upper and lower faces. The bars are provided at intervals with vertical openings D, preferably corresponding in the different bars in the same vertical plane. This method of construction is stated to provide, in addition to the metallic grid, solid columns of active material, or material to become active, forming, as it were, a supplemental grid of active material.

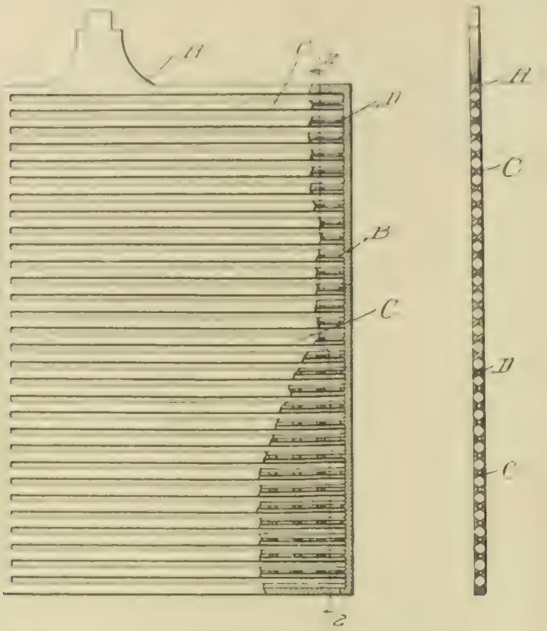


FIG. 4.—BATTERY GRID.

Battery Plate Separator.—H. C. Porter, Waukegan, Ill. Patent 792,612, June 20, 1905. Application filed Aug. 19, 1904.

The separator consists of a perforated sheet, preferably made of celluloid or hard rubber. It is provided with a series of loops, formed by two horizontal parallel cuts in the sheet, the cut portion having been bent outwardly, so as to form these loops, through which strengthening and stiffening rods can be inserted. By interposing these sheets between the electrodes the latter are not only separated from each other but are also kept out of contact with the separating sheet itself, inasmuch as they rest upon the above-mentioned rods. Cheapness of construction and durability are claimed for this separator.

Battery Grid.—H. C. Porter, Waukegan, Ill. Patent 792,613, June 20, 1905. Application filed Aug. 19, 1904.

The grid is made up of a body which constitutes the support for the active material, and is constructed with solid edges. It consists of a series of longitudinal triangular bars, which shape, however, is not essential. These bars are integrally attached to a back plate D, which is provided with a plurality of openings E. The latter have been formed after casting in such a manner that tongs are formed on the interior side of the back plate, so as to hold the material securely in place.

Galvanic Battery.—B. J. Blameuser, Chicago, Ill. Patent [illegible], June 13, 1905. Application filed Aug. 1, 1904.

The cell belongs to the carbon-zinc type, with a solution of sulphuric acid and potassium bicarbonate depolarizer. The specification describes at length various details of construction, in order to provide a more compact and simpler construction than the ordinary type.

Storage Battery.—G. K. Hartung, 797,110, Aug. 15. Application filed April 30, 1904.

In order to avoid metallic grids for the support of active material, he places the active mass in non-conducting acid-proof expansible tubes suitably mounted in frames. Carbon rods are embedded in the active mass to conduct the current to the terminals. The tubes are made of perforated, corrugated, thin celluloid.

Secondary Battery.—J. Langelaan (assigned to Pflueger Accum. Werke A. G.) 796,435, Aug. 8. Application filed Feb. 27, 1905.

Mechanical details of construction of plates of active mass, which are composed of single pieces and held together by a protective envelope. The main features are: first, horizontal supports with flanges, and second, connecting pieces which engage between the flanges of the supports upon the bent parts of the protective envelope.

Storage Battery Grid.—M. Heisser, 794,501, July 11. Application filed July 5, 1904.

The grid is a frame with a plurality of parallel cross-bars, Z-shaped in cross-section. They form V-shaped grooves in their upper and lower faces for retaining lead oxide. The flanges of the bars are wedge-shaped and terminate in sharp edges.

Battery Grid.—H. C. Porter, 793,093, June 27. Application filed Aug. 19, 1904.

Mechanical details of construction of a grid for posted plates.

Sheet Metal for Perforated Pockets of Storage Batteries. T. A. Edison, 797,845, Aug. 22. Application filed July 21, 1904.

In his nickel-iron battery the active masses are embedded within pockets of perforated nickel-plated steel. The pocket walls must be so thick that any change in the active mass during use will fall within the elastic limits of the sheet metal. However, if the thickness of the steel is chosen accordingly, it cannot be practically perforated with the desired fineness, since the dies become quickly worn out. He therefore makes the steel thinner (0.0025 inch) and the nickel-coating thicker (0.0005 to 0.00125 inch) than heretofore.

Storage Battery.—A. Meister and A. Junker, 794,240, July 11. Application filed Nov. 11, 1901.

Method of construction of a nickel electrode for use in an iron-nickel accumulator with an alkaline electrolyte. The plate is made of a large number of alternate strips of nickel and layers of pulp or other absorbing and porous material. The plate is formed as anode in a weak solution of caustic potash, containing a small quantity of phenylate against another sheet-nickel plate as dummy electrode; the anodic current density being 1 amp. per square inch.

Cathode Plate.—H. C. Hubbell, 793,078, June 27. Application filed Oct. 3, 1904.

An electrode with an active mass of silver for alkaline accumulators. A perforated foundation plate of nickelplated iron is coated on both sides with the active mass of finely divided silver; the silver extends through the perforations, and is thus held firmly to the foundation plate. Outside of the active mass of silver a layer of asbestos paper is folded around the plate, covering both sides. Around the asbestos paper is folded a strip of wire gauze of nickel-plated iron.

Cathode Plate.—H. C. Hubbell, 793,077, June 27. Application filed Oct. 3, 1904.

A nickel electrode for alkaline batteries. Hair-like nickel fibers (produced by electrolysis at a high current density) are mixed and intermingled with an oxide of nickel, which is moistened and formed into a dough-like or pasty plastic mass. This is formed or molded by kneading into the required shape, and placed into pockets of nickel-plated iron wire gauze.

Anode Plate.—H. C. Hubbell, 793,076, June 27. Application filed Oct. 3, 1904.

A zinc plate for use in caustic potash or soda, as electrolyte. The skeleton consists of a series of transverse sheet-steel troughs, one above the other; each trough forms a U-shaped gutter containing the zinc deposit.

Negative Electrode for Primary Batteries.—D. L. Winters, 797,787, Aug. 22. Application filed Jan. 30, 1905.

The active mass of copper oxide is contained in a casing of zinc-coated perforated sheet-iron. The zinc coating is intended to "set up at once a local electrochemical action and increase the conductivity of the negative electrode and the battery-electrolyte to attain an initial efficiency of the battery in a rapid manner."

Electric Battery.—P. J. Kamperdyk, 794,864, July 18. Application filed April 2, 1904.

Mechanical details of construction of a primary cell with porous plates forming compartments, each of which contains a positive-pole electrode, while a negative-pole electrode is in the interval between each compartment and the next. Means for the support of the electrodes and for the production of circulation are provided.

Storage Battery Jar.—T. S. Witherbee, 793,117, June 27. Application filed March 12, 1904.

Details of a battery jar for automobiles.

Galvanic Battery.—D. L. Winters, 795,325, July 25. Application filed Nov. 2, 1904.

When the disintegration of the zinc plate of a primary battery has reached a predetermined stage, a local short-circuit is automatically formed in this individual cell.

Battery Holder for Automobiles.—F. Jackson, 796,517, Aug. 8. Application filed Sept. 27, 1904.

Details of construction of spark-generating battery holders. The batteries can be instantaneously connected in the spark producing circuit without the necessity of wiring them to each other.

Miscellaneous.

Converting Oleic Acid into Stearic Acid.—A. de Hemptinne, 797,112, Aug. 15. Application filed Feb. 7, 1905.

Olein and like compounds are subjected to the influence of electric effluvia in an atmosphere of hydrogen, and stearic acid is formed according to the reaction $C_{18}H_{34}O_2 + H_2 = C_{18}H_{36}O_2$. Polymerides of stearic acid and analogous compounds having melting points more or less in the neighborhood of 69° C. are also formed, whereas olein is liquid at ordinary temperature.

Chemically Modifying Oils.—E. Meusel, 794,373, July 11. Application filed April 30, 1903.

Fatty oils are modified by treating them with nitrates in aqueous solution or other nitrogenous substances, and introducing denitrificating bacteria (such as cheese bacteria) to the mixture. They break up the nitrates to free oxygen, whereby not only does saponification take place, but also a partial oxidation of the glycerin and of the oleic acids. The modified oils are adapted for the manufacture of degras, lacquers, mordants for dyes, etc.

Manufacture of Prussic Acid.—W. Muthmann, Munich, Germany. Assignor to the Roessler & Hasslacher Chemical Co., New York. Patent 792,783, June 20, 1905. Application filed Oct. 17, 1903.

Basing his experiments upon Berthelot's observation that when powerful sparks from a Ruhmkorff coil are passed through a mixture of acetylene and nitrogen, prussic acid is produced and carbon is separated, the inventor found that the yield obtained by Berthelot, namely, one hundredth part of a gram of prussic acid in 90 minutes from 160 cc. of gas mixture, can be increased fifty times. This result is obtained when, instead of the spark from an induction coil, an electric "flame" of high voltage is used, and if the mixture of hydrocarbons and nitrogen, instead of being exploded in a bomb, is blown in the form of a constant current through the flame and the resulting reaction products are constantly removed. This process is stated to have the advantage over that of Berthelot, inasmuch as instead of the costly acetylene, ethylene or hexane, cheaper hydrocarbons can be used, such as methane and its homologues, the hydrocarbons escaping from the earth, illuminating gas, composed of essentially 2 per cent of carbon acid, 4 per cent heavy carburetted hydrogen, 8 per cent carbon monoxide, 49 per cent hydrogen, and 37 per cent methane, gases from coke works, mineral oils and products of their distillation, and also benzene, toluene, naphthalene and other products from the distillation of tar. If these hydrocarbons, or their vapors, be mixed with nitrogen, or with nitrogen and hydrogen, and the mixture be passed through a high-voltage "flame," as much prussic acid is stated to be produced as will render the process suitable for the commercial preparation of prussic acid. Hydrogen is added when heavy hydrocarbons are used which are liable to give a deposit of soot. The prussic acid may be condensed or absorbed by alkalies or saponified. One way of carrying out the process consists in passing for an hour equal parts of nitrogen and illuminating gas through an electric "flame" of 1000 volts and 0.08 amps. Four liters of the gases produce 0.75 grams of prussic acid. Another way consists in passing for one hour through the electric "flame" nitrogen saturated at the normal temperature with vapor of benzen in the proportion of nitrogen 92.1 per cent and benzene (carburetted hydrogen) 7.9 per cent. In this case eight liters of the gases are stated to give 0.48 grams of prussic acid, soot being deposited at the same time, which deposit may be prevented by the simultaneous passing of hydrogen. A similar treatment with petroleum, heated to from 35° to 40° C., is stated to yield in one hour 0.32 grams of prussic acid. Equal, and sometimes better, results are said to be obtained when nitrogen compounds, such as ammonia, are used in place of free nitrogen. It is not settled yet whether nascent hydrogen is evolved in the latter case and produces at once prussic acid, or whether the compounds of nitrogen and carbon produce intermediary products which yield prussic acid by secondary reaction. In one case, *f. e.*, two volumes of ammonia and one volume of acetylene, or ethylene, was passed through an electric "flame" of 1200 volts and 0.07 amps., the volume of the gas being in each case 400 cc., the pressure 72.5 millimeters, and the temperature 18° C. With acetylene the amount of prussic acid obtained is given as 1.23 grams, and with ethylene 0.2 grams. Other hydrocarbons may be used in place of acetylene and ethylene.

Apparatus for Treating Liquids.—L. Dion, New York. Patent 791,457, June 6, 1905. Application filed May 24, 1904.

The apparatus comprises a receptacle for the reception of the liquid, which has a contracted lower body portion and an enlarged upper globular portion. It is provided with a filter in the globular portion. The lower portion is equipped with two groups of interlocking electrodes, which are corrugated in various forms. It is, however, essential that the general directions of the corrugations extend transversely of the interior contracted body portion of the receptacle, so as to form a sinuous course between the electrodes and to insure contact of the liquid under treatment with the electrodes. The construction of the apparatus and the electrodes is described in detail. The apparatus is stated to be especially useful in the recovery of metals and other substances from the waters of mines and minerals and other springs.

SYNOPSIS OF PERIODICAL LITERATURE.

A Summary of Articles Appearing in American and Foreign Periodicals.

Industrial Electrochemistry.

Ozone.—A very concise summary of the various processes for the production and utilization of ozone is given by A. W. Ewell in *Elec. World and Eng.*, Aug. 12. Concerning the experimental use of the Vosmaer system in Philadelphia (our Vol. II., p. 511) the following is said: "The Philadelphia plant is in the heart of the city, and successfully purifies a million gallons a day of the very polluted Schuylkill River water. It is owned by the United Water Improvement Co., which holds the American patents of Vosmaer and Lebret. One of the towers is of glass, and the process of purification may be clearly seen. Dr. Rivas, of the water department of Philadelphia, has recently made a careful test of this plant. He found that the number of bacteria was reduced from about 1,500,000 per c. c. to between 38 and 55, when 10,000 gals. were purified per hour, and that these remaining bacteria were all harmless, disease germs being totally destroyed. This selective action of ozone is very interesting and important. Such action is not found in sand filtration."

Electrolytic Alkali Co.—In *Elec. World and Eng.* of July 15 J. B. C. Kershaw gives an illustrated description of the works of the Electrolytic Alkali Co., at Middlewich, Cheshire, England. Brine is pumped directly from the company's own brine wells into the decomposing cells without the necessity for, or expense of, any intermediate separation in the solid form and re-solution in water. There are two direct-current dynamos, each giving 2000 amps. at 60 volts. The cell contains 250 cells of the Hargreaves-Bird type, two-thirds of which are kept in constant operation. The cell is 10 feet in length, 5 feet in depth and 14 inches in breadth, the narrow chamber enclosed between the two diaphragms being filled with brine and provided with a row of anodes. The two outer compartments of the cell on each side are the cathode chambers. These are supplied with steam and CO_2 gas, and the sodium carbonate which separates on the gauze cathodes is at once washed away as a solution of sodium carbonate by the condensing steam. The carbonated soda liquors from the cells are passed through a refrigerating plant in order to obtain the maximum yield of crystal soda, and the mother liquors are finally thrown away. The decomposing cells are worked in series of 14, two of these being held in reserve for cleaning and repairs. A current of 2000 amps. is the normal one for these cells, the row of 12 thus taking the electrical output of one dynamo. The e. m. f. required per cell varies from 4 to 4.5 volts; and each decomposes about 220 pounds of salt per twenty-four hours. The anodes are made up of rows of rough blocks of gas carbon, strung upon a central support of lead-copper alloy. Cement is used to cover the latter, where it is not covered by, and in close contact with, the gas-carbon. Formerly the gas-carbon blocks were simply bored through and placed upon the copper-lead supporting shaft; but now short-rounded blocks of Acheson graphite are employed to support the gas-carbon. The diaphragms last thirty to forty days in constant work, and the anodes about the same period. The following ampere-hour efficiencies of various processes for electrolysis of brine are given: Castner-Kellner, 91 per cent; Hargreaves-Bird, 80 per cent; Rhodin, 90.2; Aussigbeil ("Glocken" process), 87.5; Acker, 91.6. The watt-hour efficiencies of the same processes are 52.3, 54, 41.4, 40.9 and 54.9 respectively.

Nitric Acid from Air.—The method of S. Eyde and K. Birkeland (which was described in our Vol. II., p. 399, in the St. Louis Congress paper of Edstrom) for making nitric acid from air by electric discharges is described in *Zeit. f. Elektrochemie*, April 21, the description being essentially a translation of the French patent 335,692, of Sept. 18, 1903, which contains some details of the apparatus.

Experimental and Theoretical.

Electrolytic Rectifier.—While the electrolytic rectifier has gradually found commercial applications for various purposes (charging storage batteries, telephony, etc.), the theory of its action is still the subject of investigation and discussion. The results of a recent research are given by S. R. Cook in the May issue of the *Physical Review*. He had found in former experiments that at least from 85 to 95 per cent of the apparent resistance of the aluminium electrolytic rectifier can be accounted for as due to a counter e.m.f. His new experiments show that this counter e.m.f. completely accounts for the high apparent resistance. He offers the following theory: Whenever oxygen is the primary or secondary product of the anion, the aluminium anode is covered with a film of hydroxide, which has a very high resistance, so high, indeed, that it acts as a dielectric to the charge on the ions. This non-conducting film between the metallic aluminium and the electrolyte produces a condenser affect with the negative charge on the anions of the electrolyte. The conditions then that always exist, when a direct current is applied to an electrolytic rectifier with a well-formed aluminium anode, are as follows: When any potential less than the critical potential is applied to the electrodes there is induced an equal and opposite charge in the electrolyte next to the respective electrodes. When the current is broken there exist bound charges in the electrolyte and on the electrodes. These bound charges on the electrodes, which may be assumed to be equal to the bound charges in the electrolyte, are what produce the counter e.m.f. The curve of decay of this counter e.m.f. with time shows the rate at which these charged ions pass through the film and give up their charge to the metallic plate. When the current is reversed, the hydrogen ions pass through the film more readily than the oxygen or other negative ions, forming a gas between the metallic aluminium and the film, thus breaking away the film and exposing the metal to the charged ions. When an alternating current is applied the film acts as a semi-permeable membrane. The hydrogen atoms having less mass and greater velocity than the oxygen or other negative ions pass through much more freely than the more slowly-moving anions. The critical potential value for an electrolytic rectifier depends upon the electrolyte used and the temperature of the cell. When the critical potential has been passed the film crystallizes, and in crystallizing exposes the metallic aluminium. The crystallization continues so long as the potential is kept above the critical value.

Use of Balanced Electrodes.—W. W. Haldane Gee read a paper on this subject on July 3, before the *Faraday Society*, which is here abstracted from advance sheets. The method of ascertaining the weight of metal electrolytically deposited by directly weighing the cathode in the electrolyte instead of removing the cathode and weighing it—after washing and drying—is considered. The real weight (M) is obtained from the apparent weight (m) by the formula:

$$M = \frac{m\,D}{D - d}$$

where D is the density of the deposited metal, and d that of the electrolyte. It is shown that since D occurs both in the numerator and denominator of the expression that for most purposes it is not necessary to know the value of D with great accuracy. In the case of copper the value of D that may be used is 8.9. An ordinary physical balance is used, and a cylindrical cathode suspended by a thin wire passing through a hole in the bottom of the balance case is counterpoised in the electrolyte. The anode also is circular and surrounds the cathode. Electrical connection is made to the cathode by a side wire whilst receiving a deposit, but is removed when the apparent increase of weight has to be ascertained. The method has been found very convenient in testing commercial types of ammeters. An accuracy of about 1 per cent can be generally obtained, and with a little care and by increasing the amount of deposit the accuracy can be brought under 0.5 per cent. In some experiments both sides of the balance may be used simultaneously when the electrochemical equivalents of metals in different combinations may be compared and current efficiencies ascertained. A number of experiments have been made with a mercury cathode and anode, the electrolyte being solutions of mercurous nitrate, or mercuric chloride mixed with potassium cyanide. The mercury cathode is enclosed within a shallow dish suspended in the electrolyte by a platinum wire. The anode is a layer of mercury at the bottom of the containing vessel. In the case of mercurous nitrate it is shown that the current efficiency as a monad is affected by the presence of mercuric and basic compounds. For good results the mercurous nitrate solution must be carefully prepared and a strong solution used. Modifications of the method are described that may be used for the purpose of finding the density of electrodeposited metals, and for studying the relative loss and gain at anode and cathode. Instead of an ordinary balance a spring balance may be substituted. The author also shows how a Nicholson's hydrometer may be used both as a cathode and as a means of finding the amount of deposit.

It is concluded that the method will be useful in the laboratory and test room as a means of rapidly arriving at results with sufficient accuracy for many commercial purposes. In the discussion, F. M. Perkin said that the drawback to the use of mercury was that only small currents could be measured therewith. What was particularly wanted was an instrument that would indicate continuously; for example, the cathode might be suspended in a helix, to which a pointer was attached, as in the Ayrton-Perry meters.

The Position of the Alkali and the Earth Alkali Metals in the Voltage Series at High Temperatures.—The relative position of the different alkali and earth alkali metals in the voltage series is of the greatest importance for the electrolytic production of these metals. There are a number of reasons why it would be preferable to deposit the metals at as low a temperature as possible. On the other hand, the molten electrolyte should have as high a fluidity as possible, since otherwise the metal deposited on the cathode tends to grow in trees over to the anode, while the surrounding electrolyte tends to solidify. The reason is that the current which, by its Joulean effect, shall maintain the electrolyte fluid will then pass no longer through the electrolyte but through the metallic tree extending from the cathode. Destruction of the apparatus, evaporation of the metal and the occurrence of undesired reactions (for instance, the solution of the deposited metal in the electrolyte) make, on the other hand, the use of a low temperature very desirable. For these reasons mixed electrolytes are preferable to pure salts, since the former have generally a lower melting point. Since the more negative metal is first deposited from a mixed solution it is of importance to know the relative position of the different metals in the voltage series at different temperatures, and it is not superfluous to emphasize that if the voltage series has been determined at a certain temperature the series of the metals may be different at another temperature. This is very well brought out in an article by H. Danneel and L. Sackem in *Zeit. f. Elektrochemie,* April 7. It appears certain that at ordinary temperature Na is more positive than Ca (metallic sodium sets hydrogen free from alcohol in spite of the very small hydrogen ion concentration; calcium does not act in this way. Further, the heat of formation of the sodium salts is higher than that of the calcium salts). If a molten bath of $CaCl_2 + NaCl$ is electrolyzed at about 800° C., Na is deposited, and not Ca, while from a mixture of $CaCl_2 + KCl$ we get Ca containing K. This indicates that in the voltage series Ca is at high temperatures between Na and K, while at lower temperatures Ca is more negative than Na, so that the relative position of the Na and Ca must reverse at a certain temperature. The authors describe several experiments which strengthen this view. Na is unable to precipitate Ca from its molten salts at temperatures above 800° C. At medium red-heat an Na-Ca alloy is formed, so that at this temperature the free energy of formation of the Na and Ca salts is not very different. It seems impossible to make metallic potassium from the cheaper metallic calcium. Finally, the authors mentioned that they did not succeed in precipitating metallic strontium from its chloride by means of potassium.

Electrolytic Manufacture of Fine Wires.—In *Comptes Rendus,* May 29 (abstracted in the *London Electrician* of June 23), Abraham describes experiments in which he prepared very fine and uniform wires by the partial anodic decomposition of a wire in an electrolytic tank. The diameter of the wire is measured by determining its resistance, and for this purpose it is attached to metallic rods at both ends. It hangs in the liquid, which does not touch the rods. The solution must be very dilute; the author used distilled water containing a few thousandths of its weight of copper sulphate for copper wire or silver nitrate for silver wire. The action must be slow, in order to allow time for the products of electrolysis to diffuse through the electrolyte. He used a current density of about 10 milliamperes per square cm. of surface of the wire, and this must be reduced as the wire becomes thinner. The solution of the copper wire regulates itself, since the resistance is highest at the thinnest portions of the wire.

Preparation of Diamonds.—A paper concerning the preparation of diamonds, by H. Moissan, is given in *Comptes Rendus,* January 30, and abstracted in *Zeit. f. Elektrochemie,* May 19. A meteorite containing diamonds showed the presence of iron sulphide and silicon. The author, therefore, made experiments to find whether the presence of the two elements, S and Si, is advantageous to the formation of diamonds. The method was essentially the same as that used by him before. The addition of sulphur resulted in the formation of little diamonds of the same size as those obtained without sulphur, but it seemed that the number of diamonds formed was greater.

Artificial Diamonds.—A paper, by H. W. Fischer on experiments made by him on the artificial production of diamonds in the electric furnace by heating to a temperature of 2500° or 2800° C., or even more, and then suddenly releasing the matrix into a cooling apparatus, is given in *Elec. World and Eng.,* of May 6. The construction of the electric furnace is illustrated and several micrographs are reproduced.

Electrolytic Oxidation of Ammonia to Nitrite.—In a paper, printed in *Chem. Soc. Trans.,* Vol. 87, p. 198, and abstracted in *Jour. Soc'y Chem. Ind.,* March 31, H. D. Law gives an account of an investigation of the electrolytic oxidation of formaldehyde, acetaldehyde, propaldehyde and iso-butaldehyde in sulphuric acid solution with platinum electrodes. The chief products were the corresponding acids, together with smaller quantities of carbon dioxide and carbon monoxide.

Electrolytic Oxidation of Ammonia to Nitrite.—In a paper, published in *Berichte,* 1905, Vol. 38, p. 778, and abstracted in *Jour. Soc'y Chem. Ind.,* March 31, E. Mueller and F. Spitzer describe experiments in which they repeated former ones of Traube and Biltz, and obtained different results. They were able to get higher yields of nitrite; under any circumstances, however, the cost of nitrite thus prepared will probably be too great to allow of the process being used industrially. The gases evolved during electrolysis consisted almost entirely of nitrogen, with a little oxygen when the nitrate concentration was high; no nitrogen oxides were observed. In neutral or very slightly alkaline solutions a certain amount of nitrogen was evolved, as the result of non-electrolytic decomposition of ammonium nitrite. W. Traube replies in *Berichte*, Vol. 38, p. 828. An ammoniacal copper solution containing fixed alkali absorbs atmospheric oxygen much more rapidly than one without it. By increasing the initial concentrations of fixed alkali and of ammonia, more concentrated solutions of nitrite can be obtained before the formation of nitrate from the nitrite begins to occur. By the electrolysis of a solution of ammonia and fixed alkali in presence of copper, containing, to start with, 11 per cent of sodium nitrite, the nitrite concentration could be increased to 17 per cent.

Electrolytic Oxidation of Organic Compounds.—A paper presented by H. D. Low and F. Mollwo Perkin, on July 3, before the *Faraday Society*, has the title: "The Electrolytic Oxidation of Hydrocarbons of the Benzene Series"; part II., "Ethyl Benzene, Cumene and Cymene." The authors have already shown that when hydrocarbons of the benzene series are electrolytically oxidized, that the aldehyde is in all cases the main product. This, however, is only the case when an appropriate solvent is employed; the only one which has given satisfactory results is acetone. In continuation of the research the authors now show that when ethyl benzene is oxidized the chief product is the secondary alcohol, small quantities of the primary alcohol also being produced. Acetophenone, the second oxidation product, is not obtained, but benzaldehyde is produced when the oxidation is sufficiently prolonged. With cumene the primary alcohol and the corresponding aldehyde are produced. The isopropyl group in cumene, however, shows itself relatively stable towards electrolytic oxidation. Cymene gives chiefly cumic alcohol and cunic aldehyde, the methyl group in the paraposition to the isopropyl group being attached in preference to the latter. In discussing the results the authors consider that the hydroxyl ion or discharge is the oxidizing agent; in other words, the oxidation is mainly a primary reaction. The work is being continued.

Electrochemical Reactions in Organic Chemistry.—The conclusion of J. Moeller's serial on this subject is found in *Elektrochem. Zeit.*, June. The chapter on the electrochemical reduction of nitrocomopnds is concluded. The author then discusses electrochemical substitution processes, and finally deals with the synthesis of organic compounds by silent discharges and with the use of electric arcs, or of incandescent filaments on gases and vapors of organic liquids.

Organic Electrochemistry.—Those interested in the applications of electrochemical methods to organic chemistry may be referred to a useful review of recent papers in this field, published in *Zeit. f. Elektrochemie*, May 5, where quite a number of papers are abstracted under the following headings: Electrolysis of Organic Compounds, Electrochemical Reductions, Electrochemical Oxidations, Other Applications of the Electric Current in the Field of Organic Chemistry.

Electrolytic Formation of Chlorate.—With respect to this subject experiments of H. Sirk, *Zeit. f. Elektrochemie*, April 28, are interesting. They show that the reaction

$$ClO_3' + 2H' + 2Cl' = OCl' + 2HOCl$$

is accelerated by platinum as contact substance; platinized platinum is specially effective. This fact may have importance for the commercial electrolytic manufacture of chlorate with the use of platinum electrodes.

PLATING.

Electrolytic Zinc Plating Versus Hot Galvanizing.—With respect to the article of C. F. Burgess in our January issue (page 17), two articles by J. Szirmay in *Zeit. f. Elektrochemie*, June 2, are interesting. His results are the same as those of Burgess, that is, that electrolytic zinc plating is preferable to hot galvanizing for general purposes. The present author first determined the resistance against rusting of zinc-covered iron wires. His experiments were made under a glass globe containing air, into which were introduced, under pressure, sulphur dioxide and carbonic acid. The analysis of the air under his globe gave in one case 12 per cent SO_2 and 15 per cent CO_2. This represented an intendedly strong exaggeration of the conditions in the neighborhood of chimneys, locomotives, etc. Moreover, he heated and cooled the air alternately between the limits of 6° and 45° C. It was found that under such conditions wire plated with zinc by electrolysis resists to a far greater degree the mechanical deformations as well as atmospheric influences than any hot-galvanized wire, as a result of the better adherence and the greater purity of the electrolytic deposit. Similar results were obtained with other iron and steel ware coated with zinc.

Electroplating Upon Aluminium.—In a note in the June issue of the *Brass World*, the necessity of reducing the oxide film from an aluminium surface before plating is pointed out. It is recommended to sand-blast the surface.

Metal Finish.—An illustrated article by A. Fournier in the June issue of the *Brass World* deals with the production and application of the gun-metal finish. Another article deals with the production of a cheap imitation French-gray finish.

STORAGE BATTERIES.

Effect of Ammonia on Storage Battery Electrolytes.—A paper presented by R. W. Vicarey to the Faraday Society, on July 3, is intended by the author to be a *résumé* of observations and experiments referring to the deleterious effect of nitrogen compounds (especially ammonia) upon the durability, efficiency and behavior of storage batteries. It would appear that ammonia must be considered as a dangerous impurity, the total absence of which, however, cannot be absolutely insisted upon. The presence of ammonia in the surroundings or neighborhood of the cell must be carefully guarded against, and the percentage actually present in the cell must be kept as low as possible by the use of waters and acids ot the purest quality.

The author points out that above certain limits of ammonia little difference is observed in the behavior of cells, but that below that limit much difference occurs. Thus experiments upon cells containing proportions of ammonia varying from 10 per cent to 0.1 per cent (7040 to 70.4 grains per gallon), the loss in capacity was 22 to 31 per cent immediately after the addition of the ammonia, whilst cells containing less than 0.1 per cent did not drop in capacity immediately, but varied intermittently from 10 to 15 per cent, finally settling at the end of eight months to a permanent loss of 15 to 22 per cent.

From the data given it appears that ammonia is very slow in its action before it makes any appreciable or apparent loss in the capacity of the cells, the average time being about twelve months, dating from the initial charge. The effects of ammonia in its various combinations appear principally in two distinct forms: (1) it causes and excessive disintegration or shedding of active material at the positive (PbO_2) plate; (2) it becomes deposited upon the negative (spongy) plate, and finally closes up the pores of the active material, thereby causing a decrease in the ampere-hour capacity or efficiency, varying from 10 to 60 per cent in about twelve months. The use of water taken from the steam boiler and termed "distilled" is mentioned as a dangerous practice that is common in central stations, etc., etc. The position of the battery and its surroundings are also touched upon. Finally, it is proposed that a series of experiments be carried out upon some cells in an attenuated atmosphere away from all manufacturing districts.

REFRACTORY MATERIALS.

Furnace Linings.—In our April issue we published a *Faraday Society* paper of E. Kilburn Scott, on refractory materials for furnace linings. In the following we give an account of the discussion which followed. H. N. Dains gave some comparative analyses showing the differences between Indian, Grecian, Styrian and American magnesites. The great refractoriness of the first was due to the small amount of impurities it contained. Magnesite shrunk in the electric furnace certainly gave the most satisfactory results, and in support of this contention he quoted several striking instances. The principle use of burnt magnesite was for lining open-hearth steel furnaces, converters, rotary kilns for cement manufacture, furnace hearts, crucibles and refractory bricks. The power-consumption given by the author was incorrect, because the experiment quoted was carried out on *lightly* calcined magnesia that had been lying about and had taken up about 20 per cent of CO_2 and water. F. Gelstharp said that if a material could be found to withstand the action of tri-silicate of soda and lime at high temperatures, and at the same time would not oxidize, it would be a great boon to glassmakers. None of the materials mentioned by the author were suitable for the port-holes of tank glass furnaces. W. Murray Morrison hoped the author would furnish them with further more exact data regarding the temperatures the various refractory materials could stand, their mechanical properties and their heat conductivities. Tar was the best binder to use for high temperatures, and the increase of conductivity it caused was unimportant; but something that would help the refractoriness of the material would be preferable. L. Gaster said that a siloxicon wash was insufficient for lining cement kilns, for example, hard bricks of the material were necessary. R. Cremer found, after much experience, that silicate of soda was a perfect binder for carborundum. He considered carborundum as good a refractory material as any that had been described, and its mechanical properties were excellent. R. S. Hutton said that if magnesite were calcined in an electric furnace the CO_2 given off would rapidly consume the electrodes. But there was another "shrinking" in the sense of increasing the specific gravity, and to this true shrinkage the valuable properties of magnesite (as of quartz) were due; this was best effected by fusion in an arc furnace, and Moissan had obtained a magnesite of specific gravity 3.654 in this way. In the author's experiments the magnesite was never more than in a semi-plastic condition; magnesia was easily melted in the electric furnace at something above 2000° C. Electric furnaces, contrary to what had been said, did not require such highly refractory linings as ordinary furnaces, because the material itself should be the effective lining.

Iron and Steel.

Dry-Air Blast.—Mr. James Gayley has followed his famous first dry-blast paper with another of a supplementary nature, additional records being supplied of the working of the Isabella furnaces from November, 1904, to March, 1905. The paper is published in the July issue of the *Bi-monthly Bulletin* of the American Institute Mining Engineers. The results are best expressed in the following summarized table:

MONTH.	Average Grains of Moisture in Atmosphere Normal Blast	Average Grains of Moisture in Dry-Air Blast.	Average Daily Production.		Average Coke Consumption.		Blowing Engine. Revs. per Minute.		Average Temperature.	
			Dry Air. Tons.	Natural Air. Tons.	Dry Air. Lbs.	Natural Air. Lbs.	Dry Air.	Natural Air.	Dry Air.	Natural Air.
November	1.99	1.02	447	386	1,816	2,279	96	111	854	750
December	1.44	0.93	455	*400	1,823	*2,309	96	*111	877	*785
January	1.245	0.635	†430	†412	†1,945	†2,346	†96	†111	†836	†744
February	1.18	0.60	412	424	1,815	2,248	96	111	800	784
March	2.25	0.95	405	411	1,837	2,274	96	111	784	850

* Average for normal blast for Dec. 1–22 only. † Average for two furnaces, each working for part of month on dry, and part on natural air, owing to a change over between furnace No. 1 and Furnace No. 3.

The above figures require very little explanation, save that the reason for the lower output for the dry-air furnace in February is due to several stoppages of the furnace using dry air on account of breakdowns of slag machine and breakouts of iron at the hearth. Notwithstanding the low content of moisture in the normal blast, the furnace equipped with the dry blast consumed 433 pounds less coke per ton of iron with a nearly equal output. As the summer season approaches the product of the furnace using normal blast will, Mr. Gayley asserts, steadily decrease and the fuel increase, while the work of the furnace using dry air will continue practically uniform.

Copper.

The Copper Queen Smelter.—A short description of the reduction works of the Copper Queen Consolidated Mining Co at Douglas, Ariz., appears in the *Engineering and Mining Journal* of Aug. 5. The method of bedding down the charges for the blast furnaces is stated to be a radical innovation. The ore from the Copper Queen mines is received in side-dump steel cars, which discharge into stone-lined pits. The ores are bedded in layers and the flux is added in the proper amount so as to make up the smelting mixture. This mixture is loaded by steam shovels into charge cars, trains of which are then run up an incline direct to the furnaces. It is stated that after the operation of this system of charging had been perfected there was no difficulty in making the furnaces run so as to yield slags of the calculated composition. The cars which take the ores from the bedding trenches to the furnaces are of rolling side-dump type, and discharge directly into the furnaces. Three 60-foot electric cranes of 60-ton capacity transfer the matte from the furnaces to the converter and the converter shells from the relining floor to the converter stands. The furnace and converter building are 650 x 150 feet, and contain eight blast furnaces and seven stands of converters of the trough type. A relining plant is located at each end, with stands for relining and drying thirty shells.

Treatment of Low-Grade Copper Ores.—In this country, and especially in the East, occur great deposits of sandstone of various geological ages which contain a certain amount of copper, either as metal or a sulphide or carbonate. Owing to their poor grade these deposits are not utilized, as the present methods for dealing with such low-grade ores, that is, the leaching and precipitation of the copper, either by chemical methods or by electrolysis, is too expensive for the metal values contained in them. In a paper before the Franklin Institute, published in the Aug., 1905, issue of the *Journal* of the Institute, Dr. N. S. Keith proposes a new method for the metallurgical treatment of these ores. It is based essentially upon a reducing treatment of them in a shaft furnace with the production of small globules of metal to be afterwards separated by concentration. As the gangue of these ores is preeminently selicious, inasmuch as they contain up to 90 per cent of silica, this operation can be performed without changing the silica to any extent, as there is no fluxing material of any account present. He proposes to smelt the ores in a vertical shaft furnace about 25 feet high. It is to be constructed of red brick with a firebrick lining. The powdered rock, powdered coal and producer gas, or a like fuel, are introduced from the top. The bottom of the interior of the shaft slants at an angle of about 45° and extends into a dust-collecting chamber, which in its turn connects with another vertical shaft. This latter has also an inclined bottom, which, however, dips towards the other incline. The interior of this latter shaft is built up with pieces of coke or small stones, over which water flows, distributed by a sprinkler near the top. The draft is downward in the first shaft and upwards in the second and is produced by an exhaust fan. The treatment of the rock as it comes from the mine consists in crushing and pulverizing it to a sufficiently fine mesh, so that the particles of rock and mineral are no longer coherent. This comminution may be as fine as 60-mesh or finer or coarser as required. During the comminution about 3 per cent of the weight of the rock of finely pulverized coal is added and intimately mixed with the rock. This mixture is continually fed into the top of the furnace, and the metalliferous particles are reduced in the lower part of the furnace to metallic globules. The heat does not fuse the silica or the lime, and the result of the smelting operation is, therefore, a sand carrying minute globular particles of copper, gold and silver. The dust and condensible gases are collected on the wetted surfaces of the coke in the condensing shaft and carried to the bottom. There the water also meets the sand and carries it in a constant stream out of the furnace upon concentrators, where the metallic particles are separated out. The latter are then dried and melted and cast into merchantable shapes. The process is claimed to be automatic and cheap, under the economic conditions of the above mentioned localities, not exceeding a cost of $1 per ton on a scale of treatment of 100 to 200 tons per day of twenty-four hours. The gas drawn from the furnace is a combustible one and may be used for heating purposes.

GOLD AND SILVER.

Tavener's Process for Treating the Cyanide Clean Up.—It has always been a somewhat debated question whether the losses by volatilization of gold in the gases drawn from a pan furnace were of sufficient amount to be taken into consideration. In order to determine this point the metallurgical department of Messrs. H. Eckstein & Co., at the Ferreira Gold Mine in the Transvaal, determined to erect a bag-house in order to filter the gases drawn from the pan furnace. The Tavener process at these works was worked for six months, and the whole length of the pan furnace culvert was cleaned up. It yielded 880 pounds of flue dust, containing only 1.71 ounces of fine gold, while during this period 16,342 ounces of fine gold had been reduced from gold zinc slime in the furnace and had been tapped in the form of lead bullion. The arrangement for collecting the fumes, described by H. Rusden in the *Journal* of the Chemical, Metallurgical and Mining Society of South Africa, April, 1905, and illustrated in Fig. 1, was as follows The furnace gases passed a baffle chamber *A* and were aspirated through the tubes of an old boiler *B*, around which water was circulated to cool them. An aspirating fan *C*, driven by the motor *D*, forced the cooled gases into a brick chamber, which was surmounted by the flannel bags *E*. Experiments were tried on the test furnace as well as on the pan furnace. In the test furnace experiments nine bags were fixed in the bag house, and the fan was run at 1750 revolutions per minute. The dust collected in this case assayed 5 dwt. fine gold and 42 dwt. silver per ton. With the same conditions as in this furnace the pan furnace could not be made to work; a larger fan had to be installed and an additional water tower *G* had to be erected for passing the gases out through a heavy spray of water. The cyanide works clean-up was then smelted in three charges, twenty-four hours being consumed for smelting. The estimate of fine gold contained in the lead bullion in this run was 1,229.97 ounces. The weight of the dust caught in the bag-house amounted to 70 pounds, while that caught in the settling tank from the water tower was only 5 pounds. The assayed value of the dust was 38 dwt. fine gold per ton. It was, therefore, proven that practically no gold was carried over from the test furnace and very little from the pan furnace, if the charging is done in a careful manner.

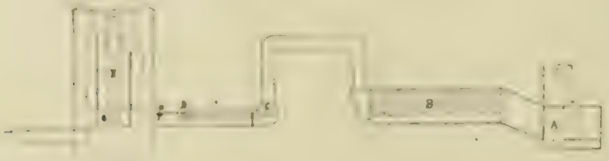


FIG. 1.—PLANT FOR COLLECTING FUMES.

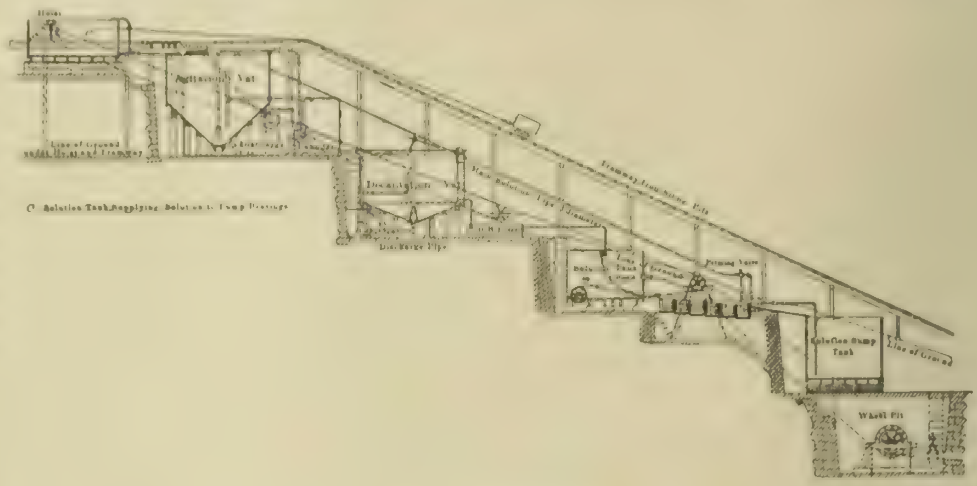


FIG. 2.—SLIME PLANT.

Cyaniding Silver-Gold Ores.—The cyaniding of unroasted silver lead ores has only been commercially successful within the last few years, and a paper on the subject of cyaniding silver-gold ores at the Palmarejo Mine, read before the American Institute of Mining Engineers by T. H. Oxnam, and published in the *Bi-Monthly Bulletin* of the Institute in July, 1905, is therefore of considerable interest. The author goes quite into details in the discussion of the cyaniding process as used on the ores furnished by the Palmarejo mines, the main value of which is in silver. The major portion of the latter occurs in the form of argentite, with a small amount of stephanite, chlorobromide and native silver. He first describes the old method of crushing, roasting and pan amalgamation, which did not prove successful in this case, as the losses were too heavy. The present system of milling consists in crushing the ore by crushers and stamps, the pulp passing over ten Wilfley concentrators, which, during the year ending July 1 1904, removed about 0.76 per cent by weight of the ore and formed concentrates which contained 17.98 per cent of the silver values and 18.28 per cent of the gold values of the ore. The sands are separately treated in a cyanide plant, which consists of twelve cyanide leaching vats, each of which is 30 feet in diameter and 4.5 feet deep, and is charged with an equivalent of 100 tons of dry sand. While being trammed to the leaching vats slaked lime is added, usually in the proportion of 4 to 5 pounds of lime per ton. The treatment of the vat is begun with weak solution carrying about 0.25 to 0.30 per cent of cyanide, in which the material is allowed to soak; weak solution being added from the top as rapidly as permitted by the leaching rate of the charge until a total of 100 to 130 tons has been applied. From 60 to 70 tons of strong solution, carrying between 0.75 and 0.80 per cent KCY, is then added and percolates through the charge in about forty-eight hours. A weak solution is then run through again as rapidly as possible, followed by a final wash of water. A thorough oxidizing of the charge is very essential for the treatment of those ores, and therefore as many charges as possible are transferred from one bath to another during the process. The solution from the leaching vats finally reaches the zinc boxes, of which there are six, five being used for the weak solution and one for the strong solution. During the year 1904, 91,793 tons of weak and 32,251 tons of strong solution passed through the boxes, an average of 251 tons of weak and 61 tons of strong solution every twenty-four hours. The five weak solution boxes are 2 feet wide and 18 feet long over all; the boxes contain eight compartments, each of which has an available zinc capacity of 24 x 24 x 18 inches. The strong solution zinc box consists of seven individual round boxes placed in series, each being 28 inches in diameter and 24 inches deep, with an available zinc capacity of approximately 5 cubic feet. The average assay values per ton of solution entering the zinc boxes are about $1.00 of gold and 2.25 ounces of silver for the weak solution, and $1.24 of gold and 3.5 ounces of silver for the strong solution. The precipitation of gold, as a rule, is practically perfect and that of the silver averages about 95 per cent. During 1904 the assay returns of the cyanide precipitate have averaged slightly over 20,000 ounces of silver and approximately $8,000 of gold per short ton, equaling about 68.67 per cent of silver 1.33 per cent of gold. The consumption per ton of sands cyanided was sodium cyanide, 2.95 pounds; zinc, 0.96 pounds, and lime 4.33 pounds. The treatment of slimes is performed in an entirely separate plant, a section of which is given in Fig. 2. The treatment consists briefly in giving the slimes about two days agitation in the agitating vats, with from two to three times their weight of cyanide solution, followed by another two, or sometimes three, days treatment in the decantation vats, during which latter portion of the treatment the charge, after having been sufficiently agitated with the addition of slacked lime, is allowed to settle as much as practicable, and the supernatant clear liquor is decanted and passed through the zinc boxes. This operation is repeated, there being ordinarily three or four decantations.

The agitation as well as the decantation vats are run by centrifugal pumps, each vat having its own individual pump. The precipitates recovered are of a lower grade than those obtained in the sand plant, because a certain quantity of suspended slimes is carried over into the zinc boxes and settles there, thus lowering the grade of precipitates. The highest grade of precipitates yet recovered from the slime plant assayed approximately $6,800 of gold and 17,300 ounces of silver per ton. The average extraction of the silver for the last three months has been 51 per cent.

RECENT METALLURGICAL PATENTS.

Iron and Steel.

H. H. Campbell (795,193, July 18) treats chromiferous iron for the extraction of the chromium as follows: Chromiferous iron ore is melted in a blast furnace and the molten chromiferous iron is charged, together with lime, into a basic Bessemer converter. Silicon and carbon are removed by being oxidized by the air blast that is driven through the charge, and this blowing process is further prolonged until the chromium contained in the iron is oxidized and becomes part of the slag. In order to prevent its return from the slag to the metal it is necessary to separate the metal and slag before recarburization, meaning by "recarburization" any process for removing oxygen from the blown metal or incorporating with it manganese, carbon or other elements usually deemed necessary in steel after decarburization of the iron. If the dechromized metal contains considerable oxygen it is charged into a second acid-lined converter together with a charge of iron free from chromium but with a higher carbon content (usually low phosphorus quality of iron commonly used in the Bessemer process). The inventor states that the raw iron which he has treated by this process had a content of chromium varying from 1 to 5 per cent. In one charge the iron contained 2½ per cent chromium. After removal from the basic converter the chromium content had been reduced to one-tenth per cent. This same decarburized product was then transferred to an acid-lined converter and 25 per cent of melted iron free from chromium was added. The mixture was then blown and recarburized. The finished steel contained only 0.08 per cent chromium.

Pyritic cinder, obtained from the manufacture of sulphuric acid from iron pyrites, contains such an amount of sulphur as to be considered unsuitable for making iron. T. C. King (794,673, July 11) treats the pyritic cinder with a binding material like tar or pitch, which has a two-fold action: first, to regulate the size of the nodules produced; second, to combine with the impurities (sulphur, etc.) and form volatile products. This binding material is cohesive at lower temperatures and volatilizes at higher temperatures. In the lower zones of heat in the apparatus or kiln the materials cohere by the adhesiveness of the pitch until by agitation nodules are formed, which are carried into and through the zones of incipient fusion. Here they are rendered permanently cohesive, while the impurities form volatile products and escape. The addition of 1 per cent of pitch to 99 per cent of an iron oxide, analyzing 67 per cent iron and 1 per cent silica, produces nodules about the size of a goose egg, which is best adapted to open-hearth practice, while the addition of ½ per cent of pitch to 99½ per cent of the same iron ore produces nodules the size of a partridge egg, which is best adapted to blast furnace practice.

T. Rouse and H. Cohn (797,150, Aug. 15) make briquets of powdered ferruginous matter, iron waste, etc., by using the following binding solution. One part by weight of alum is dissolved in 200 by weight of water, and 2 measures of water glass are added to every 100 measures of such alum solution, and about 1 part by weight of this solution is absorbed in the mixing process by 20 parts by weight of the powdered concentrated ferruginous matter.

Zinc.

W. Stewart (794,198, July 11) patents the following process of treating complex or refractory ores containing zinc together with gold and silver. The ore is pulverized and mixed with bisulphate of soda and with common salt, and this mixture is brought in a furnace to red heat and maintained thereat with a limited supply of air for about from fifteen minutes to three hours. The zinc is thereby brought into the condition of salts soluble in water or in slightly acidulated water. This is, of course, followed by leaching and by precipitation of the zinc salts.

E. C. Hegeler (794,799, July 18) patents details of construction of zinc smelting furnaces with a long flue-like horizontal retort chamber. The main point of his invention is that the resistance of the movement of the gases through this chamber can be reduced and the uniformity of the heating of the retorts can be improved by providing open space under the roof of the chamber and under the lowermost tier of retorts. For this purpose the retort chamber is provided with a series of recesses above and below the retorts, the recesses being staggered in the bottom and top rather than arranged opposite one another.

Gold.

E. C. Brown (791,872, June 6) patents an agitator for the mixing of sands and slimes with cyanide solution, without any revolving arms or the like, by the single agent of compressed air. The agitator, as shown in Fig. 1, is a cylindrical vertical tank with a conical bottom. The tank is filled with sands and slimes, which are allowed to settle. After the withdrawal of the top liquid sufficient space is left for the subsequent solution and the washes required. Cyanide solution is then introduced to the tank through the pipe *C* supplying the circular pipe *G* (which is indicated in the cross-section figure by two small circles only). From this circular pipe *G* the cyanide solution discharges through jets G^1. The solution is allowed to flow into the tank until it has risen nearly to the top of the central pipe *B*. Compressed air is then admitted to this central pipe *B* through the tube *F* in the bottom, with the result that a mixture of sand, solution and air rises through and is discharged through the top of the central pipe *B*. The solution, or liquid, issuing from the circular pipe *G* through the small pipes, spouts or nozzles or other devices G^1 impinges in the sides or walls of the lower conical part of the tank and causes the body of the sand or like material in the tank to settle down, the whole charge becoming a homogeneous mixture of solution and sand, as it is drawn into the bottom of the central tube or pipe, and delivered at the top of the tank so rapidly that no opportunity to settle is afforded. As soon as the surface of the mixture in the tank has risen to the top of the central pipe *B* the solution is shut off from the circular pipe *G*, and the circulation of the mixture will go on as long as required by the agency of the compressed air or gas.

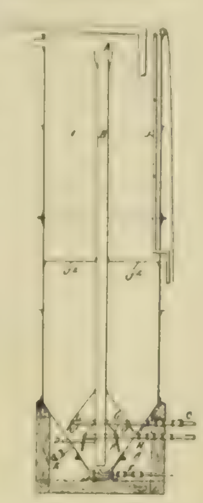


FIG. 1.—AGITATOR.

A. E. Johnson (791,424, May 30) constructs a barrel filter of lead. Lead has sufficient strength and weight to render unnecessary an additional bed or support, and it may be perforated so as to perform the filtering function itself. For this purpose the top of the filter is provided on its under surface with depressions, whereby the metal is made sufficiently thin to be perforated. The perforations are made by punching from the inside and with such a tool as to form conical orifices, smallest at the top and increasing downwardly to prevent clogging.

E. L. Godbe (793,720, July 4) patents the filter shown in Fig. 2 for separating slimes from metal-bearing solutions; 3 is a revolving drum covered with a filtering medium; the air within the drum is exhausted by means of a pipe going nearly down to the bottom of the drum (not shown in the figure) and connected to a suction pump. During the revolution of the drum the solution is sucked into the drum and the slimes adhere to the filtering surface. They are then first sprayed with a solution of cyanide through the nozzle 12, and then with water through the nozzle 13. The action of both liquids which was through the filter is to displace the metal-bearing solution from the slimes. The slimes are scraped off by means of the brush 15 and the scraper 18 and fall into the hopper 16.

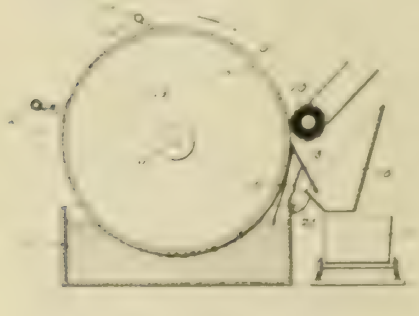


FIG. 2.—FILTER.

T. D. Jones (795,744, July 25) patents details of construction of a revolving barrel for the chlorination process. The barrel is provided at one extremity with a filter, and at its opposite extremity with a valve-controlled inlet for subjecting the contents of the barrel to pressure. For charging the barrel and for filtering the barrel is placed in upright position.

R. A. Kerr (789,325, May 9) patents details of construction of a device for treating slimes and consisting of a filtering tank and agitator. The latter consists of blades revolving on a shaft within the slimes.

BOOK REVIEWS.

Le Four Electrique, son origine, ses transformations et ses applications. By Adolphe Minet. [Libraire Scientifique, A. Hermann, Paris.] Part I. Price, $0.80.

A really satisfactory book on the electric furnace, to be used for reference purposes, is certainly needed, but so far has not appeared. Attempts have been made to produce such a book, but, unfortunately, they have not been successful, their failure being due to the lack of practical knowledge of electric furnaces on the part of the author. An enormous amount of material has been published in the shape of patents, scientific papers, and so forth, in relation to electric furnace work, and it requires no little knowledge on the part of the would-be compiler to discriminate between what is valuable and what is worthless.

Judging by the first of the proposed five numbers of Mr. Minet's "Le Four Electrique" we shall have a valuable reference book when the work is complete. The task Mr. Minet has set himself is by no means an easy one, as may be supposed, but he seems to have approached it in the proper manner and to have taken great care in the preparation of the first part.

This number contains much interesting historical matter, a classification of electric furnaces, a description of the laboratory furnaces invented and used from the year 1808 to 1886, while the remainder is occupied with various theoretical considerations, such as physical units, laws of electrolysis, etc.

In spite of the care that has evidently been exercised in preparing the work, some errors have crept in. Thus, in the classification of electric furnaces on page 11 the Ruthenburg furnace should not appear in the class "La résistance n'est pas en contact avec les matières traitées," the sub-class, "Résistance solide," and the specification "Four d'induction," but should appear in the class "La résistance est constituée par les matières traitées." In this class also should appear Acheson's graphite furnace and the original Cowles furnace. As an example of omission we find that Colby's induction furnace is not mentioned, much less described.

However, in spite of some minor errors, Mr. Minet's work is welcome, and we look forward with much interest to the next number, which will discuss the industrial electric furnaces that have been developed since 1886.

Les Fours Electriques et leurs applications industrielles. By Jean Escard. [Vve. Ch. Dunod, Paris.] Price, fcs. 18.

This work is simply in the nature of a collection of accounts of electric furnaces taken from various sources, no great discrimination being shown in the compilation. We are informed, for example, that the Conley furnace is working on a large scale for the manufacture of steel in the United States, though to the best of our knowledge the Conley furnaces have not yet been worked commercially anywhere. Errors, of which this is an example, are likely to enter into a book largely made up of information picked up from various sources, unless the collector goes to considerable trouble in verifying the material he obtains.

We conclude that considerable space is devoted to the electrolytic manufacture of aluminium, because the bath is kept in the fused state by means of the electric current. If this is so it does not seem logical to omit reference to Castner's process for manufacturing sodium and Acker's process of making caustic soda and chlorine.

A great deal of the book is taken up with a description of Moissan's work with the electric furnace. For our part we much prefer studying work of that kind in the original when, as in this case, it can easily be obtained.

Some errors have been overlooked in reading the proofs, for we are told that the Pittsburgh Reduction Co. uses 1500 hp. in manufacturing aluminium in America. Probably this is a misprint for 15,000.

There are many illustrations in the book, some being very poor, as that showing the general view of Keller's furnace.

The Electric Furnace. By Henri Moissan, translated by A. T. de Mouilpied. [Edward Arnold, London.] Price, $2.75.

In spite of the great interest of Prof. Moissan's work on the electric furnace, the number of those interested in it is limited. It, therefore, appears to us that Prof. Moissan should not have permitted two translations in the English language to appear.

The present translation is fairly satisfactory, although the adherence to the original French is in many places too close.

Complete Expansion Gas Engine.

The Sargent engine was not only the first successful double-acting tandem gas engine, but the first and only one to expand the burning charge to practically atmospheric pressure, to vary the point of cut-off of the admission inlet with the load, and to advance the time of ignition as the mixture gets weaker and the inflammation slower. The advantages thus secured over the ordinary gas engine are: Increased efficiency, increased regularity in speed, and smooth running under early cut-offs.

The general design, as shown in Fig. 1, is not only symmetrical and pleasing to the eye, but is such that all the strains come in a straight line, yet the free expansion of the cylinders and rods is provided for. The sub-base, the top of which comes flush with the floor line, gives a flat planed surface, upon which the engine is easily erected and aligned. The sub-base and main frame are bolted to the foundation, and the cross-head guide, cylinders and distance head, which fasten to the main frame, can come and go as the temperature varies, sliding on the hollow supports rising from sub-base, which not only maintain the cylinders in line, but convey the gas and air from the hollow divided sub-base to the explosion chambers, thus doing away with the pipes, ordinarily so unsightly and in the way.

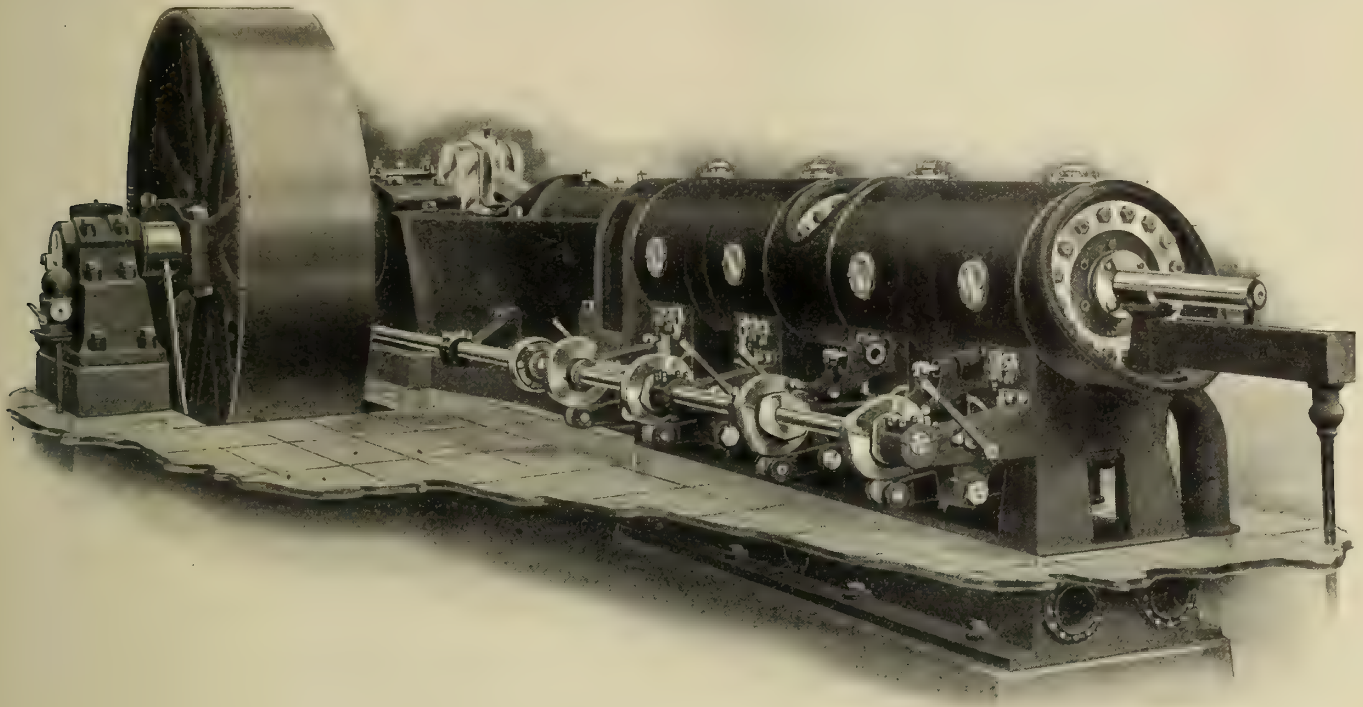


FIG. 1.—500-HP. SARGENT DOUBLE-ACTING TANDEM GAS ENGINE.

In the ordinary gas engine, the piston at full load draws in a cylinder full of combustible mixture, which after compression and ignition is expanded to the original volume and released at a pressure of 25 to 40 pounds absolute, and a temperature of from 900° to 1500° F. In the Sargent gas engine the admission of gas and air at full load is cut-off from five-eighths to three-quarters of the admission stroke, depending on the fuel used, which, after compression and ignition, is expanded to the cylinder volume and is released a little above atmospheric pressure with a corresponding temperature of about 400° F. The heavy line (Fig. 2) shows the indicator diagram of the ordinary gas engine at full load, and the light line extension shows the increased power with the same fuel obtained from the Sargent complete expansion engine. The point of cut-off, while constant for the full and most economical load of the engine, is made earlier by the governor as the load gets lighter or later as the load gets heavier, giving a flexibility unobtainable in the ordinary internal combustion engine.

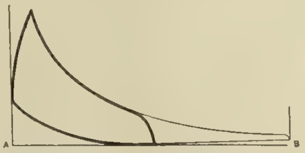


FIG. 2.—INDICATOR DIAGRAM.

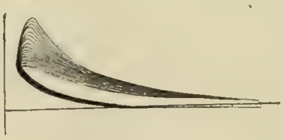


FIG. 4.—DIAGRAM OF VARIABLE LOAD.

No throttling action of the incoming charge, with its corresponding loss of efficiency, so apparent in the ordinary gas engine, is possible, and owing to the more complete expansion of the burning gases turning the heat into work instead of throwing it away in the exhaust, a lower terminal temperature, and, consequently, a lower average temperature of the cylinders is maintained throughout the cycle. For this reason engines of this type below 100 hp. require no water-cooled pistons or rods.

The mechanism of the Sargent engine is distinctively simple

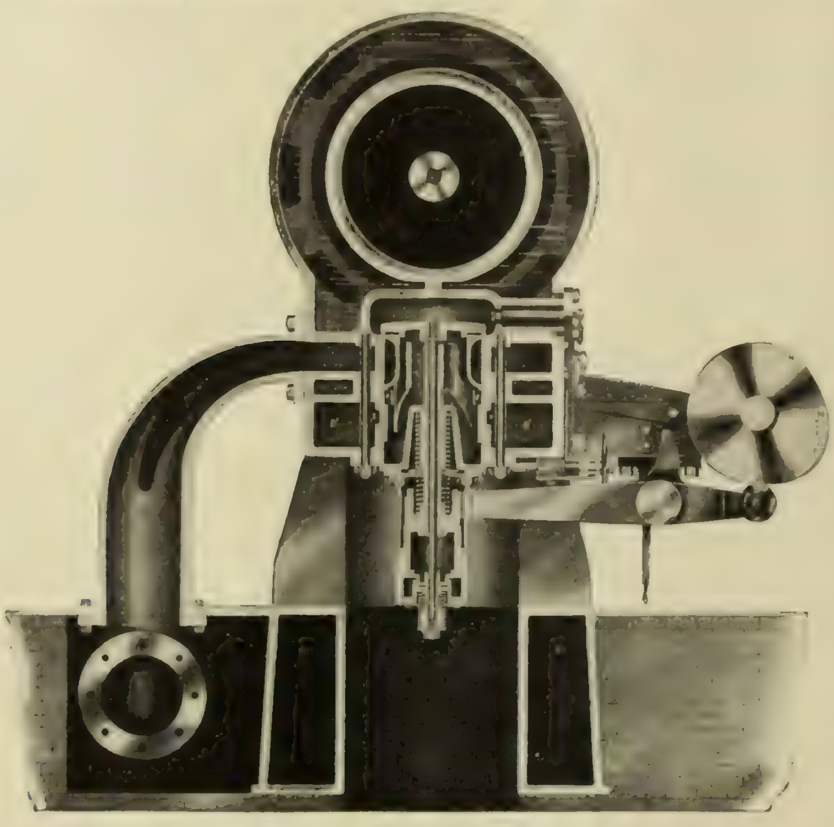


FIG. 3.—VALVE SECTION.

A side shaft, driven by the crank shaft and governor through a pair of worm gears running in oil, carrying two cams for each explosion chamber, one for the igniter and one to operate the valve, comprises all the moving mechanism except the valves and lever, as shown in Fig. 1, the other side of the engine being perfectly plain.

Fig. 3 shows a section through the valves of one of the explosion chambers. By removing six nuts, valve bushing and valves can be removed from the cylinder for regrinding or inspection.

Gas is piped to the chamber A in the sub-base and air to the chamber B, which pass through the cylinder supports to the chambers A′ and B′, ready to pass into the mixing chamber when the cam depression M N passes the roller and the ports F in the piston valve register with the ports E and D in the bushing. When the piston valve goes down to this position the confined air in the piston valve dash-pot forces open the poppet valve, thus giving free admission to the charge. When the point N of the cam reaches the roller it is forced down, while the other end of the lever goes up, carrying the piston valve, which cuts off the admission. The poppet valve seats and both valves remain in normal position during compression, ignition and expansion, or until the point L on the cam pushes the roller down and the piston up, which opens the poppet, and the exhaust gases pass out through ports K and the elbow W to the exhaust pipe under the floor.

The poppet valve seals the opening in the combustion chamber during compression and inflammation, and the piston valve, holding against no pressure, works loosely in its bushing, cutting off the admission and guiding the exhaust.

As the poppet valve controls both the inlet and outlet gases, both the valve and seat keep cool and need to be ground not over once or twice a year. The mechanism is simple, and as the roller is always bearing on the cam the valve motion is practically noiseless.

By revolving the piston valve by the index wheel, the blind port S varies the mixture to suit the gas, whether it has 100 or 1000 B. T. U. per cubic foot.

The speed of the Sargent engine is controlled by the well-known Rites inertia governor in the fly-wheel, which advances the valve shaft in time ahead of the crank shaft, as the speed increases, thus diminishing the mean effective pressure with the load. As the load gets lighter the cut-off becomes earlier, taking less of a constant mixture of gas and air into the cylinder, but as the burnt products in the clearance do not get less, the mixture gets weaker and slower burning. If the ignition were at the same point as at full load, as is the case in the ordinary gas engine, the highest pressure would not be attained till the piston approached the middle of the stroke, where the cooling surface is so increased that the greater part of the heat goes into the water-jacket rather than into the work. In the Sargent engine the time of ignition advances with the cut-off, getting earlier as the mixture gets weaker, thus maintaining the highest pressure at the begining of the stroke, irrespective of the load, as shown by Fig. 4, an actual diagram of a variable load from the Sargent gas engine.

While the governor controls the speed through all ranges of load, the gas engine, like the steam engine, is not economical with very light loads. Where a variation in the turning moment is not objectionable, one or more of the explosion chambers may be cut out at will by the engineer, by simply raising to horizontal position a controlling lever, which holds the exhaust open and the gas and air closed. This is very desirable in blowing engines, as they can be designed so that two explosion chambers will furnish air at 15 pounds and the four chambers at 30 pounds pressure

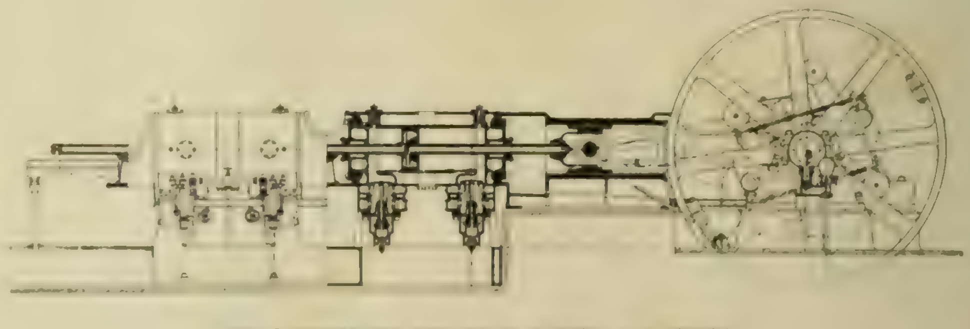


FIG. 5.—COMPLETE EXPANSION TANDEM GAS ENGINE.

If at full load the Sargent engine is cutting off at three-quarters stroke, an overload may be taken care of by a later cut, though at a decrease of efficiency.

The cylinders are oiled by a force feed-pump or check valve lubricators. The valves get sufficient oil from the cylinders.

The side shaft and out-board bearings are self-oiling. The cross-head guides, pin crank, pin and main bearings, which must be thoroughly lubricated at all times, are copiously oiled by the worm gears, which act as a pump.

Two electric ignitors are placed in each explosion chamber in such a position that they are surrounded with a pure mixture at the time of ignition. Either will fire the charge, and should either become short-circuited the engineer is immediately advised.

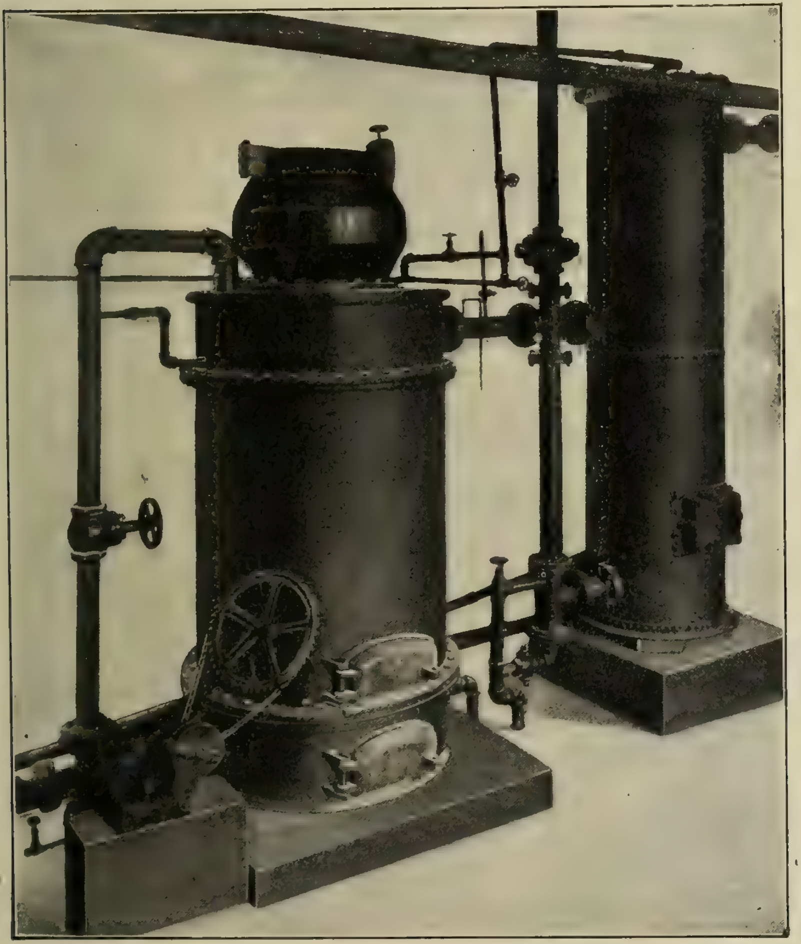
The engine is started by compressed air. The pressure of the air when turned on at one cylinder puts the starting mechanism into operation and at the same time puts the cylinder out of service as a gas engine.

When the engine is up to speed, turning off the air puts the gas engine into commission without changing a valve, cam or lever.

The Sargent engine is built by the Wellman-Seaver-Morgan Company, of Cleveland.

Suction Gas Producers.

Suction gas-producer plants are being more and more introduced with great success in this country on account of their simple working and great economy in running and maintaining expenses. They are used in connection with gas engines for developing power for all kinds of industries and shops. A general discussion of suction gas producers was already given on page 432 of our Vol. II.



SUCTION GAS PLANT.

A suction gas producer, as shown in the cut, is now working successfully in the machine shop of the F. W. Horstmann Co., East Newark, N. J. This shop, which was using before illuminating gas in connection with a gas engine, is now using a gas producer. The plant, which was built by Dr. Oskar Nagel, of New York City, is working according to the guarantee given with the plant, that is, the development of 1 B. H. P. hour from 1¼ pounds of anthracite (pea) on full load, 1 1-3 pounds on three-fourths load, and 1½ pounds on one-half load.

The plant has a capacity of 20 hp., and consists of a hand-blower, producer with evaporator on top of same, and a scrubber, which is shown on the cut. The overflow water pot which belongs to the plant is in the pit between the producer and the scrubber, and a small equalizing tank is connected on the floor below to the engine, so as to connect the scrubber with the engine.

By the sucking action of the engine the air is drawn over the surface of hot-water in a water-jacket and saturated with steam, and this saturated mixture of steam and air is drawn through the fuel, whereby the producer gas is generated. From the producer the gas goes through the scrubber, which is filled with coke, and where it is freed from dust and tar by means of water. From the scrubber the gas goes through the equalizer, which is a simple iron drum, to the engine.

This plant is provided with a hopper, which is sufficiently large so as to contain fuel for the whole working day, so that no refilling of coal is required during the working hours. After shutting down over night the lower door is kept somewhat open and also the valve leading to the flue, so as to keep up a little draft and maintain the fire. In the morning, before starting up, the fire is cleaned from the ashes and the hopper filled with coal, and then the fire is blown hot for about ten or fifteen minutes by means of a hand-blower, until the gas is burning well at the test cock, and then the engine is started as usual.

During the working day the plant does not require any attention except cleaning the grate once or twice a day.

Pyrometer.

The improvements in the economy of working which have been realized in recent years in chemical and metallurgical industries are due to a great extent to the systematic attention which is now being paid continually during the manufacturing process to all factors which determine its efficiency. The temperature is one of the most imporant factors. In a great many cases the reaction which is desired will take place only between certain limits of temperature, and to get the best efficiency the temperature must be restricted within still narrower limits. To accomplish this in practice it is, of course, first necessary to measure the temperature exactly, while for purposes of control, etc., a continuous record of the changes of temperature is highly desirable.

The requirements of chemical and metallurgical practice have led to the design of various types of registering pyrometers, most of which have been described in these columns. It is, of course, understood that it would be wrong to consider one single type of registering instrument as the best one for all purposes, since the type to be chosen depends upon the temperature to be measured and on the conditions of the operation.

The accompanying illustration shows a very interesting new pyrometer for temperatures between 0 and 2500° C. (*i. e.*, up to about 4500° F.). The pyrometer was invented by Mr. F. de Saintignon, and is built by Messrs Richard Frères, in Paris.

The principle is extremely simple. A double metallic tube *EF* is placed in the room, or apparatus, the temperature of which is to be measured. If water is now passed through this tube with a constant speed sufficiently high to prevent evaporation, the difference of the temperatures of the water when entering and leaving the tube indicates the desired temperature if the speed of the water is known.

The whole apparatus, shown diagrammatically in Fig. 1, consists essentially of the double metallic tube *EFG*, which is called the "explorer," and the two thermometers T^1 and T^2 The water is supplied from a reservoir in which it is held at a constant level. One of the two thermometers T^1 and T^2 measures the temperature of the water while passing into the metallic tube, the other while passing out.

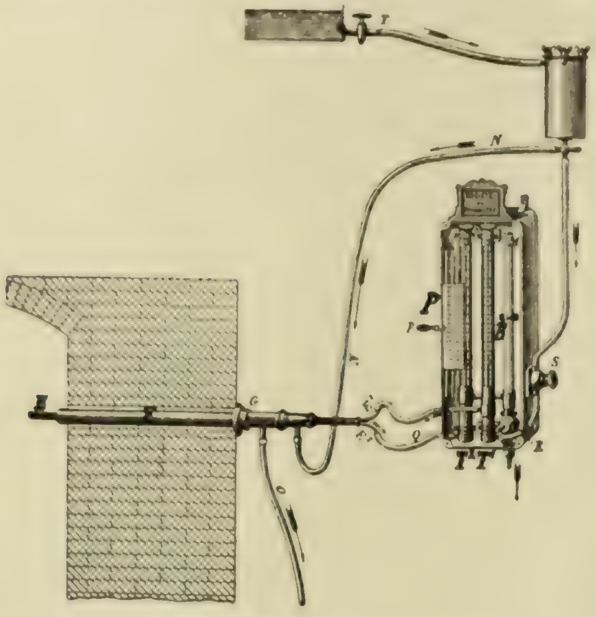


FIG. 1.—PYROMETER WITH FLOWING WATER.

The speed of the water is regulated by means of the cock *R*, the height of the water in the manometer being held at the level *B*. In order to read the temperature it is only necessary to adjust the zero point of the scale *P* to the temperature of the incoming water. Then the temperature of the outgoing water (which is now also the difference of both the temperatures) gives on scale *P* directly the desired temperature in the furnace.

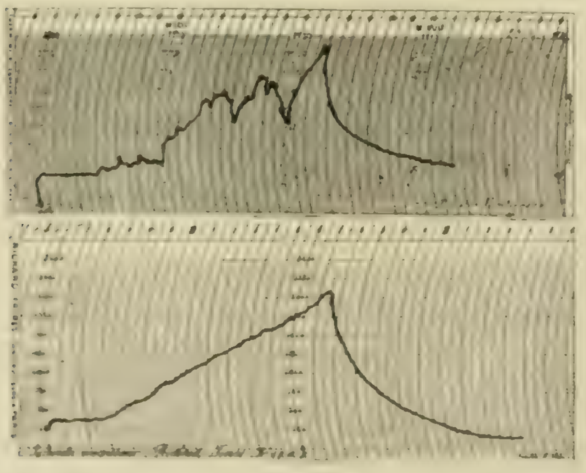


FIG. 2.—TEMPERATURE DIAGRAMS.

The company has recently rendered the instruments self registering, and [illegible] instruments of this kind are now in successful use in France in numerous glass and china factories as well as in open-hearth furnace plants. By taking a record of the temperature during a total run it has been possible to trace sources of inefficiency and to improve the operation essentially. Fig. 2 is interesting in this respect; the upper diagram was taken when an instrument of this type was first installed in the Say refinery in Paris; it is evident that the regulation of the temperature was not at all as desired. After this had been recognized it was easy to correct this, and the result is shown in the lower diagram.

Continuous records made by recording instruments are the safest indication whether the operation is at all times being carried out under the best conditions.

The instrument described is now being introduced in this country by Mr. Ernest H. du Vivier, 16 Church Street, New York City.

Industrial Notes.

PLATINUM.—Messrs. BAKER & Co., Inc., refiners and manufacturers of platinum, of Newark, N. J., and New York City, are making extensive additions to their Newark works. The rapidly increasing consumption of platinum in the industrial arts has made this extension necessary, especially in their refining department, which will be enlarged over 100 per cent. New and larger offices are also being erected, and the entire plant equipped with the most modern appliances for the economical manufacture of their varied and high-grade products.

NEW WANNER PYROMETER.—Patents have been allowed on the construction of the Wanner pyrometer for measuring temperatures ranging from 620° C. to 1000° C. This instrument supplies a means of making determinations up to that temperature at which the original high-temperature type of Wanner pyrometer begins. The new instrument embodies the same principles, and is known as the Wanner low-temperature optical pyrometer. It is particularly adapted for measuring temperatures in tempering steel, and is said to be the first pyrometer on the market especially adapted for this purpose. The instrument is well finished throughout and is easy to handle. The incandescent light used in standardizing is a 2-volt osmium lamp, thus reducing the size of the battery necessary, and making the instrument still more portable and convenient. Messrs. EIMER & AMEND, of New York, are the agents in this country for the instrument.

GOLD MILLING IN THE BLACK HILLS.—Prof. H. O. Hofman's highly interesting paper, read before the American Institute of Mining Engineers, on the methods of gold milling used in the mills of the Black Hills of Dakota, has been reprinted by the ALLIS-CHALMERS Co. as special publication No. 4000. The pamphlet is profusely illustrated with sectional diagrams and exterior views of gold milling machinery.

TUBE MILLS.—An illustrated pamphlet on Krupp tube mills, with continuous feed and discharge for dry grinding of all materials, has been sent us by the American representatives, Messrs. THOS. PROSSER & SON, New York.

We have received from the AMERICAN PLATINUM WORKS, of Newark, N. J., a handsome and nicely finished 15-inch desk rule.

The CROCKER-WHEELER Co. is in receipt of an order for a 200-kilo volt-amp., three-phase, 60-cycle, engine-type alternating-current generator for the Ivorydale (Ohio) lighting and power plant of the Proctor & Gamble Co. This machine is a duplicate of the first Crocker-Wheeler alternator ever built, which was installed ten months ago in the Atlanta plant of the Proctor & Gamble Co. The high efficiency and excellent performance of the earlier machine lead the purchaser to cancel an order for another type of generator and to order a duplicate of the first machine.

SMALL MOTORS.—Bulletin No. 60 of the CROCKER-WHEELER Co. gives data on the construction of their small direct-current motors from ¼ to 3 hp., with numerous illustrations of the application of such motors to the operation of various industrial machinery, with some elementary but useful information on electric units, the size and speed of driving and driven pulleys, the weight of belts and the carrying capacity and dimensions of wires.

Nernst Lamp.—No. 8 of the Nernst Central Station Bulletin, published by the Nernst Lamp Co., of Pittsburg, Pa., deals with "the use of the Nernst lamp as a peak broadener," by enabling special inducements to be made to desirable customers by means of a combination of a two-rate system of charging with the use of high efficiency lamps like the Nernst.

We have received from Messrs. Chas. Cooper & Co., 194 Worth Street, New York, their monthly price list of chemicals for August. The prices are mostly given per pound or gallon; a statement which is made in various cases whether the prices are advancing or declining will be found useful.

Storage Batteries as Voltage Regulators.—Bulletin 2 of the National Battery Co., of Buffalo, N. Y., deals with the use of storage batteries for line voltage regulation on a hydroelectric system, and describes the application of this principle on the system of the Syracuse & Suburban Railroad Co.

Motor Control.—Circular 1108 of the Westinghouse Electric Mfg. Co. gives fully illustrated descriptions of regulating and reversing controllers for direct-current motors and polyphase induction motors.

Smelting Furnaces and Accessory Equipment.—Catalogue No. 122 of the Allis-Chalmers Co. (55 pages) deals with smelting machinery. A brief introductory chapter on the metallurgical principles of smelting in general is followed by two concise and very interesting chapters on modern blast furnaces for lead smelting and for copper smelting, and shorter notes on the use of hot blast, water-jackets, forehearths, slag, matte and bullion pots, charging apparatus and blowers. The pamphlet is very fully illustrated by diagrams and views of the latest types of smelting machinery.

Hot Blast.—We have received from Mr. George Mitchell, 52 Wall Street, New York, a pamphlet on his economic hot-blast furnace. The well-known advantages of the use of hot blast for the smelting of many sulphide ores with a considerable reduction of the quantity of coke required, are pointed out. In the Mitchell furnace the blast is not heated by means of a separate heating stove with extra fuel, but the heat of the furnace itself is utilized. The blast is delivered from positive pressure blowers to the furnace, where it enters primary air jackets, and thence travels in a zigzag course, and is heated in its passage by direct contact with the walls of chambers; the air is then taken to the secondary air jacket, where it is again heated, and passes from there through a series of pipes to the tuyeres. Details of construction are shown in a diagram in the pamphlet. An incidental advantage is that the incoming supply of air maintains the walls of the furnace above the feed floor opening at a sufficiently low temperature to render the use of firebrick or other refractory lining unnecessary, and permits of the use of a metallic furnace section for conveying away the heated gases and unconsumed products of consumption. The pamphlet states that in the whole twenty-nine Mitchell furnaces are in successful operation or in course of erection in Mexico and Arizona.

Some Details as to Smelting Practice and Equipments.—Under this title the Colorado Iron Work Co. have just issued their catalogue No. 12 (157 pages), which is a very interesting and profusely illustrated discussion of modern practice and tendencies in the reduction of gold, silver, lead and copper ores. A considerable portion of the book deals with the use of hot blast; the true economy of this method is discussed in a very suggestive way. Then follow illustrations, with brief descriptions, of a great many types of modern smelting furnaces and accessory smelting machinery.

Gas Power.—A very neat and fully illustrated pamphlet of the De La Vergne Machine Co., of New York City, gives a detailed description of the Koerting 4-cycle gas engine, as well as illustrated notes on gas producers. The following are the results of tests of two Koerting engines, operating on gas of a calorific value of about 500 B. T. U. per cubic foot:

Engine 4,190.	R. P. M.	B. H. P.	Cubic Feet Gas per Hour.	Thermal Efficiency.
Full load........	161	108.10	1892.9	28.22%
Two-thirds load..	165	74.27	1472.6	
One-third load...	167	37.38	987.4	
Engine 4,249.				
Full load.........	140	114.22	1902.6	31.96%
Two-thirds load..	140	76.64	1453.5	

Some illustrated notes are given on gas producers and an appendix gives useful engineering data in form of tables on tests of producer gas plants, comparative data of steam and gas plant tests, thermal and chemical data regarding combustion, characteristics of power gases, calorific values of elementary gases, conversion factors of units and comparative tables of thermometer scales.

Copper Mining.—Some recently published data on copper mining in various districts of the United States show the most remarkable growth of this industry during the last years. While only three years ago some of the best students of the lake copper business considered 200,000,000 lbs. per year as the ultimate productive capacity of the Lake Superior mines, this figure has already been exceeded in 1904, and will be largely exceeded during 1905, barring serious accidents of an unforeseen nature. Of the important old producers the Calumet and Hecla has work under way which should result in increasing its production to approximately 100,000,000 pounds a year within the next few years. From the Butte district it is stated that there were shipped from the Washoe smelter in June more than 15,000,000 pounds of copper, and that the July shipment was but little less. As the Washoe produces little more than half the copper of the Butte district the total output was in excess of 30,000,000 for each of the two months; the production for August was lower. The Guggenheim Exploration Co., of New York, is stated to have taken the entire issue of $3,000,000 of bonds made by the Utah Copper Co., composed of Colorado Springs capitalists. Of the issue $750,000 will be applied to the redemption of a previous issue. The remainder will be applied to the construction of a 6000-ton mill at Bingham, Utah. Work will begin at once, and by spring installation of machinery will begin. In connection with the reduction works at Bingham the company will have a total capacity of 7000 tons.

Exports from United States.—The Department of Commerce and Labor has sent to manufacturers and exporters a circular, requesting their coöperation in the efforts being made by the department, through the Bureau of Manufacturers, to extend the foreign trade of the United States. In furtherance of this work, it is proposed to establish a comprehensive card index, which will enable the department, upon application, to furnish information desired by manufacturers, or by intending purchasers. If the necessary authority shall be granted by Congress it is contemplated to extend the system to the principal consulates. For this purpose the bureau has prepared a blank, which is to be filled out and to be returned to John M. Carson, Chief of the Bureau of Manufacturers, Washington. The information requested on the blank refers to name and address of main office, location of branch establishments, description of product, capital, capacity per day, month or year; the places where the product is sold (stating for foreign countries the ports to which it is shipped), etc. The idea is undoubtedly an excellent one, and the information contained in such a card index should prove extremely valuable in future, if our manufacturers comply with the request for information made by the department.

Obituary.

John Herbert Bagg, secretary of the General Chemical Co., died Aug. 16 at his residence at Bay Ridge, N. Y., aged 37 years, of typhoid fever. Mr. Bagg was born near Brockville, Ontario, and was educated there. He became connected with the Brockville Chemical Works in 1888, and moved to New York in Feb., 1889; since that time he has been actively identified with the chemical industries, first in connection with the Nichols Chemical Co. and later with the interests consolidated in the General Chemical Co. In a resolution adopted by the board of directors of the General Chemical Co., and expressing their profound sorrow at Mr. Bagg's death, it is said that "by his unusual sagacity, his devotion to duty, his loyalty and high standard of honor, Mr. Bagg earned the unbounded respect and confidence of his business associates; while his pure and irreproachable life, his sturdy manhood, his generous impulses, his amiable and sympathetic nature, won the love and esteem of all with whom he came in contact." Mr. Bagg was married to Miss Mary Christine Carter, of Baltimore, and leaves besides his widow a young son.

Digest of U. S. Patents.

Compiled by Byrnes & Townsend, Patent Lawyers, National Union Building, Washington, D. C.

Electric Smelting and Reduction Processes.

(*Continued.*)

No. 614,927, Nov. 29, 1898, George D. Burton, Boston, Mass.

Roasts and reduces ores containing sulphur, arsenic, antimony, zinc, lead, etc., in a resistance furnace. The furnace is a vertical cylindrical chamber having walls of electrically non-conducting material glazed inside and out with a mixture of clay, 5 pounds; sodium chloride, 2 pounds; borax, 1 pound, and glass, 1 pound. The resistor is a perforated central pipe with a wide, hollow, spiral flange, perforated at the periphery, consisting of an alloy of copper, 3 parts; antimony, 1 part, and tin, 1 part. The ends extend through the top and bottom of the furnace and are connected to the terminals of an electric generator giving 8000 to 12,000 amps. at 8 to 50 volts. The resistor is mounted to rotate, to stir the charge when necessary. The furnace cover has a charging opening and a gas outlet leading to a valved horizontal flue containing baffles, the end of which dips into a tank of water. The ends of horseshoe magnets may extend into the flue, to retain magnetic particles. The furnace has a drop-bottom, with tap-hole. Valved pipes are connected to the upper and lower ends of the resistor. In using the furnace, the ore may be mixed with charcoal, coke, carbon dust, lime, a flux, etc. As the ore is heated, "obnoxious gases" and "moist vapors" are eliminated and pass through the openings in the resistor and out through its ends. The heat is then increased and arsenic and sulphur pass into the flue, wherein they are deposited. Air is then admitted through the resistor, burning out the remaining sulphur. The heat is further raised until the metal of lowest fusing point is reduced, whereupon it is tapped out. Additional flux may be added, the heat is again increased and the metal of higher fusing point is reduced and tapped out. Some ores require no reducing agent. It is stated that the furnace may be used for the manufacture of calcium carbide, the resistor then being of carbon.

No. 626,635, June 6, 1899, Gustav Schwahn, St. Louis, Mo.

Volatilizes alumina or aluminium fluoride or chloride, and reduces the vapor by incandescent carbon, carbon monoxide and fluorine, chlorine or aluminium fluoride or chloride. The alumina is volatilized by an arc; other compounds by heat of combustion. The charge may consist of the aluminous compound 98 parts, a fluoride 1 part and carbon 1 part. The reduced aluminium collects and is tapped or allowed to run out, the gaseous non-metallic compounds being led off. Four types of furnaces are shown: That for treating alumina comprises a hopper which delivers the charge through a rotating sieve into and through an arc sprung between the edges of horizontal carbon plates. The vapor passes into a horizontal reduction chamber, having walls of magnetite, calcium aluminate or fire-clay. This chamber is traversed by two grids of vertical carbon rods, which are heated to incandescence by the passage of an electric current. The upper ends of each set of rods are clamped between water-cooled metal or graphite holders; the lower ends extend into dished carbon blocks holding molten aluminium, to make good contact. The reduced metal is continuously withdrawn through a trap. The second furnace is similar, but employs an external combustion furnace to volatilize the aluminium compound. The third and fourth furnaces are similar to the first and second, respectively, except that the reducing chamber is filled with broken carbon, heated to incandescence either by the passage of an electric current through it or by a combustion furnace below.

No. 648,119, April 24, 1900, Emile Vielhomme, Froges, France.

Produces rich ferrochromium by electrically smelting a charge of 100 parts of chromite, containing oxides of chromium, iron, magnesium, calcium, aluminium and silicon, sufficient broken coke to effect reduction and 20 to 25 parts of a flux, as lime, sand, kaolin or fluorspar. When the metals are completely reduced and fused the heating is continued until the iron is wholly or partially volatilized. The flux gives a fusible slag which acts as a resistance, facilitating easy regulation and the production of a high temperature. It also partially volatilizes and carries off the iron vapor.

No. 648,439, May 1, 1900, Auguste J. Rossi, New York, N. Y.

Produces ferrotitanium, mainly by the reducing action of a molten bath of aluminium or zinc. Carbon may be added, to assist in reduction. The iron is supplied or reduced from its oxide in the charge. The heating is preferably effected by springing an arc between a graphite crucible containing the charge and a depending electrode. Examples: (1) Aluminium, 100 parts, is melted in the crucible, and iron, 50 parts, is then added and melted. Into the bath are fed 275 parts of a briquetted mixture of titanic oxide, 56 per cent, and iron oxide 44 per cent. The charge gradually disappears with incandescence, whereupon the electric current is decreased until the mixture is all melted. The temperature is finally raised, to give a homogeneous alloy, containing 40 to 62 per cent of titanium. (2) A bath of aluminium, 60 to 63 parts, and iron, 125 parts; the charge, 275 parts of a briquetted mixture containing titanic oxide, 56 per cent; iron oxide, 44 per cent, and carbon, 35 to 37 parts. The resulting alloy containing 29.46 per cent of titanium. (3) Titaniferous mixture, 275 parts. Reducing agent, 83 parts, to wit: carbon, 58 to 70 per cent, and aluminium 25 to 30 per cent. Iron, 250 parts. The resulting alloy contains 19.82 per cent titanium. (4) Titaniferous slag, containing 60-62 per cent of titanic oxide, 100 parts: titaniferous ore, 18 per cent titanic oxide, 175 parts; iron, 200 parts; aluminium, 90 parts. The resulting alloy contains 14.08 per cent titanium.

No. 648,463, May 1, 1900, Richard I. Knaur, Austria-Hungary, Harold W. Buck, Schenectady, N. Y., and Charles B. Jacobs, East Orange, N. J.

Produces silicon hydride by electrically heating a silicious material to incandescence or fusion and treating it with a gas containing hydrogen, *e. g.*, water gas. The silica, clay or other silicious ore is heaped upon the bottom of a furnace chamber, through the end walls of which pass horizontal carbon electrodes. The ore is heated by an arc, sprung between the electrodes near its upper surface. The hydrogen gas is introduced through the bottom of the furnace, and the hydride escapes from the top. The residue, aluminium silicate, is useful as an abradant.

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Bethlehem Meeting of the American Electrochemical Society.

That electrochemists are not yet masters of the elements was proven by the inclement weather during the three days of the Bethlehem meeting of the American Electrochemical Society. But it is characteristic of the spirit of the members that the weather could not detract in any way from the good feeling which reigned supreme. As a matter of fact, the meeting was very much enjoyed and will long be remembered, thanks to the excellent arrangements made by the local committee. A full account of the proceedings and papers is given elsewhere in this issue, so that it may suffice here to point to the very broad range of subjects treated in the papers presented. The programme was broad enough to satisfy any taste; and this is necessary for the success of the Society. Electrochemical science and engineering is a special field, but one which is bound to make itself felt in a great many different industries and professions. The great interest with which electric furnace methods are now considered in the iron and steel industry was indicated by a long and animated discussion on the subject in connection with a paper of Mr. Gin. Otherwise, there was not very much discussion. Several decisions as to some special points of the future policy of the Society call for comment. There will be only one general meeting a year in future. This will undoubtedly lead to concentration and better attendance. However, arrangements may be made to hold a second meeting in conjunction with one of the other national engineering societies, as the American Institute of Mining Engineers, the American Chemical Society and the American Institute of Electrical Engineers. If carefully handled, such joint meetings must lead to an expansion of the interests of electrochemistry. Finally, the Society will encourage—and to some extent support—research work. Here is a most promising field, and any member of the Society can do useful work in this respect. A circular letter is to be sent out to all members asking for suggestions as to such research work as is considered of special importance. Even if only a small part of this work can be carried out with the support of the Society, yet a list of such suggestions should prove exceedingly useful, if the members respond to the request in the right way and in the professional spirit which has been from the beginning such a distinguishing feature of the Society.

◆

The Huntington-Heberlein Process.

A chain is no stronger than its weakest link. A fleet is no faster than its slowest ship. A scientific investigation is no more accurate than its weakest determination. And, as in all these three examples, the metallurgy of a metal is no stronger than its weakest sub-process. In lead smelting there has been great recent progress in the mechanical charging of the blast [illegible] the mechanical handling of slag and matte, in the chemistry of slags, in the knowledge of the proper reduction of iron, and the elimination of this reduced iron by its reaction with sulphide or oxide of lead to form ferrous sub-sulphide or ferrous silicate respectively. Large advance steps have also been made in concentrating lead ores and in the mechanical and chemical sides of refining the base bullion. In the department of roasting lead sulphide ores, the attempts for improvement have met with but little success in this country until the present year. In fact, the Murray plant of the American Smelting & Refining Co., which was built recently by the combined experience of the best metallurgists of the trust, was equipped largely with the standard old-fashioned hand-rabbled reverberatory furnaces. Rotary roasters, such as the Bruckner, stand idle at Pueblo, while the old types of hand roaster, which do not "snowball" the ore or cause excessive flue dust losses, are operated at a labor charge of $2.00 per ton. Furnaces, as the Ropp, Brown and McDougall, used with such success in roasting gold, zinc and copper ores, at a cost for labor and repairs of often as low as 25 cents per ton, are not capable of use on lead ores, because of the tendency of lead ores to form accretions on the hearth.

Accordingly, it is seen that in the lead business there was a great chance for improvement for a process founded on correct chemical facts and carried out on correct engineering lines. Such a process must succeed if it dispenses with that uncertain and unsatisfactory factor, the hand-rabbled roasting furnace. The Huntington-Heberlein process which fills this catagory has been developed in Europe and Australia in the past ten years, and is about to be exploited with modifications and improvements by the American Smelting & Refining Co., the purchaser of the rights for the United States and Mexico. This process combines into a homogeneous whole two facts long known to well-informed metallurgists. One of these is that the number of pounds of ore roasted per square foot of hearth area could be tremendously increased if it was possible to pass a current of air through the ore by means of porous or perforated hearth. The other is the fact that lime has a great affinity for sulphur, and will take unto itself sulphur on the slightest provocation. This is the case in the basic open-hearth furnace which uses a basic slag to eliminate the sulphur of the pig iron, and it is also applied in the iron-blast furnace by working the slag as basic as possible. Dr. Héroult uses in his electric furnace, with a very high temperature, a slag so basic as to be infusible in the lower temperature of the open-hearth furnace, and he can thereby easily "wash" all the sulphur from his metal. In the "Tarnowitz" process, working on lead ores, recourse has been long made to the desulphurizing properties of dolomite. And at the last meeting of the American Electrochemical Society, Brown and Oesterle corroborated the experiments of both Percy and Prost as to the extent of lime in desulphurizing zinc ores. Their paper will be found on another page of this issue.

In the Huntington-Heberlein process green or partially roasted lead ores are mixed with limestone and charged into a conical cast-iron converter, which has a perforated false bottom, air is blown gently through the ores, the temperature at first being held at 700° C., and then reduced to 500° C. At the end of this operation the mass is partially sintered, the sulphur being reduced to a low percentage. The converter is then tipped, the roasted product dumped, allowed to cool, broken up and charged into the lead furnace. The process has many advantages. The fuel cost and labor charge are both insignificant. The product is fitted by its physical characteristics for rapid smelting in the shaft furnace. The flue losses are extremely low, because the process is operated at such a low heat. A further advantage is the fact that the waste gases are high enough in sulphur dioxide to be used in the chamber sulphuric acid process. There are many successful imitations of this process; among which might be mentioned the Savelsberg and the Carmichael-Bradford. But all processes depend on the lime-sulphur reaction and on the ability of limestone to keep the charge free and open for the uniform flow of air. The chemical theory of any of these processes is not too well known, but the experimental facts are certainties, and after all experimenting is the basis of all our metallurgical knowledge. We are glad to chronicle this advance in lead smelting, for it shows a certain brotherhood in the metallurgy of metals, and beside, is a step in advance in the art of making metals from ores.

The Chemistry of Electrochemistry.

Dr. Wilder D. Bancroft's experimental lecture at the Bethlehem meeting of the American Electrochemical Society emphasized a point of view which has been generally neglected in the development of electrochemistry. The specific action of the electric current in electrolysis is considered here to be nothing but the transport and discharge of the ions. What then happens is looked upon as a problem of pure chemistry. By purely chemical experiments Dr. Bancroft endeavors to show which one of the various reactions that would be *a priori* possible, will actually take place. A weak point in this procedure would apparently be the difficulty to decide what the ions in a certain solution are. Dr. Bancroft says that this should be determined by some such method as Hittorf's, but it becomes daily more evident that such a method is not sufficient for the purpose. For instance, in an aqueous solution it is impossible to decide by measurements of the concentration changes alone whether there are complex ions (combinations of the simple ions with the molecules of the solvent) or not. However, for the purpose of Dr. Bancroft this is really a small matter. Take as example the development of the theory of the lead accumulator. Plante and many after him thought that the action during discharge was decomposition of water with a resulting change from lead peroxide to lead oxide on one side, and from spongy lead to lead oxide on the other side. Now, if they had looked on the problem as a chemical one, as Dr. Bancroft urges, the experiment would have suggested itself to make chemically lead oxide plates and place them in sulphuric acid. Then it would have at once been evident that something happens, that there is no chemical equilibrium. As a matter of fact, it was through Gladstone and Tribe's chemical researches that the sulphate theory of the accumulator was established. What shall be assumed as the hypothetical "primary" reaction in a certain case will depend on the hypothetical assumption of the nature of the ions; but whatever this assumption may be, chemical research will show what, under this condition, the subsequent reaction must be. From the point of view of Dr. Bancroft it will always be possible to determine the whole action, and this is the main point.

Against this viewpoint of Dr. Bancroft, Mr. C. J. Reed emphasized, in a communicated discussion, the thermochemical point of view, as pronounced in Berthelot's principe du travail maximum. That process will take place which has the greatest heat of reaction; and since, according to Thomson's rule, heat of reaction is proportional to e. m. f., the determination of the e. m. f's corresponding to the different possible reactions from thermochemical data will show what reaction will really take place. If we only modify this view by substituting free energy of reaction for heat of reaction and Helmholtz' formula for Thomson's rule, then there is nothing whatever to say against this point of view. It will lead to the solution of the problem. But it would be wrong to believe that this is the only way of looking at the matter. What will take place depends on the free energy of the reaction, and this may be found by measurements of e. m. f's or by purely chemical research. The former viewpoint has been prominent in the past. Dr. Bancroft is right in pushing the chemical viewpoint now to the front. What we need more than anything else is a broad way of looking at things. Only by attacking a problem from all sides can we master it. It is to be hoped that Dr. Bancroft's lecture will contribute toward this end.

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Recent Progress in Ore-Dressing.

It is commonplace to-day to say that this era is an era of growing complexity. Indeed, complexity is common to all subjects, and as we have repeatedly pointed out on these pages, it is a logical outcome of evolution. All things, from a zoöphyte to a monster blast-furnace, are subject to this great doctrine. This complexity affects our social life. It affects our religious thought. It also affects our commercial world. Science has revolutionized theoretical dogma and applied science has created our wonderful industrial structure. In all things, then, we can look for its profound influence. Metallurgy has advanced in the past hundred years, by leaps and bounds, from a simple to a wonderful art. All its branches have expanded. But of all, ore dressing, we believe, has shown that great and astounding growth that is most representative of our American industry. It is an example of complex application of scientific principles.

Ore dressing takes the crude ore, subjects it to a mechanical process, and makes a high-grade product from low-grade ore. Its *raison d'être* is its cheapness of treatment. The low cost of ore dressing compensates for its high losses. Any improvement in such a basic process vitally influences all the other subsequent processes, because it makes from low-grade ores high-grade products available for the latter processes. The perfection of the high-power crushing machinery has rendered the first work of mechanical segregation of the constituent minerals cheap and efficient. The improvement in "sizing" or separating these minerals in various classes, according to size, has brought about increased efficiency of the old-time apparatus, such as the Hartz jig and vanner. Next, the introduction of the shaking riffled table, such as the Wilfley, has worked a revolution in the concentrating mill by filling a long-felt want. All these improvements were simple developments along the old line of separation by making use of the fact that particles of minerals of different specific gravity and same size fall through water at different speeds. Concentrating experts are, however, using at least four different properties of minerals—magnetic permeability, electric conductivity, susceptibility to chemical action, and susceptibility to being "wetted" by oils. All these developments are a bit startling.

The magnetic separator developed for the iron business could not succeed in face of discoveries of the cheap Mesabi ores. But it has been intelligently applied to the zinc ores of the Franklin mine, New Jersey, and has greatly enhanced the money value of that large deposit. It has also been applied with success to the complex zinc ores of the Rocky Mountains and Wisconsin. Copper ores are often well treated by it, and, in fact, it has found a wide range of application in the whole field. Its rival and coöperator, the electrostatic generator, has been made also a practical machine. The "Blake-Morscher" electrostatic separator is used in commercial work in large mills in Denver, Park City, Butte and Wisconsin. It is a distinct success. Very favorable reports are heard of another electrostatic separator, the "Dolbear," developed in Boston, which is a bold innovation, in using the high-tension transformer instead of the electrostatic generator for the exciter of the separator. This machine has been evolved from the laboratory stage by the aid of some distinguished electrical engineers. We hope to give it in the future the prominence it deserves in our columns. The Elmore oil separator has a peculiar sphere of action in the ore-dressing field. Some minerals are easily "wet" by crude oils and some are less easily so affected. Such so-called "grease processes," of course, can be nicely applied where everything else fails. One of the original grease processes was discovered at the Kimberly diamond mines by a boy, who, while eating his lunch at work, accidentally dropped a piece of bread, "butter-side down," on the traveling belt on which the diamond-bearing clay passed. He found that where the grease was, there also were the diamonds. For some time this youngster enjoyed "easy money," and was finally only persuaded for a consideration to tell his secret.

Another prominent development is the "acid-flotation process," used on the complex sulphide ores of the Broken Hill, Australia. These enormous deposits of zinc-lead ore have made large tailing piles of great potential value. The "Delprat-Potter" and the "Bradford" process have solved the question of treating them, and the zinc ore market of the world is affected by this discovery. The treatment of these complex sulphide ores is thus being worked out by a complex method, it would seem. Almost all these methods are really due to original scientific discoveries. Science, simple in itself, as all philosophy is simple, can be infinitely expanded. Its expansion to complexity is natural and beautiful. This brief and imperfect review of human achievement in this line of complexity leads us to believe that the next decade will see more advance in this line. Such advance is but an evidence of the spirit of the age. This is an age of metallurgical industry, and we feel proud of our title as indicative of just the spirit which it is our sincere desire to represent.

Editorial Notice.

Owing to the considerable space allotted to the report of the Bethlehem meeting of the American Electrochemical Society, this issue is somewhat circumscribed in the scope of its contents, and portions of our regular departments had to be reserved for our next issue. The second part of Dr. Keller's paper on labor-saving devices in laboratories and the conclusion of Mr. Ladoff's article on the electrochemistry of the electric arc, will also be published in that issue.

Notes on Electrochemistry and Metallurgy in Great Britain.

(From Our Special Correspondent.)

Primitive Tin Mining of To-day.

While exploring a quiet cove, within 10 miles of Land's End, with a view to finding a place for a bath, on a recent holiday, I stumbled upon an isolated metallurgical industry of a very primitive order. A solitary man—the industry had been carried on by him on that spot for some seven years—was to be seen collecting in buckets the darker patches of black-looking sand and carrying these to little heaps. Near by, when tin was far cheaper and metallurgical methods infinitely crude, the refuse from a neighboring mine had for some years been dumped over the cliff into the sea. As a result the sand contained innumerable black nodules of tin. This solitary worker collected the blacker looking sand, afterwards washing it in baffle boxes through which a stream of water ran. The lighter particles of silica and crushed sea shells were carried away, leaving a residue of almost pure tin. By working day in and day out in all states of weather, this man managed to recover about a ton of tin per annum, for which he received about £90 by selling it in small quantities at a time. Industrially the matter has no immediate financial importance, but looked at in another light, it is curious to think of the metallurgical wealth of the world wasted by the present and earlier generations. Perhaps our successors may find much wealth in the scrap and rubbish heaps of to-day which are now the resort of those low down in the industrial scale. To-day the Chinaman forages among abandoned workings on Californian gold fields, and I have seen on the Durham coast, when a good steam coal (that is, good for Durham) was fetching $3 per ton, women and girls picking lumps of coal from a beach upon which the sea had returned a portion of the colliery refuse dumped into it a few hours before. Perhaps, among the now despised refuse of foundries, furnaces, mine tailings and rubbish heaps of alkali works, the scientist of the utilitarian to-morrow may peg out claims for remunerative working. Ideally there should be no waste, but the practice of to-day is another tale. It is for the scientist to begin to consider these things. Of all wealth which the world possesses, mineral wealth is the most ephemeral, for though the fellaheen of to-day grows his corn, gathers his dates, and makes his mud bricks as it was in the days of the Pharaohs, the sources of the world's then mineral wealth are merely the subject of enlightened conjecture.

The Recent Liége Congress.

Immediately after mailing my last letter I received the complete set of papers read at the Congress, and was surprised to find that a paper by Prof. Le Chatellier had not received extended notice in the British technical press. Every metallurgist knows how to prepare metallic specimens for microphotographic purposes. M. Le Chatellier, therefore, claims for his paper nothing new in the way of discovery, but a description of carefully conceived methods of preparing the specimens, so simple and exact that by following them excellent specimens may be prepared by elementary student.

Obtaining a Flat, Smooth Surface.—Specimens detached from the main block with a saw or hammer are best treated with an emery wheel. To avoid bringing about hardening of the specimen by the percussion use a high-speed wheel, and do not press hard. In the case of tempered steel, a slow speed and frequent mostening are necessary to avoid letting it down. In spite of precaution, a thin external skin or layer of the specimen is altered in condition by the friction of the wheel. This must be removed by emery paper, applied by hand. To make the process more rapid, moisten the paper with turpentine. To delay it, grease the paper, which also causes it to scratch less. Angles should be rounded off on the emery wheel to facilitate polishing.

Finishing.—Emery paste should be prepared by the operator, the emery of commerce containing grains of unequal size. Proceed as follows:

First Application of Emery.—Emery powder is used, and that which is passed through a sieve of 2600 meshes to the square centimeter and is held on a sieve of 4900 meshes to the square centimeter is retained. The grains thus selected will be 1/10 millimeter in diameter.

Second Application.—Emery paste. The finest commercial emery paste is washed by a stream of water ascending, and the portions taken along by the stream are used. The author uses an apparatus thus described: A glass tube 40 cm. long to 50 mm. in diameter is closed at either end with corks; the lower cork is pierced by the pipe of a glass funnel, of which the trumpet is inside the large tube. The mouth of the funnel is covered by a piece of metal gauze. The water enters through this funnel under pressure. A short distance above the funnel is placed a double cone of tinfoil, about three times as long as the diameter of the base, and cutting it in half, thus obtaining a segment of a cone, or funnel of wide mouth, and a smaller cone. The segment is placed in the large tube, smaller end downwards, and the small cone inside it point upwards, in such a manner that the plane of the base is in the same plane as that of the upper end of the segment. A long glass tube, about 5 mm. in diameter, is led through the upper cork of the large tube to just above the point of the small cone, and leads to it the emery. The emery slides over the cone into the ascending water. The finest grains are carried to the upper cork, and find egress through a tube shaped like the letter S, the top hook of the letter entering the bottle. The curve of the lower hook of this tube has a small hole in it, through which the water and emery in it passes into a receiving bottle. The end of the hook should be carried up some distance above the large tube. The level of the liquid in this ascending pipe gives a measure of the pressure of the water employed, and this pipe is marked after experiment, to enable the speed of different washings to be readily adjusted by raising or lowering the water supply bottle. The tube through which the emery is let into the washing or large tube should be from 3 to 5 mm. in diameter (inside). The washings are received in a bottle having two apertures, one leading to the side to bring the water in at an angle, and the other the ordinary orifice at the top, with a tube to carry off the water.

Third Application, Alumina.—M. Le Chatellier refers to a former description of the method of preparing alumina of ammoniacal alum as a base, and washing by the process used by Schlesing for separating out clay in earths. He recommends that the alumina be beaten in a mortar to separate the grains. It is then washed with water acidulated with one in 1000 of nitric acid, then with distilled water, and finally with water to which 1 or 2 cubic centimeters of concentrated ammonia per litre has been added. Generally, after a certain number of washings with the water acidulated with nitric acid, the alumin is found to remain in suspension, *i. e.*, it is sufficiently washed. To obtain the alumina use a large glass tube, somewhat similar to that of Schlesing. The tube should be 50 cm. long, and holding 1 litre. The lower end is conical, with a slope of 3 to 1. In it is an opening 3 mm. in diameter at the most, so that air may not enter. The top of the tube is connected to a smaller pipe, having a copper top and orifice for sucking.

The mixture of alumina and washing liquid is drawn up into this tube by exhausting the air, and the top is then closed. The alumina falls to the bottom and the top is then opened slightly to let it drop, at the rate of one drop per 10 or 20 seconds. The grains emitted during the first quarter of an hour are coarse, and up to 3 hours from opening the top they are unsuited for the highest class of finishing, though useful for rougher work. The best material is precipitated after 3 hours dropping. The tap may then be opened full after 3 hours and the rest taken out. A spray bottle is the best vessel to keep the mixture in.

Fourth—Cloths, etc., for Using with the Pastes.—M. Le Chatellier uses very fine flannel stretched on glass and clamped down at the edges with slips of wood. To cause the powders to adhere to the stuff he uses a mixture of oleix soap and manganate of soda, with a little glycerine or black soap. The soap solution should be filtered to extract grains. The soap should be of a kind remaining soft. For the final polish with alumina, revolving discs may be used, either of wood, covered with fine cloth, or felt.

By these methods specimens 15 mm. in diameter may be surfaced and polished in a quarter of an hour. The best size for specimens is 15 mm. diameter by 10 mm. thick.

Treatment of Alloys with Reagents.—Numerous experiments have been carried on in M. Le Chatellier's laboratory for the separation of alloys. M. Igewsky has experimented with coloring matters, and finds that only the nitrogen derivatives gave interesting results.

The use of a 5 per cent solution of picric acid in alcohol has become general. M. Kourbatoff has investigated alcohol and advises a 4 per cent solution of nitric acid in amylic alcohol as to the picric acid. It is also useful in combinations to color out the constituents of steel, for instance: Four per cent solution nitric acid in ordinary alcohol, 1 pint; saturated solution of nitrophenol, 3 parts; with color out, troostite and like constituents, leaving the rest uncolored.

M. Le Chatellier finds that cementite may be easily colored by the addition of boiling alkaline solutions with the addition of any oxydizing agent, especially picric acid.

A 25 per cent solution of caustic soda, plus a 2 per cent of picric acid, warmed to 100° C. in a water bath is recommended. This reagent does not color deeply, and the colored surface is easily removed by polishing slightly.

The Microscope.—M. Le Chatellier has previously recommended the use of the mercury vapor lamp for microphotography, the ultra-violet rays being excluded by a bath of acid sulphate of quinine. The lamp is, however, very fragile. He prefers the use of a Nernst lamp with two filaments, taking 1 amp. per filament. The lamp should be adjusted so that the filaments superpose their rays one on the other. A little more clearness of the image is gained by using isochromatic plates and interposing a 1 per cent solution of picric acid and 1 cm. in thickness, but with Zeiss apochromatic objectives and ordinary plates, the result is good enough for ordinary work.

He recommends the use of diaphragms placed before the source of light, directly under the objective, as near as possible to the back lens. Various diaphragms may be mounted on a circular plate for convenient use; 5 mm. is a suitable diameter for the orifice. By using them slightly out of the direct line irregularities of surface on the specimen may be accentuated.

M. Pellin has adapted to his microscope an adjustable table for the specimen, controlled by thumb screws as used on ordinary microscopes. The beam of light is now reflected by a prism which can be moved so as to direct it either on the eyehole or the lens.

With the mercury lamp the time of exposure is 15 minutes with the horizontal photographic chamber, three times as long with the vertical chamber. With the two-filament Nernst lamp the time of exposure is 2 and 5 minutes for brilliant steels. It is 10 to 30 minutes for non-reflecting bodies, such as glasses, slag, etc.

Zeiss' apparatus has facilities for the adjustment of the two lenses, which permit of a very flat image being obtained.

Sterilization of Sewage Effluents.

In my letter, published in the January issue of Electrochemical and Metallurgical Industry, I dealt with the general outlook concerning the use of sodium hypochlorite, prepared electrolytically as a sterilizing agent. I am not aware that any new works are in operation since I wrote. A paper read by Dr. S. Rideal before the Royal Sanitary Institute, discusses the properties of various agents, and deals satisfactorily with pretty well every aspect save that of the actual cost. The sparing solubility of ozone was pointed out, and its practical limitation as a finishing agent for previously filtered canal or river water emphasized. Dr. Rideal expressed his own preference for chlorine and its oxides as easily soluble, generally more manageable and as powerful as ozone, when "the main part of the organic matter has been removed or reduced." "Although these processes," Dr. Rideal says, "are commonly called chlorine methods, it must be remembered that the purification is mainly effected by oxygen liberated in an active state through the action of chlorine on water, the proportion of the element that can serve in this way being termed 'available chlorine,' as distinguished from that present in chlorides, such as common salt, which is almost inert. The amount of oxygen liberated by chlorine itself, as in the addition of chlorine water or the gas, is doubled when it is present as hypochlorite, and my experiments indicated that the oxygen value of the electrolytic solution, containing chloride oxides and other compounds, was still greater."

The arsenious acid test cannot be regarded as wholly satisfactory for the testing of hypochlorite solutions, as Dr. Rideal states that in a large number of trials in which the sterilizing medium was added in varying amounts, showed that nearly a constant proportion, about 60 per cent, of the available chlorine was taken up almost immediately, while about 40 per cent remained and declined more gradually, pointing distinctly to the chlorine existing in the reagent in two forms, one which acts at once on readily putrescible matter, the other remaining to attack bacteria and resistant organic substances.

As regards the mineral acids, their employment in a dilute form kills most organisms, and 4 grammes of sulphuric acid per gallon will sterilize typhoid bacilli in the drainage from an isolation hospital.

The first place is thus given to hypochlorite solutions, in regard to the preparation of which Dr. Rideal is silent. The cost depends partly upon the electrochemical efficiency, partly upon the life of the anodes, partly upon the stability of the solution produced, and partly upon the ratio which exists between the amount of salt which is converted into sodium hypochlorite, and the far larger quantity which remains unconverted and is wasted.

One important factor is due to Dr. Rideal's investigations, and that is a preliminary scientific investigation of the amount of hypochlorite required, which is that *the amount of oxygen absorbed in 5 minutes, multiplied by 1.7, gives the amount of available chlorine required in parts per* 100,000.

Market Quotations During August.

Some slight changes, mostly of the downward order, have to be chronicled concerning the chemical trade during the month. Bleaching powder, 35 per cent, is 2s. 6d. cheaper at £4.7.6. Sulphate of ammonia is also cheaper at £12.11.3, as against £12.17.6, while the coal tar products have receded fractionally. Borax is now fetching £13 per ton, an increase of £1; copper sulphate has risen to £22 per ton. Shellac is steady at £9 per cwt.

With regard to metals, copper reached, on Aug. 26, the high price of £72.10; the demand being in excess of the supply, prices are being maintained at a high level. Tin has been somewhat irregular, reaching £152 per ton on Aug 2, falling to £148.15 within a week, but, subsequently mounting to £152.10. There has since been another reaction in the price, but owing to the scarcity of supplies a further rise is expected. Lead has closed at £14.15; Cleveland pig iron has risen to £2.8.4½, while the growing demand for all classes of material indicates a marked improvement in the steel and iron trade.

London, Sept. 5.

CORRESPONDENCE.

Lead Poisoning.

To the Editor of Electrochemical and Metallurgical Industry.

Sir.—The International Labor Office of Basel (Switzerland) have invited an international prize competition for the prevention of lead poisoning, and are desirous of making this competition known. They have asked me to assist them in giving their proposition a widespread publication in the United States.

Knowing that your paper will be an excellent means for reaching a great many persons interested in the above competition, I trust that you will publish the conditions of the prize competition of which I beg to enclose you a copy herewith.

To those of your readers who are not acquainted with the aims of the International Labor Office the following data may be of interest:

The International Labor Office, which was organized by the International Association for Labor Legislation, began its work in May, 1901. This association was founded by persons advocating the legal protection of workmen, among whom were prominent statesmen and scholars of Europe. Sections of the association have been formed in Austria, Belgium, France, Germany, Great Britain, Hungary, Italy, Netherlands and Switzerland.

The International Labor Office is subsidized by the governments of the following countries: United States, Belgium, France, Germany, Hungary, Italy, Luxemburg, Netherlands, Sweden and Switzerland.

According to its rules the International Labor Office has to serve as a scientific organ, and it has been entrusted with the task of preparing and publishing a bulletin, containing an account of all the new laws concerning the hygiene and safety of the working classes.

Due to the efforts of the International Association the Swiss Federal Council called out an international labor conference, which took place in Berne in May, 1905; its special object was to discuss the abolishment of the use of yellow phosphorus in the manufacture of matches and the prohibition of night work for women in the industry. F. G. Hartmann.

New York City.

[According to the printed rules of the competition the following prizes are offered:

One prize of $1.200 for the best treatise on the prevention of lead poisoning in the operation of mining and milling lead ores or ores containing lead. One prize of $2.400 for the best treatise on the prevention of lead poisoning in smelting and refining works. Two prizes, viz. one first of $600, one second of $360, for the best treatises on the prevention of lead poisoning in the chemical application of lead, as in white lead works, manufacture of other lead paints, of electric accumulators (storage batteries), etc. Four prizes, viz.: one of $360, one of $240, two of $180 each, for the best treatises on the prevention of lead poisoning in the trades of house, ship, coach-painting, interior decoration, varnishing and the like. Four prizes, viz one of $360, one of $240, two of $180 each, for the best treatises on the prevention of lead poisoning in those trades where raw and manufactured lead are consumed or handled on a large scale, as in type foundries and printing offices.

Each treatise to contain a systematic review of the special causes giving rise to lead poisoning, in conjunction with a description of the various processes of manufacture, pointing out the dangers occasioned at every phase of procedure, including handling and transportation.

Reference to be made also to the causes due to working at places in which a prolonged occupation is liable to affect the health, to want of cleanliness, lack of proper guidance and instructions, carelessness, poor and inadequate food, irrational way of living and unhealthy dwellings of the workmen. In connection with the statement of the causes of lead poisoning, measures for their prevention are to be proposed.

Substantial evidence should be given for the proposed preventives as regards their technical, hygienical and economical feasibility.

The comparative injuriousness of the different phases of manufacture, as well as of the conditions referred to above, must be demonstrated as far as possible by a properly graduated classification of the risk of lead poisoning, from the highest to the lowest degree.

In proposing new installations or alternations of the existing mode of operation, the probable increase or decrease of expenses incurred by the change should be approximately stated, e g., in proposing mechanical appliances to supersede manual labor the primary cost of these, as well as the increase of working expenses by depreciation and interest of the capital invested would have to appear, as well as the saving of wages or other factors of ecónomy, viz.: the avoidance of fluctuations and changes in the working staff and, in consequence, the maintenance of a properly trained crew with a corresponding improvement of their capacity.

It is desired that the treatises should promote the development of the present provisions of the various States by outlining or drafting legal or administrative regulations or suggestions to the proper authorities, aiming at the realization of the preventive measures proposed by the competitors. Short precepts against the danger of poisoning might also be given preferably in a form suited for posting in workshops, building grounds, etc.

A compilation of the existing government regulations may be found in the volume *Gesundheitsgefahrliche Industrien*, edited by Prof. Stephen Bauer (Jena, G. Fischer, 1903), and in the *Bulletin des Internationalen Arbeitsamtes*, 1901-1904; which publications can probably be had at the libraries in large cities.

A clear and concise summary of the measures proposed by the competitors is to be given at the end of the treatise.

The papers may be written either in English, French or German. Already printed books cannot be taken into consideration by the jury. The ready manuscripts must be put in an envelope bearing only a motto, and lodged with the International Labor Office at Basel on or before Dec. 31, 1905.

The full name and address of the author should be stated under a separate sealed cover bearing on its outside the same motto as above.

Contributions arriving later than Dec. 31, 1905, will be excluded from the competition.

By the award of a prize the International Labor Office is entitled to publish a treatise, leaving, however, its author the exclusive property; the office may also concede to the authors the right of publishing their treatises. Manuscripts which are not secured for publication will be returned to their authors.

In case the jury should not see fit to award the total sum of $6.480 reserved for the prizes, the remainder shall be employed upon recommendation of the jury by the "International Association for Labor Legislation," for other purposes concerning the hygienic welfare of the working classes.

All letters, inquiries and other matters pertaining to this competition are to be addressed to the International Labor Office at Basel, Switzerland.]

BETHLEHEM MEETING OF THE AMERICAN ELECTROCHEMICAL SOCIETY.

The eighth general meeting of the American Electrochemical Society was held in Bethlehem, Pa., from Sept. 18 to 20, and was very greatly enjoyed by those who attended it, although the inclement weather spoiled to some extent the elaborate preparations which had been made for excursions around Bethlehem.

On the forenoon of Sept. 18 a directors' meeting was held at the Eagle Hotel, at which some important decisions were reached, as was announced by Dr. W. D. Bancroft in his speech at the banquet. In the first place, the intention is to hold in future only one meeting a year, but it has not yet been decided whether the spring or the autumn meeting is to be retained. On the other hand, this decision does not prevent holding a second meeting jointly with one of the older national engineering societies, as the American Chemical Society, the American Institute of Electrical Engineers and the American Institute of Mining Engineers. Dr. Bancroft transmitted to the society an invitation from Cornell University to hold the next meeting in Ithaca, in May, 1906.

BETHLEHEM MEETING OF THE AMERICAN ELECTROCHEMICAL SOCIETY.

An important decision reached at the directors' meeting was that the society will use some of its funds—probably in the beginning $100 a year—to encourage research work. For this purpose a circular letter will soon be mailed to all members to make suggestions as to subjects which need research. It is hoped that a valuable list of suggestions will thereby be obtained.

MONDAY SESSION.

The first session was held on Monday afternoon in the chemical laboratory of Lehigh University. Dr. J. W. Richards, as chairman of the local committee, welcomed the members to Bethlehem, and introduced President Drinker, of Lehigh University, who in a felicitous little speech expressed his pleasure at the fact that his first public official act as president of the university was to welcome the members of the society to the freedom of Lehigh. Dr. W. D. Bancroft, as president of the society, pointed in his reply to the happy combination of educative and commercial activity in Bethlehem, as represented by Lehigh University on one side and by the steel, zinc and cement works on the other side. Bethlehem had provided the American Electrochemical Society with its first president, Dr. J. W. Richards.

Colloids.

A brief note by Prof. W. S. **Franklin** and Mr. L. A. **Freudenberger** was presented by Mr. Freudenberger. The leading idea of their experimental research was to get some distinct idea of the nature of a colloidal solution, by spinning a tube containing the colloid at a high speed. The hope of the authors was that the centrifugal force would produce a new condition of equilibrium within the colloidal solution; this new equilibrium would then be a function of the speed at which the tube is revolved, and it would be possible to study, by optical means, the new condition of the colloid as changing from point to point when getting away further from the axle of rotation. This would then have given some idea of the size of the particles, etc. However, these hopes were not fulfilled. It is true that if the speed of revolution is high enough, a new equilibrium results, but this is simply that the particles are first thrown down at the end of the tube and this is followed by a sort of growth of the precipitate along the tube. Nothing whatever happens until the centrifugal force is great enough to bring about at the terminal of the tube the precipitation of a nucleus, which then results in further precipitation. The paper was briefly discussed by Dr. J. W. Richards and Dr. W. S. Franklin.

The Chemistry of Electrochemistry.

An experimental lecture of great interest was presented by Dr. **Wilder D. Bancroft**. The object was not to propose a new theory but rather to present a point of view which has been generally neglected in the development of electrochemistry. The fundamental idea of Prof. Bancroft is that the action of the current in an electrolytic cell is to bring and discharge the ions at the electrodes. What then happens—everything that happens over and above the discharge of the ions—is a question of pure chemistry, and should be looked upon as a purely chemical problem. To understand an electrochemical reaction it is of fundamental importance to supplement the investigation by parallel chemical work.

"Let us consider first the reactions at the cathode. If we pass a current through a slightly acidified solution of copper sulphate we get the precipitation of metallic copper. This is merely the normal action of the current setting free the cation, copper, at the cathode. If we acidify the solution still further, adding sulphuric acid in one case and nitric acid in another, we find that a given amount of nitric acid will prevent the precipitation of copper, while much more sulphuric acid must be added to cause the same effect. The reason of this is clear when we consider the chemistry involved. Nitric acid attacks copper readily and sulphuric acid does not. While this is perfectly obvious when stated in this way, it has not been obvious enough in the past to induce anyone to test quantitatively the relation between the decreased current efficiency in a nitric acid solution and the rate of corrosion by that particular solution. We know that the current efficiency increases with current density, which is just what we should expect if the chemical remains constant; but more than that we do not know."

WILDER D. BANCROFT.
Pres. American Electrochemical Soc'y.

The difficulty of depositing sodium from an aqueous solution is due to the rapid chemical action of water on any sodium that may be deposited. To precipitate sodium from the fused salt the temperature must be high enough to have fusion, but must not be raised so high that the sodium would react with the bath. Sodium amalgam reacts with water much less rapidly than does metallic sodium; therefore, it is possible to form sodium amalgam electrolytically by using a mercury cathode.

If a very high current density is used in an aqueous caustic soda solution with a lead cathode, a black cloud is given off from the cathode. The fact is that a sodium-lead alloy is formed which breaks down practically at once, scattering pulverulent lead through the solution in a black cloud. That this is the correct explanation was proven by Haber, who supplemented this electrochemical experiment by chemical research, and showed that a sodium-lead alloy, prepared chemically, reacts exactly in the same way when put into water.

"If we electrolyze a solution of copper sulphate plus zinc sulphate we get a precipitation of copper. The reason why no zinc can precipitate as such becomes clear when we dip a rod of zinc into a copper sulphate solution. The copper precipitates and some zinc dissolves. When no other disturbing factors occur, we find that the more rapidly one metal precipitates another from solution, the less readily will that metal be precipitated electrolytically from a solution containing the two metals. Thus more care has to be taken in separating silver from copper than in separating copper from zinc or cadmium. There are some apparent exceptions to this, but a study of the chemistry involved clears up matters. In sulphate, nitrate or acetate solutions, massive aluminium or massive nickel will not precipitate copper. Under certain conditions iron will not precipitate copper. In all of these solutions copper precipitates electrolytically before these other metals. If we examine into the matter we find that while nickel, for instance, does not precipitate copper, neither does copper precipitate nickel. We are, therefore, not dealing with a reversible reaction, and the presumption is that the conditions are not as stated. As a matter of fact, we find that the aluminium, the nickel and the iron are covered with a surface film, so that the metals do not come in direct contact with the copper sulphate solution at all. With the iron a mere scratch suffices to break the film and to cause the precipitation of copper."

With respect to the electrolytic precipitation of alloys it is very important to study the chemical equilibrium between the alloy and the solution.

"The electrolysis of a solution of the sulphates of zinc and copper has been discussed as a separation of copper from zinc. We may also consider the electrolysis of an acidified copper sulphate solution as a separation of copper from hydrogen. Since the copper precipitates before the hydrogen we should expect this to mean that hydrogen precipitates copper from solution. At first sight this seems not to be true. One can bubble hydrogen through a copper sulphate solution forever without precipitating any copper. We are not doing our chemical experiment under the same conditions that we used in the electrochemical experiment. In the latter the copper was precipitated at an electrode. If we place a piece of metal, preferably platinum, in a copper sulphate solution and pass in hydrogen gas we shall get a reduction by the hydrogen to metallic copper. It is this apparent inertness of hydrogen gas which has led to the conception of nascent hydrogen. In all cases where we have effects due to so-called nascent hydrogen, we have a metal present, and the effect is either due to the metal or can be duplicated by hydrogen gas in contact with a metal."

With respect to the "excess voltage" which is required to set hydrogen free at cathodes of certain metals over that required on other cathodes, Dr. Bancroft made the following suggestion:

"Zinc is a metal giving a particularly high excess voltage. It is owing to this that it is comparatively easy to precipitate zinc or cadmium from a solution which is perceptibly acid. This may be connected with the fact that pure zinc does not dissolve readily in acid solutions; but beyond that our knowledge does not go as yet.

"If we electrolyze the salt of a metal which can have a varying valency we may meet special difficulties, but they all become intelligible when we consider the chemistry of the solutions. If we start with a solution of ferric sulphate or chloride we do not get metallic iron precipitated, because iron and ferric sulphate or chloride react to form the ferrous salt. Since the equilibrium is reached only when practically all the ferric salt has disappeared, it is necessary to reduce the ferric salt practically completely to ferrous salt before metallic iron can be obtained."

For zero current density the electrochemical and the chemical equilibrium must be the same for all reversible reactions. For any finite current density we have to consider reaction velocities. In general we shall diverge farther from the chemical equilibrium the higher the current density. This is merely another way of saying that the so-called primary reaction is favored by low-current density.

Dr. Bancroft then discussed in the same way a great many reactions at the anode, especially the different products which can be obtained in the electrolysis of chloride solutions, etc. The success of the investigations of Foerster and his collaborators in clearing up these problems has been due to the fact that they studied the chemical and electrochemical phenomena simultaneously.

With respect to sulphate solutions Dr. Bancroft referred to the following point, which emphasizes our ignorance:

"In Germany the oxygen evolved during the electrolysis of sulphuric acid is believed to come from the water, according to the equation

$$SO_4 + H_2O = H_2SO_4 + O.$$

In France the oxygen is supposed to come from the sulphate radical itself, according to the equations

$$SO_4 = SO_3 + O$$
$$SO_3 + H_2O = H_2SO_4.$$

So far as I know, no one has ever brought forward any proof which would enable us to decide between these two views. We know that solid ammonium or potassium persulphate breaks down into sulphate, sulphur trioxide and oxygen, which is an argument in favor of the French point of view. The formation of hydrogen peroxide from persulphuric acid falls more in line if we consider the oxygen as coming from the sulphate radical. On the other hand, the formation of persulphates seems to be an argument on the other side. Whatever the final decision may be, I wish now merely to emphasize the fact that this is a problem which has not been faced in the past, largely because we have kept our chemistry and our electrochemistry more or less apart."

Finally, Dr. Bancroft referred to the following point:

"It has been found that electrolytic reduction takes place more readily at certain cathodes than at others, zinc, copper and tin being the best in certain cases, lead in others. It appears to be a specific action of the cathode in each case. In the Boehringer patents for reducing nitrobenzene it was stated that one could substitute a platinum cathode for a tin or copper cathode, provided one added a tin or a copper salt to the solution. It looks, therefore, as though we were really dealing with the chemical action of the pulverulent metal on the substance to be reduced, and that the current merely regenerated the reducing agent. This brings out again the importance of studying the electrochemical phenomena with special reference to the chemical phenomena. Recent work of Tafel shows that some of the mysterious changes in cathode efficiencies are due to the precipitation of minute quantities of platinum, and we have long known about the disastrous effects of platinum on storage cells."

Dr. Bancroft's general conclusion is that we get a clearer view of the subject as a whole if we consider the chemistry of electrochemistry, and that this point of view is valuable in showing us concrete instances of research work which ought to be done.

Dr. Bancroft's paper, which was illustrated by a great many experiments, was discussed in a lively way by Mr. Carl Hering, Mr. Anson G. Betts, Dr. J. W. Richards and Prof. C. F. Burgess. Mr. Hering read a communicated discussion of Mr. C. J. Reed, who did not agree with the point of view of Dr. Bancroft, and maintained that the only right way to look at the question which reaction will happen in an electrolytic cell, is to find the energies of reaction corresponding to the different reactions which would be a priori possible. The value of the energy of reaction (or of the voltage which is proportional to it) decides whether a reaction will go on. This rule was applied by Mr. Reed to a discussion of several examples from Dr. Bancroft's paper. Prof. Burgess thought that the value of Dr. Bancroft's point of view is just the absence of any electrochemical theory. His point of view forces us to get right down to facts.

Metal Deposition.

A very suggestive paper by Mr. Anson G. Betts was then presented. The object of this paper was to find the underlying principles which decide whether an electrolytic metal deposit will be good or bad. Mr. Betts first took up several suggestions, which have been made to explain in special cases the specific effect of additions, etc., on the quality of a deposit. In this way he discussed the effect of colloids, the effect of reducing agents, the influence of the strength of an acid and of the alkalinity of a base, etc. He described several experiments in which he had tested his former suggestion, that the e. m. f's. set up on the ridges and in the hollows of an electrodeposit, due to the difference of concentration at the two places, might decide the quality of a deposit.

Mr. Betts then referred to the assumption that differences in crystallization habits of the metals might be the causes of different forms of cathode growths. But he pointed out that we are not able to separate the metals in two groups, one of which crystallizes according to one system and gives, perhaps, good deposits, while another group crystallizes in other habits and gives non-solid deposits. Mr. Betts, therefore, concluded that we cannot assign to the crystallization habit of a metal the controlling force of the character of deposits made with it. Moreover, there is no evidence that many of our so-called crystalline deposits are crystalline at all. "Just as a rough pile of bricks may look like crystals, while they are alike in shape, just because an agency has been at work making clay into similar shapes, so some agency, for example, changes in concentration, may be at work in shaping a metal deposit of similarly shaped particles. We have all seen streaked electrodeposits with vertical streaks traversing the surface. If we also happened to have some other agency than the rising current at the cathode at work, only in a direction at right angles thereto and parallel to the surface, the surface would be laid off in little rectangles, which might be assumed, a few times at least, to be crystals." There is in many cases a striking similarity in appearance of electrolytic deposits with rocky cliffs and quartz.

Mr. Betts then discussed the counter e. m. f. of polarization under varying conditions in electrolytic cells. For instance, in a lead fluosilicate solution, containing 5.2 grams of lead and 3.5 grams of fluosilicic acid per 100 cc., the polarization of e. m. f. differs if an addition of gelatine is employed or not (the addition giving a smooth, hard deposit). The increase of potential at the cathode in this case is about 0.007 volt when gelatine is added. Mr. Betts put up the question, what becomes of the energy which this increased voltage represents, and which amounts to about 1.7 kw-hours per ton of lead produced.

Before he answered this question, Mr. Betts gave a table, in which he divided the metals into two great classes: First, those which yield solid deposits with appropriate solutions, and, second, those which produce loose deposits. From this table he concluded that the one distinguishing property between these two groups of metals is their hardness, and he gave the general rule that the harder a metal the better the deposit. An approximate value of the hardness may be found by the rule of Bottone, according to which the hardness is proportional to the specific gravity divided by the atomic weight. Mr. Betts then discussed the influence of valency in the phenomena of colloids, and expressed the belief that in electrolytic deposition the effect of the basic radical will manifest itself in causing more or less firm cohesion, according to its valency.

The general theory which he, therefore, proposes is that a hard metal will separate in solid, compact form, and a soft metal will deposit in loose crystalline form. When the hardness is intermediate the valency of the metal becomes a determining factor, high valencies producing solid deposits and low valencies loose deposits. When we come to that part of the scale of atomic density (specific gravity divided by atomic weight) where it is questionable whether a deposit may be solid or not, it is quite possible that factors of less importance will assume a more important rôle.

If a metal is not hard of itself, it is possible to deposit it in smoother form if it can be made harder. This is, according to the author's view, the case in lead depositing, as the result of the addition of gelatine and the excess of consumed energy, amounting to 1.7 kw-hours per ton of lead, as stated above, is consumed in hardening the lead.

The author then discussed the rôle which surface tension plays. He proposed that the greater the hardness of a metal, the greater is its surface tension, and as a consequence in depositing a hard metal we have much more potent forces at work keeping the surface smooth than in depositing soft metal. When surface tension comes strongly into play, the growth will not be by the application of thin layers of crystal masses, with projecting points forming the foundation of more points, but by the application of layer after layer of uniform thickness and non-crystalline structure, so that the effect of high-surface tension is to smoothen the deposit.

Mr. Betts then suggested another general rule, that is, that the greater the hardness or the cohesion of the material, the less its solubility. He also expressed a belief that there is some relationship between surface tension and valency. In passing over to the surface tension of metals, as influenced by solvents and solutes, Mr. Betts arrived at the rule that if the surface tension of an immersed solid is greater in the solution than in the solvent it follows as a result that dissolved substances in the solution will harden, and consequently smoothen an electrode deposit.

This is about the gist of Mr. Betts' long and interesting paper. For the great many details contained in the paper the reader must be referred to the full paper, which will be published in the Transactions of the Society. The paper was discussed at some length by Messrs. Burgess, Franklin and Richards. Prof. Burgess showed some micrographs of electrochemically-deposited iron. He stated that the majority of electrolytic deposits show crystalline structure. Several speakers doubted whether some of the generalizations of Mr. Betts, for instance, on the relation between hardness and solubility, could be maintained in such a broad form as stated.

In the evening the ladies' committee gave an informal reception at the house of Prof. and Mrs. J. W. Richards. This occasion was very greatly enjoyed. Afterwards there was a smoker at the club house of the Lincoln Republican Club, just opposite the old Moravian Church yard, which holds, besides many other graves of historical interest, that of the "Last of the Mohicans."

TUESDAY MEETING.

Aluminium as Reducing Agent.

The first paper of the Tuesday morning session was by Dr. **Oliver P. Watts**, on the use of aluminium as a reducing agent. In the absence of the author it was presented by Prof. C. F. Burgess. The success of the production of pure metals free from aluminium, by means of aluminium as reducing agent, is due to Dr. Hans Goldschmidt, who employs the aluminium in powdered form. The success depends upon the attainment of so high a temperature that the reaction takes place completely and that the products are perfectly fluid.

If iron is to be reduced from iron oxide by means of aluminium, the energy required for the reduction of the iron is so small compared with the heat developed by the oxidation of aluminium, that the iron is obtained in form of a highly overheated steel, suitable for welding and repairing purposes. In a similar way Dr. Goldschmidt has produced a long series of carbon-free metals.

However, for the reduction of the oxides of titanium and tungsten to the metals the required high temperature cannot be obtained in this way, although their alloys with iron can be made. The addition of the oxide of iron to the charge supplies the heat necessary for complete reduction and for the agglomeration of the resulting iron alloy.

In order to produce pure titanium and tungsten, the author combines the Goldschmidt method with electric arc heating. He has found that Goldschmidt charges can, by a slight change, be fired with safety in an electric furnace of the arc type. Powdered fluorspar, or cryolite, is added to the charge to reduce the velocity of reaction. Other inert powders can, no doubt, be substituted for these. The charge is placed in the cavity of a furnace with a magnesite lining, the cover put on, and an arc started just above the charge. The usual heating in the experiments of the author was for 5 minutes at 300 amps. and 70 volts, followed by 5 minutes more at 600 amps. and 80 volts. About fifty reductions were made by this method. The reaction was occasionally vigorous, but never explosive, although 350 grams of powdered aluminium were used in some cases. The metallic oxides used were those of iron, nickel, cobalt, chromium, manganese, molybdenum, titanium and tungsten. The amount of retarder was diminished in case of the less readily reducible oxides. In a few cases the reaction could have been brought about by the usual method of an igniter, but owing to the presence of the silica, boric anhydride, or both, most of the charges required the heat supplied by the electric arc to complete the reaction.

The paper was briefly discussed by Drs. Bancroft, Burgess and Richards.

Electric Steel Furnace.

A paper by Mr. **Gustave Gin** on his new process for the electric manufacture of steel was read, in the absence of the author, by Dr. J. W. Richards. The construction of the furnace may best be seen from the plan shown in Fig. 1. It comprises essentially: First, a crucible (1) for fusion and refining by oxidation (corresponding to the pig end in open-hearth practice); second, a compartment (2) for the reduction of oxide of iron and recarburization; third, a chamber (3) for the color test, to get the proper carbon test.

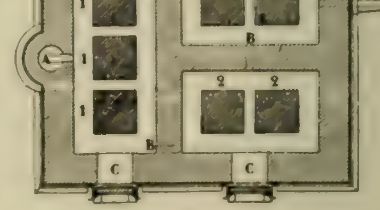

FIG. 1.—GIN STEEL FURNACE.

The electrodes of the compartment (1) are connected to one of the terminals from the source of electricity, and the electrodes of compartments (2 and 3) are both connected with the other terminal. The current passes from the electrodes to the metal through a layer of slag, which is the principal seat of the heating action of the current. The baths of metal communicate by openings B of sufficiently reduced section, in order that by reason of the Joule effect the metal may not remain solid.

To put the furnace into operation, melted iron or liquid steel is introduced through the orifice (A). It distributes itself through the three compartments on the floors of which scrap iron is strewn, the pieces being placed in connection with (B). Then arcs are created, and little by little the material is introduced.

The oxidizing slag of compartment (1) is composed of some mineral or slag rich in oxide of iron, in addition some lime if the metal to be refined is phosphatic. The slags of compartments (2 and 3) are neutral and little reducible by carbon.

Aluminates of lime and magnesia, made from a mixture of bauxite, with limestone or dolomite, give favorable results. The addition of calcium fluoride makes it more fusible and mobile.

By means of the method of circulation adopted, the refined metal in (1) passes next into compartment (2), where the reduction of the dissolved ferrous oxide and the recarburization of the metal takes place.

The author makes the point that the electrodes can be raised or lowered at will, and that this is important when the metal is tapped.

The paper brought forth considerable discussion, in which Messrs. J. W. Richards, W. D. Bancroft, C. J. Russell, W. Smith, S. S. Sadtler, A. G. Betts, C. F. Burgess, E. F. Roeber and B. MacNutt participated. The general opinion expressed was that Mr. Gin had apparently given up the fundamental idea of his former steel furnace, which resembled somewhat an incandescent lamp filament in greatly magnified proportions. He has now adopted the use of a slag in which the electrodes dip. This arrangement has proven very successful in the furnace of Heroult. Dr. Richards pointed out that Heroult is here evidently on sound ground, since with a very high temperature of his furnace he can employ more refractory slags, and can produce reactions which are impossible in an open-hearth furnace. He can thus very effectually remove sulphur and phosphorus.

It was doubted by various speakers whether the Gin furnace has ever been tried in practice, since its construction would involve various difficulties in operation. It has a large radiating surface. There will be some difficulty in keeping the contents of the three compartments separate. The design of the Gin furnace also seems to violate the metallurgical principle that one should not try to carry out essentially different processes in the same apparatus.

In the discussion it was also brought out that the first commercial application of the electric furnace for the manufacture of steel in the United States will be at the plant of the Halcomb Steel Co., of Syracuse, N. Y., the plant being for the production of 80 tons per day. The Heroult furnace will be used there.

Dr. Leonard Waldo then presented what was essentially a new paper, giving very interesting information on the development of the electric induction furnace in this country by Colby and Waldo, and in Sweden by Kjellin. Dr. Waldo pointed out that in electrothermic processes very heavy electric currents must be handled, and this involves considerable difficulties. In all of the old electrometallurgy it has been necessary to deliver current through electrodes; in this case the management of the electrode becomes extremely difficult. At some time or other the current through the electrode either passes through a mixed charge with varying resistance or makes an imperfect contact. When that happens, the current arcs; and when it arcs there is the practically uncontrollable temperature of 3850° C., and there results a corresponding impurity and a corresponding difficulty and a corresponding intense action at the single point of the arc. This is the reason why a furnace without electrodes suggested itself. Mr. Edward Colby built the first induction furnace in this country, and almost contemporaneously the engineering firm of Sprague, Duncan & Hutchinson independently experimented with one of their own construction; but, unfortunately, at that time the problem of the alternating-current dynamo and of the transformer were by no means in that shape as they are to-day. When Kjellin began later his work in Sweden he had the advantage of finding these alternating-current engineering problems solved.

Dr. Waldo then sketched the principle of the induction furnace as the special case of a transformer with a single secondary winding (the molten bath). It represents the ideal condition in metallurgy. Here we have the containing crucible with no electrodes; no contact with fuel in any shape; the efficiency is high; the melting takes place at any rate desired. This control of the rate of heating is a most important advantage. Everybody knows how long it takes after a crucible reaches a white heat before the final complete fusion takes place. The induction furnace permits adding quickly the energy units which make final minutes in the melt take the place of almost hours, when the high temperature is causing costly radiation and racking the mechanical stabiltiy of crucibles and furnace construction alike. At high temperatures approaching the melting point, it is possible to push the rate of temperature increase to 1° C. per minute, by adding 2 kw for every 100 pounds of steel; or to put it practically, the steel is raised the last 200° C. by adding 40 kw for 10 minutes.

If the furnace chamber is built of the proper material, its life is indefinite, since the heat is generated wholly within this chamber and not in excess of the fusion requirements. Dr. Waldo indicated the efficiency of the best induction furnaces by stating that they melt 1 ton of steel with an energy consumption of 500 kw-hours. Of course, in this type of furnace the coal efficiency rises very much over the coal efficiency in the crucible process.

Dr. Waldo spoke in a very interesting and humorous way on some observations peculiar to the operation of an induction furnace. The temperature in the metallic bath follows almost instantaneously any change of the energy supply in the primary circuit, apparently without any time lag. If the charge is made up of a series of steel rings to be melted, then when the furnace is started the rings will rise as though they wanted to get out of the furnace, and the furnace begins to sing, often with a beautiful, clear note, and the workingmen begin to look for their overalls and see what the chances of escape are.

Dr. Waldo remarked that a complete theory of the induction furnace and of the special features which distinguish it from the ordinary transformer would be very valuable.

Refining Silver.

A paper by Mr. Anson G. Betts on his electrolytic process for refining silver was then presented. As was already mentioned in Mr. Betts' article on page 145 of our April issue, the feature of the process consists in using salts of strong non-oxidizing acids, forming soluble silver salts. Dithionic acid gives fairly good results, but the best results are obtained with methyl—sulphuric acid. An interesting new feature mentioned in Mr. Betts' paper is that a small addition of carbon disulphide improves the quality of the deposit; otherwise the process, as described, is the same as mentioned in our April issue.

Parting of Bullion.

An interesting paper by Mr. F. D. Easterbrooks, of the Raritan Copper Works, of Perth Amboy, N. J., discussed the question of electrolytic versus sulphuric parting of bullion.

With the sulphuric acid method refined gold can be produced, ready for shipment, 24 to 30 hours after placing the bullion in the kettles, which is several days earlier than is possible with an electrolytic method. This is advantageous in those cases where the ore is high grade and interest charges have to be considered. A daily cut-off of the gold treated can be made, and thus a quick check obtained of output on input. There is also almost no silver tied up in plant, as in the electrolyte and cathodes of an electrolytic process.

The acid solution from the reducing tank containing the copper used to reduce the silver sulphate solution is disposed of in either of two ways: the bluestone crystallized on lead strips and the mother liquor concentrated and used again in the kettles, or, as is the case in copper refineries, added directly to the electrolyte, in which case the value of the acid added per ounce of silver parted, after deducting the power cost of depositing the copper in insoluble anode tanks, determines the amount at which an electrolytic parting process must operate below the sulphuric in order not to exceed the costs of the latter, since with an electrolytic parting process sulphuric acid must be bought especially for the copper electrolyte.

There is a loss of acid at the kettles while dissolving silver in sulphuric acid. This loss depends partly upon the rate at which the process is conducted and on the efficiency of the condensing apparatus, but in any case amounts to a considerable percentage of the amount used.

Another objection to the acid method is the non-elimination of tellurium from the silver during parting. This element materially affects the physical properties of the silver with which it is alloyed, small quantities causing it to crack at the edges while being rolled.

In minting operations the fine silver is first alloyed with 10 per cent copper. One part per thousand of tellurium in the silver will cause this alloy to crack when rolled out, so that five-tenths to six-tenths part per thousand (.05 to .06 per cent) is the probable safe maximum amount of tellurium that can be present in silver and meet the strictest requirements. To produce silver from material containing much tellurium, so that the refined product will not contain more than the above maximum allowable amount, requires, when parting with sulphuric acid, an exposure of the bullion in the cupel to air and nitre for a considerable length of time. This is shown by the following illustration.

The usual quantity of slimes were melted in a cupel furnace. After skimming the slags, two tuyere pipes 2 inches in diameter were placed so that air under 5-ounce pressure blew directly on the surface of the bullion. Nitre was used freely. Twelve and one-half hours were required to eliminate sufficient tellurium to secure a refined product of the requisite fineness. The analysis of the several samples are given in Table I., which shows also the early removal of the selenium. This bullion, represented by sample E, was parted with sulphuric acid, and the refined silver contained .6 part of tellurium. A portion of the same lot parted electrolytically contained not a trace.

TABLE I.—ELIMINATION OF TELLURIUM AND SELENIUM CHARGE NO. 59.

Time Taken		Parts Fineness			
		Te.	Se.	Ag	Au.
Sample A—April 8,	5.00 A.M. . .	34.30	13.45	903.72	15.08
" B "	9.00 A.M. . . .	23.40	.81	939.47	16.53
" C "	1.00 P.M. . .	5.30		968.61	17.10
" D "	3.00 P.M. .	2.41		973.35	17.25
" E "	5.30 P.M. .	1.31		973.46	17.04

With electrolytic parting we have a choice of two distinct systems of depositing silver on the cathode, one in a loose crystalline form at a relatively high-current density, as in the Balbach and Moebius methods, the other with the aid of gelatine in an adherent form at a lower current density.

The electrolyte used is a copper-silver nitrate solution, although recently Betts has proposed using a silver methyl-sulphate solution.

These methods all have in common the characteristic of parting and refining bullion free from gold and tellurium at one [illegible] the deposited silver being melted and poured into [illegible] without any further refining, as in the sulphuric acid process. Silver placed in the tanks as anodes is not handled until taken out as refined silver, whereas in the acid method the silver either in solution or as cement must be transferred several times with the aid of siphons, steam, etc., before it is in a condition to be melted. For these reasons it is possible to operate an electrolytic parting plant with a higher degree of neatness and cleanliness (such as the value of the material treated requires) than is possible with acid parting.

A parting plant using the Balbach method is simple in construction and operation. Fig. 2 shows the cross-section of a tank. The cathode is made of ½-inch Acheson graphite slabs fitted to the bottom. Two silver contact pieces rest respectively on the bullion to be parted and the graphite slabs. Bullion cast in thin, square slabs is contained in a cloth case, which is supported on a wooden frame suspended over the tank. The gold slimes accumulate on the under side of the bullion, between it and the cathode, increasing the resistance as the operation continues. Each tank has a cathode surface of 8 square feet and a current density of 20 to 25 amps. per square foot is used. The voltage averages 3.8 per tank, and an average ampere efficiency of 93 per cent was obtained on a continued run, while occasionally an efficiency of 98 per cent was secured. The power required is 31.5 watt-hours per ounce of fine silver produced.

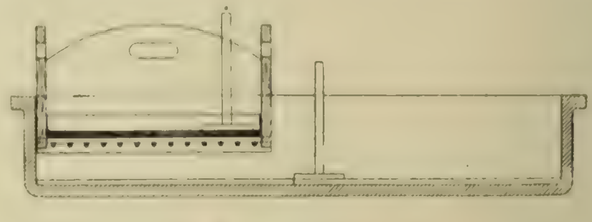

FIG. 2.—BALBACH TANK.

Most of the silver is deposited on the cathode surface directly under the anode, and the reduction of the distance between anode and cathode is limited by the space necessary to reach in and remove it, which has to be done frequently on account of the silver bridging across to the cathode. This serves also to agitate the electrolyte. There is gassing in this tank, and the consumption of nitric acid is much higher than in the Moebius method.

At 20 amps. per square foot about 32 per cent of the daily output of each tank is held permanently in stock in electrolyte and contacts, which is less than is retained in the Moebius method.

In Fig. 3 is shown the cross-section of a Moebius tank. They are arranged in units of six, placed end to end, each unit being provided with apparatus for raising the boxes containing the deposited silver together with the anodes and cathodes, and with arrangements for imparting a reciprocating motion to the wooden scrapers. There is no system of circulating the electrolyte, but the scrapers moving back and forth agitate it. The anodes are contained in a cloth frame which holds the gold slimes, and the silver is brushed off from the silver cathodes by the wooden scrapers, and drops into a box with hinged bottom. It is removed by raising the boxes above the top of the tanks and emptying it into a tray placed beneath. This operation requires one-half hour per day per unit. Each tank has a cathode surface of about 16.5 square feet, and a current density of 20 to 25 amps. per square foot is used. The voltage between electrodes is 1.4 to 1.5, and the power cost is 13.2 watt-hours per ounce of silver deposited. An average ampere efficiency of 94 per cent is obtained. At 20 amps. per square foot 41 per cent of the daily output of each unit is permanently held in stock in cathodes and electrolyte.

The necessity of cutting out of service the units of a plant using the Moebius method to remove the silver, and the frequent siphoning off and replacing of portions of the electrolyte in each tank in both the Balbach and Moebius methods, to maintain it of fixed compositions, are objections overcome by depositing the silver on the cathode in an adherent form.

This method permits of an arrangement of tanks and electrodes and a system of circulation of electrolyte similar to that used in the multiple system of copper refining.

The finely divided condition of the gold in the bullion, which in commercial work rarely contains more than forty parts per thousand, requires that the anodes be inclosed in a cloth frame to keep the deposited silver free from gold, as the light, fine particles do not fall to the bottom of the tank with sufficient rapidity. A current density of 10 amps. per square foot is used, and the power cost is nearly identical with the Moebius method; 28 to 32 per cent of the daily output is retained in cathodes and electrolyte.

Gold from either the sulphuric or electrolytic parting process can easily be produced of a fineness of .990 or above, free from silver. The government charge for melting and assaying such gold is so low, varying from .6 to .9 cents per ounce, depending on the weight of the bar, that there seems to be no immediate possibility of an electrolytic gold-refining process, such as the Wohlwill, becoming an attractive commercial proposition, except in those cases where platinum is present and the output large. Such a process would increase costs by delaying shipments, with loss of interest on gold in process, and that permanently held in stock in cathodes and electrolyte, and by increasing labor costs with no increase in the value received for the gold.

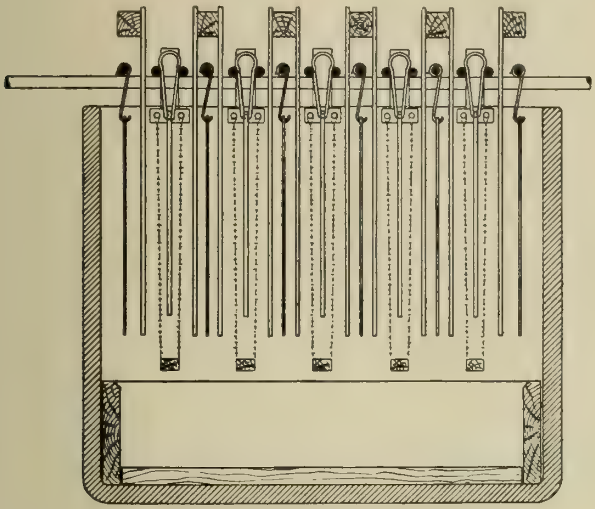

FIG. 3.—MOEBIUS TANK.

The paper was briefly discussed by Messrs. Betts, Sadtler and Bancroft.

Insulating Paints.

An interesting paper was then presented by Dr. Maximilian Toch, who pointed out that electric traction is responsible for a new chemical industry, that of insulating paints. Traction engineers as well as municipal authorities discovered that an old steel bridge, over which a new trolley line passes, soon shows the effects of the decomposition of the steel, and this deterioration is now called electrolysis. A road which is run by the third-rail system soon shows evidence of oxidation in spots hitherto unnoticed. Against these ravages insulating paints, compounds and varnishes are used, and while many of these materials are excellent when they are first applied, they become useless in proportion to the conditions to which thy are subjected.

Dr. Toch pointed out that this subject is really so vast that he could only touch on the topic in a general way. An important point is the recognition of the fact that materials which are designed for a tunnel are useless in the sunshine. The materials used in tunnel work must withstand moisture, and unless they do they will be disintegrated until eventually they form conductors. As a rule, the paints for tunnel or underground work are composed of bitumens, and are black. The bitumens as a class are of the polymethylene group, and their structure is one atom of carbon and two of hydrogen, combined many times. Moisture will neither add to nor subtract from this structure. On the other hand, there are the fatty acid oils which form the bases of many good insulating paints, and when these are used in damp places they are not split up but added to, and their value as an insulating paint is lessened.

While bitumens are good for tunnel work, a study of the behavior shows that sunlight destroys them, and the two atoms of hydrogen combine with oxygen to form water, while carbon is set free. Consequently, it is noticed that a bitumen insulating varnish in sunshine soon becomes smutty, due to the liberation of carbon, and this justifies the deduction that for exposed use bitumen is not to be relied on, excepting where it can be replaced every few months.

If we take sugar and add to it strong sulphuric acid, a reaction takes place which demonstrates that sugar is composed of carbon and water, and inasmuch as sulphuric acid has a great affinity for water, the reaction is complete. This experiment was shown by Dr. Toch, who pointed out that many of the bitumens and waxes are acted on by acid in this manner. Since we have 14 per cent hot acid solutions to deal with in the electrolytic deposition of copper, it is most interesting to observe the action of the acid on the various electrolytes, and those that appear best in the beginning are often the soonest to fail.

The time of drying is often a factor in the life of an insulating paint. The slow-drying paints should never be subjected to aqueous liquids until drying is complete.

For cross-arm painting a very high resistance is not so essential as one that will withstand the weather, and a liquid medium which soaks into the pores of the wood is better than one which dries by oxidation and forms a film which peels.

Concerning third rails, the author pointed out that they are subject to oxidation in so violent a manner that it is often found necessary to replace them sooner than the running rails. The sides or web of the rail can be protected with a paint which will resist corrosion and short circuiting. The top is sometimes covered with a wooden guard. This guard must be painted with a fire-resisting and insulating material.

Concerning return wires, Dr. Toch pointed out that it is remarkable how corrosion takes place along the same. As a matter of fact, the metal surrounding these places is usually coated with an oil paint containing lead and zinc, and it has taken practical demonstrations to convince some mechanics that such paints produce, or superinduce, electrolytic corrosion. Strict vigilance and copious applications of the correct material are necessary in such places.

In the conclusion of his paper, Dr. Toch recommended to professors of electrochemistry to take up this subject with their students and allow them to work out for their thesis subjects of this kind. There is no reason why a third-rail paint should not be made one that will stand both dampness and sunshine for a year. There is also no reason why insulating materials should not serve as protector to metals against oxidation as well as electric currents. Fortunately, we have records of the preservative action of many paints on wood and metal, and it only remains now to work out the proper combination, and to this end chemists who have a technical training ought to be employed by colleges to give students occasional lectures on raw materials, so that they obtain a fundamental training in the theory of coatings. A student who knows these things has a better future in store for him and an easier chance for employment.

Dr. Toch stated that very extended research work in his own laboratory has established the fact that from the way a paint is made up it is possible to predict what it will do.

A peculiar observation made on the New York subway as well as on the elevated road is that the steel connected to the negative side of the electric system (hence forming the cathode) shows greater corrosion than the positive side, although corrosion is observed on both sides.

This last point was the subject of some lively discussion by Messrs. Bancroft, Hering, Larchar, Richards and Sadtler, who suggested various explanations.

Dr. Toch stated that there are three different kinds of rust. Electrolytic rust is a mixture of Fe_2O_3, $2H_2O$ and Fe_3O_4 $1H_2O$.

Active Oxygen, Electrically and Chemically Produced.

An interesting paper by Dr. **Richard von Foregger**, of the Roessler & Hasslacher Co., was then presented. The author discussed the utilization of active oxygen for sterilizing, bleaching, metallurgical and other purposes, produced from the series of peroxides which are now being made by the Niagara Electrochemical Co. (as was already mentioned in a paper of Mr. Fitzgerald, page 255 of our July issue).

There are two kinds of autoxidizers which are direct sources of active oxygen; one is ozone, the other is represented by these peroxides, chiefly of the alkaline and earth alkaline metals. In their decomposition, *i. e.*, in their formation of active oxygen, the action is that ozone is split up into O_2 and active oxygen, while the dioxides are split up into lower oxides and active oxygen.

These two sources of active oxygen, compared from a practical side, show ozone as a gas and the dioxides (with the exception of hydrogen peroxide) as a powder. Ozone gas has to be produced at the place of consumption by electrical energy. It is an unstable gas which cannot, for practical purposes, be stored, and has to be utilized immediately after its formation. On the other hand, the peroxides are stable, transportable, storable and enter automatically into reaction whenever required with a fixed amount of energy.

Oxygen is a vital part of life, and organic life has been described as a slow process of oxidation. Besides this, oxygen has in nature three main features, which in proper reproduction will become indispensable factors in industrial work. Oxygen disinfects, it purifies and it bleaches.

Dr. von Foregger discussed at some length the sterilization of water, and gave some data on the use of ozone as produced by silent electric discharge. He then compared this with the use of active oxygen obtained from peroxides.

"Assuming that we have a 65 per cent calcium peroxide, *i. e.*, a preparation containing 65 per cent dioxide (CaO_2)—the rest being monoxide or oxyhydrate—corresponding to 15 per cent of active oxygen, and assuming that the cost of 1 kg calcium peroxide is $2, we find that 1 kg contains 150 grams available oxygen, and, therefore, the cost of 1 kg active oxygen is $12.50.

"Taking the figures of Siemens & Halske of 0.67 grams active O per m^3 water as a basis, we find that the cost per m^3 water sterilized by means of calcium peroxide is 0.84 cents. Allowing the high addition of 20 per cent for operating expenses and amortization of the little machinery that is required, we obtain a total cost of 1 cent per m^3 sterilized water."

"To be just, it must be said that this total cost occurring with calcium peroxide will not be reduced further than to a certain limit above 0.84 cents, even if the operation takes place on the largest scale, as in this event the calcium peroxide counts almost as exclusive expense. On the other hand, for the same reason, the total cost can hardly surpass 1 cent per m^3, even if the operation is done on a very small scale."

Compared with ozone processes, the cost of using calcium peroxide is slightly higher for large plants. But it is for smaller plants that the use of calcium peroxide offers advantages in cost. It also has the great advantage of simplicity, there being practically no machinery required. A tank fitted with a stirring apparatus and eventually, but not absolutely necessary, a filter and one man to handle it, is all that is required.

The calcium peroxide is applied in powdered form as it comes from the manufacturer, and an equivalent of sulphuric acid, or an organic acid, forming an insoluble calcium salt, as, for instance, citric acid, is added to hasten the liberation of active oxygen and to precipitate the calcium. The bactericidal action is complete in the time from 5 to 30 minutes, and the resultant water is just as pure and good as could be.

With respect to the time required for the peroxide process this is much less than by the use of ozonized air. In the latter case there is only a limited amount of ozone available per unit or time and unit of space.

The main value of the peroxide method consists, however, in its use outside the sterilization plant and without the aid of any apparatus. In form of calcium peroxide, or magnesium peroxide, we are able to "carry available oxygen around in our pockets ready for sterilization of our drinking water at any place." Some details are given on very satisfactory results of bacteriological tests of the effect of this treatment of water.

Dr. von Foregger briefly referred to the sterilization and preservation of food by means of ozone or peroxides, and then passed over to the subject of bleaching. The success with which sodium peroxide has been introduced in this country as well as in Europe for bleaching textile and fiber and straw, is believed to vouch for its commercial adaptability. The author stated that perborate (one other material of the series) is becoming extensively used in Germany for bleaching olive, cotton and linseed oils, and thinks the cost is 1.06 cent per kilogram of oil. In a series of experiments which he made with cotton-seed oil, as a substitute for olive oil, he used calcium peroxide, and found that ⅛ per cent of this material furnished satisfactory results as to shade of bleach, and that in taste and flavor the oil was rather improved and by no means hurt. He figures that the cost of this process will be about ¼ cent per kilogram of oil.

Dr. von Foregger then turned to a discussion of the use of active oxygen in gold metallurgy.

He referred to the controversy between McArthur, the inventor of the cyanide process for extracting gold, and Elsner. "McArthur bases the reaction occurring in the cyanide process upon the following principle:

$$2Au + 4KCN + 2H_2O = 2KAu(CN)_2 + 2KOH + H_2$$

He always pretended, and still pretends, that the presence of oxygen during the process is of no value whatever. Elsner, McLaurin and Gogder, on the other hand, pretend that the presence of oxygen is essential, and describe the reaction as follows:

$$4Au + 8KCN + 2H_2O + O_2 = 4KAu(CN)_2 + 4KOH$$

"Since publication of the results of investigations made by Bodländer (*Zeitschr. f. Angew. Chem.*, 1896, p. 583), it can be said that this question is cleared. He proved that in the reaction the amount of oxygen which is put into activity is nearly the same as the amount of oxygen which combines (about 72 per cent). With the theory of dissociation on hand he then came to the following reaction:

$$2Au + 4\overset{+}{K} + 4\overset{-}{CN} + 2\overset{-}{OH} + 2\overset{+}{H} + O_2 =$$

$$= 2\overset{-}{Au(CN)_2} + 4\overset{+}{K} + 2\overset{-}{OH} + H_2O_2$$

"Gold, during its solution in potassium cyanide, at the presence of air or oxygen is then a pseudoautoxidator, *i. e.*, it produces first the secondary autoxidator K or H, and K or H produce the autoxidation forming potassium peroxide hydrate or hydrogen peroxide. The secondary process then of the solution of gold may be expressed as follows (see Engler and Weissberg: Autoxidation):

$$2Au + 4KCN + H_2O_2 = 2KOH + 2KAu(CN)_2$$

"The more peroxide formed—dependent on the presence of oxygen—the quicker the solution of gold. Gold-mining experts may have come to this result by practical experience, despite the contradictory assertions of McArthur. It is probably due to this fact that considerable quantities of sodium peroxide are being used by gold-extracting concerns. Of late, sodium perborate has attracted the attention of that part of mining experts, as borax, which besides peroxide of hydrogen is formed in the solution of perborate, seems to be a suitable medium.

"A practical confirmation of the theory of the value of oxygen in the cyanide process is found in the so-called double treatment, consisting in the use of two vats, the contents of one vat after having undergone a treatment being discharged into the second vat for renewed treatment, during which proceeding the material is exposed to the air, drawing its advantage in oxygen supply.

"The importance of oxygen is fully recognized in the refinement of gold bullion and cyanide precipitates. Kirke Rose (*Inst. of Mining and Metallurgy*, April 13, 1905) concludes after a series of elaborate experiments that "base metals can be removed from gold and silver bullion by means of a current of oxygen passed through them." Rose in his experiments used air as well as oxygen gas, and found that air can do what oxygen does, only more slowly and without such intense action, and that it is obviously a cheaper oxidizing agent than oxygen in any other form."

Dr. von Foregger then referred to some tests made at the laboratory of Borchers, on Aachen, on the use of oxygen in the copper converter process. These experiments proved that in using blast air containing 5 per cent to 9 per cent above the normal contents of oxygen, first, the temperature was remarkably raised, and, second, the time of blasting was considerably shortened, viz.: to two-thirds and one-half of the time needed with ordinary blast air.

The author finally discussed and demonstrated "oxone" which is fused peroxide of sodium, prepared to generate oxygen gas instantaneously, upon being immersed in water. "While sodium peroxide in its powder form dissolves in water, forming peroxide hydrate or atomic oxygen, not entering the gaseous state, the fused peroxide by heat of dissociation in the instant of solution drives out all its available oxygen. That this oxygen, which is in the molecular form, does contain a certain percentage of ozone, it is fair to assume."

It is found that 1 pound of oxone furnishes on an average 2.15 cubic feet oxygen gas. As oxone has a small volume, the gas production amounts to 312 times its own volume, or 130 times the volume of water.

Oxone is adaptable for shipment to any distance and offers no danger of explosion or combustion, and no inconvenience whatever in the use except the strong causticity, which has to be taken into consideration when handling the same.

The paper elicited considerable discussion. Dr. Toch thought that the new peroxides seemed a good thing, and might fill a want for bleaching linseed oil, making the same siccative, etc. Dr. Doremus referred to the possibility of using calcium permanganate or potassium permanganate. Dr. Richards spoke of the use of sodium perborate in connection with refining gold and silver. Mr. Fahrig thought that for the extraction of gold and silver from ores, condensed active oxygen is not so much required as a large surface of oxidizing air. Dr. Doremus compared the cost of using peroxides for making oxygen in large quantities with that of using compressed oxygen gas. From the discussion it appeared that the production from peroxides is even somewhat cheaper, but Dr. von Foregger pointed out that these peroxides are really expected to prove useful rather for generating oxygen in situ, so as to produce certain sterilizing, bleaching, etc., reactions, than for the manufacture of large quantities of oxygen gas. He also referred to the following valuable property of oxone: When dissolved in water, oxygen is evolved, while simultaneously the sodium hydrate, formed on the decomposition of the sodium peroxide, absorbs carbon dioxide. This property of simultaneously producing oxygen and absorbing carbon dioxide is of importance for use in submarine boats, in mines, etc.

Arsenic in Pickling Solutions.

A paper on this subject was presented by Prof. Charles F. Burgess. It has been known for some time that the sulphuric acid pickle, which exerts a vigorous dissolving action upon an iron or steel surface, may be rendered almost completely inert by the addition of a small amount of arsenic. In the tests made by the author a five-normal solution of sulphuric acid was used, and to a portion of this solution was added a small quantity (a small part of 1 per cent) of arsenic in the form of As_2O_3. To determine quantitatively the influence of the pure and the arsenic-containing solutions, strips of spring steel were subjected to their action, and the dissolving and the weakening of the steel was measured.

It was found that in 1 hour's time the amount of iron dissolved by the arsenic-free solution was about thirty-four times that dissolved by the impure solution; also that during the first 20 minutes the ratio between the action of the two solutions was 9.1 to 1, and during the last 40 minutes the ratio increased to 94 to 1, showing that the protective action of the arsenic increases with the time. In the pure solution the rate of corrosion increases, as shown by the fact that over four times as much iron was dissolved during the last 40 minutes as during the first 20; while with the arsenic-containing solution the rate of solution materially decreased. It was also noted that when first immersing the steel in the impure solution hydrogen was given off, but this evolution quickly diminished, and at the end of 1 hour the evolution of hydrogen had ceased, while in the pure acid the evolution of this gas was vigorous.

While arsenic is harmful as an addition to an acid pickle, the same phenomenon may, of course, also be applied to retard the action of acid on steel when no such action is wanted. On account of this strong action of acid solutions of steel, certain classes of steel articles, such as steel piano wire, clock springs, measuring tapes and the like, cannot be pickled, and it is claimed by some that they cannot be nickel-plated without damage. In this connection the effect of arsenic may be put to good advantage. By means of a special machine, which enabled him to test the brittleness of a strip of steel, Prof. Burgess showed that arsenic exerts a marked influence in diminishing the rotting effect of the sulphuric acid solution. A solution of hydrochloric acid having a corresponding strength was also tested, and it was found that while the presence of arsenic in it exerts an influence similar to that exerted in sulphuric acid, its influence is not as pronounced.

It is generally recognized that the nickel-plating process, as usually operated, has a decided weakening effect on thin steel springs and the like, and the supposition is justified that it is the pickling part of the process that is largely responsible for the trouble. If a steel spring is pickled in an arsenic-containing pickle, and subsequently nickel-plated, it is found to be stronger than when pickled in pure acid before plating.

Prof. Burgess urges to carry out further experimental researches on the quantitative effect of arsenic and to determine the underlying cause of this effect.

The paper was briefly discussed by various speakers, who offered one or the other possible explanation of the influence of arsenic.

Electrochemical Card Index.

Mr. C. F. Carrier, Jr., gave a brief and interesting account of the thorough work being done by Mr. A. Voege and the Concilium Bibliographicum, in Zurich, in connection with the preparation of a complete card index relating to articles, patents, etc., on electrochemical subjects. Practically all periodicals of the world are to be covered. Prof. Bancroft made some complimentary remarks on this work, and Dr. Waldo emphasized that it is absolutely necessary to use exactly the "standard" size of card.

For details concerning the undertaking the reader may be referred to two papers by Mr. Voege, published on page 313 of our Vol. II., and page 107 of our Vol. III.

Electric Zinc Smelting.

A paper by Prof. **Oliver W. Brown** and **W. F. Oesterle** was then presented, in the absence of the authors, by Mr. S. S. Sadtler. In the introduction the authors referred to the present zinc distillation process as the most wasteful metallurgical operation used industrially at the present time. The fuel consumption, the cost of repairs and the loss of zinc are very large. The average life of the retorts used in many works is only 40 to 50 days; while the loss of zinc is seldom less than 10 per cent, and sometimes more than 25 per cent of the zinc in the ore.

Another great drawback to the process is that the retorts will hold only about 65 pounds of ore. The zinc retorts most commonly used are fire-clay tubes about 4 feet 2 inches long, 8 inches in diameter and 1¼ inches thick. They are made closed at one end, and into the mouth of each retort is fitted a fire-clay condenser. They are supported at each end by projections from the walls of the furnace; and are placed in horizontal rows in the furnace in such a manner that the flames entirely surround them while they are being heated.

The authors then briefly described the operation of the present zinc distillation process, and pointed out its inherent drawbacks. Electrochemists have attacked the zinc problem along three different lines. The first is the conversion of the zinc in the ore into a salt soluble in water, and the subsequent electrolytic precipitation of the metal from an aqueous solution Of such processes that of Hoepfner is the only one which appears to have approached a commercial success. Brunner & Mond have during the last few years turned out a very pure electrolytic zinc from their works at Wilmington, near Chester, England. (See Meyer, our Vol. III., page 9.)

A second line of research relates to electrolysis of a fused salt of zinc. In this field the Swinburne-Ashcroft process "seems to have a very bright future." (See our Vol. I., page 412, also Vol. II., page 404.)

The third line of endeavor has been the smelting of zinc ores in the electric furnace. The Cowles Brothers, C. Casorette and F. Berboni (Brit. patent 472, May 19, 1900), De Laval (see our Vol. I., page 425; Vol. II., page 423; Vol. III., pages 54 and 209) have designed electric furnaces in which roasted ores are smelted in the ordinary manner except that the heat is generated by electrical means.

A Dorsemagen has tried to produce simultaneously zinc and carborundum from zinc silicate ores according to the reaction $Zn_2SiO_4 + 5C = 2Zn + SiC + 4CO$.

Dankwart (U. S. patent 746,798, Dec. 15, 1903) has devised a process in which he smelts mixed sulphide ores containing zinc, with lime, coke and sodium sulphate. He mixes the different materials in such proportions that the sulphur of the ore is converted into alkaline and alkaline earth sulphides, while the zinc is liberated as vapor and is condensed in the usual manner.

This is similar to a process of Kirkpatrick-Picard (U. S. patent 692,148, Jan. 28, 1902), except that the latter does not use sodium sulphate. Kirkpatrick-Picard's process is represented by the following reactions:

$$ZnS + CaO + C = Zn + CaS + CO$$
$$PbS + CaO + C = Pb + CaS + CO$$

$$4ZnS + 2Fe_2O_3 + 6C = 4Zn + 4FeS + 6CO$$
$$4PbS + 2Fe_2O_3 + 6C = 4Pb + 4FeS + 6CO$$

He forms the mixture into briquets before the distillation.

The authors of the present papers, Brown and Oesterle (our Vol. II., page 32) have worked in the laboratory of the University of Indiana and have devised a process based on the electrical smelting of changes of unroasted zinc blende, lime and carbon, mixed in proportions corresponding to the following equations:

$$(a) \quad 2ZnS + 2CaO + 7C = 2Zn + 2CaC_2 + CS_2 + 2CO$$
$$(b) \quad 2ZnS + CaO + 4C = 2Zn + CaC_2 + CS_2 + CO$$

The authors first tried the reduction of a charge containing equal molecular weights of zinc blende (59.6 per cent zinc in ore) lime and carbon (coke) in an enclosed electric resistance furnace lined with firebrick. This the same charge as that used by Kirkpatrick-Picard. When this charge was heated in the electric furnace the ore was readily reduced, the metallic zinc distilled out and a portion of it condensed in an iron tube which served as condenser. The material remaining in the furnace after the distillation, consisting of impure fused calcium sulphide, contained only 0.13 per cent of metallic zinc.

They next made a study of the electric reduction of charges of zinc blende, lime and carbon (coal or coke), mixed in proportions corresponding to the equations (*a*) and (*b*) given above.

When a charge containing 194 grams of zinc blende (59.6 per cent zinc), 112 grams of lime and 84 grams of coke was heated in an enclosed firebrick-lined electric resistance furnace, with a current of 50 amps. at 30 volts for 2 hours, zinc distilled and condensed, and the impure carbide remaining in the furnace contained only 0.036 per cent of metallic zinc and 2.89 per cent of sulphur. The other impurities in the carbide naturally depend on the purity of the ore treated.

When working with the very small furnace required to smelt charges containing only 194 grams of ore, it was not deemed desirable to endeavor to condense the largest possible proportion of the distilled zinc in a solid metallic form. As experience has shown, the difficulties in condensing the zinc as liquid metal, instead of zinc dust, disappear as the size of the furnace is increased.

The analyses of gases coming from the furnace during the operation, of the materials in the furnace after smelting, etc., show that the changes which take place in the smelting are probably best represented by equations (*a*) and (*b*).

Some additional experiments which were carried out during 1903 and 1904 by Prof. Brown in Prof. Burgess' laboratory at the University of Wisconsin, on the reduction of zinc blende, furnished the following results:

When the following charge: 3.98 kg zinc blende (containing 58.0 per cent of zinc), 2.24 kg lime (containing 38.2 per cent of magnesia), 0.84 kg of carbon (equal weights of coal and coke), was smelted in an enclosed electric resistance furnace, the material remaining in the furnace after the distillation contained only 0.10 per cent of metallic zinc. This experiment shows that even when a lime very high in magnesia is used, practically all of the zinc is reduced and distills out of the furnace. However, it is hardly necessary to remark that good calcium carbide cannot be made when a lime containing 38 per cent of magnesia is used.

The internal dimensions of the furnace in this experiment were: Length, 12.5 inches; depth, 8.5 inches, and width, 4.5 inches. The inner walls and bottom of the furnace were of magnesia brick. A layer of dry lime was placed on the outside of the magnesia brick, then a layer of firebrick, followed by another layer of lime, and finally the whole was encased in a sheet-iron jacket. A round Acheson graphite electrode, 2 inches in diameter, entered the furnace at each end. These electrodes were fastened firmly into plates of Acheson graphite 0.5 inch thick, which were placed vertically at each end of the furnace, and reached from the bottom to within 1.5 inches of the top. In order that the furnace should be gas tight, the portions of the graphite electrodes which passed through the walls of the furnace were packed in powdered magnesia brick.

The materials comprising the charge were passed through a twenty-mesh sieve and thoroughly mixed, before placing in the furnace.

Connection between the two end electrodes was made by two cores of broken carbons(broken to pieces about 0.5 to 1 inch), which were imbedded in the charge. The lower core of granular carbon was placed 2 inches from the bottom of the furnace, while the second core was about 4 inches above the lower core.

The charge was filled into the furnace to within 1.5 inches of the roof, and was covered with a layer of broken pieces of coke. A carbon tube, 0.5 inches internal diameter and 12 inches long, penetrated the side wall of the furnace, about 2 inches from one end and 2 inches from the top. The tube passed nearly horizontally through the walls of the furnace, the outer end being about 0.5 inch lower than the inner. On the outer end of this carbon tube was fastened a short piece of 1.5-inch iron pipe, which was surrounded with asbestos, and served as condenser for the zinc. The outer end of the pipe was nearly closed with fire-clay. At intervals during the distillation of the zinc the fire-clay plug was removed, and the melted zinc, which had condensed, was allowed to flow out.

The furnace was closed at the top by an Acheson graphite plate, 0.5 inch thick. A layer of lime was placed over the graphite plate, and on this were placed three layers of fire-brick. When the furnace was sealed with dry powdered lime in this manner, it was quite tight, and very little gas escaped except through the condenser.

A current of about 172 amps. and 68 volts was passed through the furnace for 6 hours and 40 minutes. Zinc vapors had stopped coming from the furnace when the current was broken.

After the furnace was allowed to cool and the cover removed, it was found that the charge was a loose porous mass, which could nearly all be removed with the hands. The temperature had not been sufficient to fuse the charge, although all but 0.1 per cent of the zinc was expelled from the furnace.

In order to gain some idea of the temperature required to reduce and distill the zinc from various charges, and also the relative value of various charges, four different mixtures were made up and heated in a hot coke fire in a wind furnace.

The materials in each charge were pulverized to pass a twenty-mesh sieve and well mixed. The blende used contained 58.6 per cent of metallic zinc. The charges were as follows:

No. 1 contained zinc blende, 97; lime, 28, and coke, 24 grams, and corresponds to the equation,

$$2ZnS + CaO + 4C = 2Zn + CaC_2 + CS_2 + CO$$

No. 2 contained zinc blende, 97; lime, 56, and coke, 42 grams, and corresponds to the equation,

$$2ZnS + 2CaO + 7C = 2Zn + 2CaC_2 + CS_2 + 2CO$$

No. 3 contained zinc blende, 97; lime, 56, and coke, 12 grams, and corresponds to the equation,

$$ZnS + CaO + C = Zn + CaS + CO$$

No. 4 contained zinc blende, 97; silica, 120, and coke, 24 grams, and corresponds to the equation,

$$2ZnS + SiO_2 + 4C = 2Zn + SiC + 2CO + CS_2$$

The four charges were placed in separate assay crucibles, loosely covered, and placed in a hot coke fire. Charges 1, 2 and 3 were heated two and three-fourth hours, while charge No. 4 was heated two and one-sixteenth hours. A zinc flame burned from the mouths of crucibles 1, 2 and 3 while they were being heated, but no zinc flame was given off from crucible containing charge No. 4, indicating that no reduction took place in No. 4.

An analysis of charges 1, 2 and 3 after heating showed that about 55 per cent of the zinc had been reduced and volatilized in each case. A practically complete reduction and volatilization of the zinc takes place when either charge 1, 2 or 3 is heated in an enclosed electric furnace, with a current of 50 amps. and 30 volts for 2 hours or less.

Only 5 per cent of the zinc in charge No. 4 was lost during the heating. This loss was probably due to mechanical loss in removing the ignited charge from the crucible after firing.

A charge made up in the same proportions as in No. 4, but containing zinc blende, 194; silica, 260, and coke, 48 grams, was heated in an enclosed electric resistance furnace for 1 hour, with a current of 120 amps. and 50 volts. Only a very small amount of metallic zinc (about 1 gram) was found in the condenser, and an analysis of the powder which collected in the condenser showed that it contained only about 4 per cent more metallic zinc than the charge which was placed in the furnace. The material remaining in the furnace after heating contained 3.98 per cent of metallic zinc.

These data show that zinc sulphide is not reduced to metal when mixed with silica and carbon and heated to a very high temperature. The temperature attained in this experiment was high enough to volatilize most of the charge out of the furnace, yet the zinc ore was not reduced to metal.

The authors conclude that it is a question of only a few years that the electric furnace will be a commercial accomplished fact in the zinc industry. Two lines are promising. First, roasted zinc ores may be smelted in the electric furnace with coke or coal and just enough other material to form with the impurities an easily fusible slag, which may be easily tapped from the furnace. The advantages of the electric process are that it can be made entirely continuous, that the walls of the furnace can be constructed of materials impervious to zinc vapors, and not corrugated by the slag so that the temperature may be raised to a point at which all of the zinc will be expelled from the ore, and the saving of energy due to the application of the heat internally. Ores high in iron and other impurities could be easily smelted, since the walls of an electric furnace can be made of materials which will withstand the corrosive action of the slags.

A second line of possible commercial developments would be in the smelting of zinc sulphide in the electric furnace without the preliminary roasting, by mixing ore, lime and carbon in proportions required for producing metallic zinc, calcium sulphide and carbon monoxide. The fused calcium sulphide could then be tapped from the furnace and the sulphur subsequently recovered. All but traces of zinc can be distilled from such a charge.

The paper was briefly discussed by Prof. C. F. Burgess, who said that the endeavor to make calcium carbide and zinc simultaneously in the electric furnace contradicts the fundamental metallurgical principle that it is impossible to carry out in a single apparatus simultaneously two reactions which require very different conditions for their most efficient operation. Dr. J. W. Richards thought that pyrometric measurements would have given very interesting information, and pointed out that the reactions (*a*) and (*b*) are strongly endothermic reactions. He also presented some notes communicated by Mr. Woolsey McA. Johnson.

Mr. W. McA. Johnson's remarks deal with general considerations of the zinc market and look into possibilities of future improvements in this line. He points out that in the past ten years all metals have shown a wonderful increase in production, and, of course, in consumption in the United States. The figures average a yearly increase of 7 to 10 per cent per year. Of course, in ten years 7 per cent increase compounded will mean a doubled production. This fact is bound up with our great industrial expansion since 1896, and perhaps the figures 7 per cent less certain correction charges for insurance was the normal rate of interest for the capital invested in metallurgical industry. Mr. Johnson points out that since zinc is made in small retorts of a capacity of 50 pounds of ore and a daily production of some 20 pounds of metal, for these reasons zinc was restricted in its normal growth. Whereas the copper, iron or lead furnace could put through an enormous tonnage of ore with no increased labor charge per unit as size increased, the zinc business was held rigid and fast by the smallness of its unit. Mr. Johnson also makes some suggestions as to the limited market for spelter. Spelter has only one indispensable use, namely, brass making, and its great use for galvanizing is restricted by the fact that any increase in the price of spelter increases the price of galvanized sheets to such an extent that construction work will be preferably done with ordinary black sheets.

He also points out that fire-clay, labor and coal have advanced in price immensely the past ten years, and that this, with the lower grade of zinc ores used, increases the cost of producing spelter to an extent not marked in the other metals as lead, iron and copper. Now, there are several possible means for improvement in this line. First, an improvement in the old retort which Mr. Johnson thinks has possibilities not known to those who had not practical experience in the zinc business. He also points out that great progress is being made in the mechanical concentration of low-grade zinc ores, and makes reference to the electromagnetic and electrostatic separators. Attention is also called to possibilities of the pressure blast furnace and also to the old blast furnace used for the production of a high-grade oxide, to be retreated in the old retort. Mr. Johnson dismisses as impractical the leaching with subsequent electrolysis and also the Swinburne-Ashcroft process; in view of the fact that so much has been written and so much claimed about these processes, he thinks that no material evidence of their success is a proof that they are impractical. Reference is also made by Mr. Johnson to the electric furnace with which he, as is quite well known in the business, has made extensive experiments.

The electric furnace, it is pointed out, is especially adapted, as all electrical and metallurgical processes are, for the refining of a crude material previously made. This has been the experience of the electric steel furnace, the electrolytic lead process, the electrolytic nickel process, and the electrolytic copper process. In general, it may be said that electricity is an expensive agent, and should only be used for putting on the finishing touches. Mr. Johnson says in conclusion that while he does not think the treatment of ores in the electric furnace to be impossible, yet he now thinks it is fraught with great metallurgical, practical and commercial difficulties.

Electrometallurgy of Antimony.

A paper on this subject was presented in abstract by Mr. **Anson G. Betts**. Since antimony ore quite often contains gold, the question of the purification of crude antimony is quite interesting.

Acid ferric chloride solution is said to attack stibnite with ease, producing a solution of ferrous and antimonious chlorides and a residue of sulphur, and from this solution metallic antimony and ferrous chloride should be recovered by electrolysis. Mr. Betts has applied the same idea to the treatment of crude antimony and has found that the electrolytic operation can be quite easily carried out. Carbon anodes and copper or lead cathodes may be used. To avoid the necessity of using a diaphragm for the separation of the anode solution of ferric chloride and the cathode deposit of antimony, the following precautions are taken. The ferric chloride generated at the anode is heavier than the main solution, and continually runs down the anode surface and collects at the bottom of the cell, whence it may be drawn off, while fresh ferrous chloride-antimonious chloride is added at the top.

Mr. Betts operated with a current efficiency of 90 per cent. The electromotive force of the element $Sb\text{-}SbCl_3.FeCl_2\text{-}SbCl_3.FeCl_3\text{-}C$ is about 0.75 volt. The solution should be quite strongly acid to prevent the deposition of antimony oxychloride from the solution as a white precipitate.

Mr. Betts then passed to the discussion of antimony trifluoride which is much superior to antimony trichloride for electrochemical purposes. When a solution of it is electrolyzed with insoluble lead anodes, the main action at the anode is the liberation of oxygen as gas, and the formation of hydrofluoric acid in the solution. When iron is present a good deal of ferric salt is formed, particularly with low anode current densities.

Antimony oxide is obtained in some metallurgical operations, such as the roasting of antimony sulphide, and this may be dissolved in hydrofluoric acid and electrodeposited from the solution with lead cathodes and lead anodes as a beautiful solid deposit, and, if the solution is free from copper and antimony, of high purity.

Copper may be said to be the most troublesome metal in the electrometallurgy of antimony. In the chloride solution it stands very close to antimony in the electromotive force series, but in the fluoride solution it stands below antimony, and may be precipitated from solution by antimony. The explanation of this appears to be found in the fact that copper will exist in the fluoride solution only as a cupric ion, which has a different electromotive force of solution from the cuprous ion of the chloride solution.

Mr. Betts finally made some remarks on the electrolytic refining of antimony. The chloride solution is not supposed to be suitable. When the fluoride solution was first tried, alkaline sulphates and sulphuric acids were added to improve the conductivity. There were, however, troubles, and these Mr. Betts has traced to the presence of the sulphion.

With the omission of sulphion there is no difficulty at all. The anodes as readily dissolve as copper or lead anodes and the cathode deposit of antimony leaves nothing to be desired. No analyses have as yet been made of the antimony, but there is no reason to think that it is not very pure. These deposits do not tarnish at all, even in a laboratory full of fumes, when other metals tarnish badly, so that an electrolytic coating of antimony should be very useful as a protection against corrosion, especially on objects which are not bent, as the antimony coating is very brittle.

Pyrometer.

Mr. **F. F. Schuetz** then briefly described and demonstrated a new Bristol thermoelectric pyrometer which has certain distinct practical advantages. The thermoelectric couple is made separately from the leads to the instrument (which is a Weston instrument), and is connected with the leads by means of a flexible joint. If the couple should be hurt, it is easy to replace it, without replacing the leads themselves. It is possible to place quite a number of thermo-couples at different points of the furnace, and measure the temperature at these different points with one single instrument. We hope to describe the instrument at greater detail in a future issue.

Excursions to Plants of New Jersey Zinc Co. and Bethlehem Steel Co.

At 12.30 the visiting members and friends were entertained at luncheon by the Trustees of Lehigh University in the Gymnasium. Afterwards a group photograph was taken in front of the university chapel.

At 2 P. M. a start was made for the zinc works of the New Jersey Zinc Co. of Pennsylvania. Here the party were met by the genial manager of the works, Major de Saulles, and inspected the plant. The zinc ore, obtained mostly at Franklin, N. J., is there magnetically concentrated, and comes to the works as magnetic heads, rich in iron and manganese, and the willemite tailings, almost free from impurities. The former are treated in long runs of Wetherill "blowing hearths," in which, on a bed of anthracite, the zinc is volatilized, and rising from the hearth is burned by air to zinc oxide. The latter is drawn off by powerful fans and forced into the long bag filters, which are 30 inches in diameter by 40 feet long, and of which several hundred are used. The product is white zinc oxide paint material. The residue from these hearths is reduced in a small blast furnace to spiegeleisen, carrying 15 to 20 per cent of manganese. The visitors were fortunate enough to see the operations of tapping both slag and metal.

The other part of the plant was then visited, where willemite ore is reduced in retorts to metallic zinc. The operation of making a retort in a hydraulic press was watched with interest, and the large gas-fired Belgian retort furnaces with long runs of retorts were inspected with great attention, the small scale of the units used and the large amount of labor required being objects of wonderment to those seeing them for the first time.

It really does seem as if modern large-scale methods are constitutionally inapplicable to the present metallurgy of zinc.

The party then passed into the adjoining plant of the Bethlehem Steel Works, where between 7000 and 8000 men toil night and day, and iron and steel are transformed from dull red ore to brightly polished shafts and guns. The eighteen large open-hearth furnaces, capable of producing 1000 tons of steel a day, the powerful steam hammers forging massive bars, the still more powerful hydraulic presses squeezing 2 feet of red-hot iron like cheese, the 8000-ton hydraulic press, under which the fluid steel was placed under compression to set solid, the immense steel foundry, where patterns as large as the side of a house were being moulded into refractory sand to make 50 and even 80-ton castings, all impressed and almost overpowered the visitors by their titantic strength and massive proportions.

On the next building were seen an overwhelming display of guns in all stages of manufacture, shafts, cranks, machinery castings, 1800 x 400 feet of space, filled as closely as possible with the finest of modern machinery turning out the finest of modern steel work. A glance at the iron-casting shop and a visit to the mechanical and chemical laboratories found the available time exhausted, and most of the party also ready to admit that the excessive heat and consequent bodily fatigue rendered them incapable of further sightseeing.

Assistant Manager Acker, Superintendents Rawle and Wright and Messrs. Heilig, A. D. Meixell, L. N. Meixell, Brush, Wright, Hartz and Kautz were the all-efficient guides on this most enjoyed expedition.

Banquet.

On the evening of Tuesday a subscription banquet was held in the Eagle Hotel. It was very greatly enjoyed on account of its familiar and informal nature. Dr. J. W. Richards proved to be an excellent toastmaster. In his opening remarks he said that, as its founders predicted, the American Electrochemical Society had not weakened any single one of the older electrical and chemical societies, but had on the contrary done excellent work by bringing them together; the society has been successful in getting as members advanced men from all the branches which it represents.

Dr. Wilder D. Bancroft made a very happy speech, full of humorous and pertinent epigrams. He mentioned the more important resolutions which had been adopted the day before at the directors' meeting, and which are mentioned above in the first part of this report. He extended a very hearty invitation to the society to hold its next meeting at Ithaca, and promised that although they could not show the members "the inside of a big zinc plant or steel plant nor the outside of an electrochemical plant," there were some interesting industrial developments in Ithaca, and then there is, of course, Cornell University. Prof. W. S. Franklin told some good jokes about the last meeting in Boston and about bygone days he spent in Berlin.

Prof. A. G. Rau blew the horn (or rather the trombone) of Bethlehem, and spoke in a most interesting way of its history, of the culture of music and of the general uplifting of the community by the Moravians. Mr. Acker, of the Bethlehem Steel Co., gave a sketch of the wonderful development of these world-renowned works, which now employ 7000 to 8000 men, and expect to employ in few years 10,000 to 15,000. Mr. H. B. Coho, in behalf of the strong New York delegation, made a felicitous, humorous speech, and celebrated Bethlehem as an economical city, referring to the 1-cent toll on the bridge to the university in South Bethlehem. Mr. C. F. Carrier, Jr., spoke of his years of study in Germany, and discussed the difference in the way Americans and foreigners look upon scientific subjects. Mr. A. von Isakovics, as secretary of the New York section, was pleased to see New York so well represented.

After the banquet there were songs in the parlor of the Eagle Hotel, with Dr. G. Drobegg at the piano. This was followed by a meeting of the "Free Ions" at the Brighton.

WEDNESDAY MEETING.

Specific Resistance of Electrolytes.

The first paper presented related to a standard method of determining the specific resistance of electrolytes, the author being R. Threlfall, F. R. S. He uses essentially the potentiometer method and connects the potentiometer with the electrolytic resistance to be determined by means of single reversible electrodes, such as that of Ostwald. The author claims for his method a high degree of accuracy.

The essential part of the apparatus consists of a uniform glass tube, which contains the electrolyte under test, into which are fitted an each end two glass T-pieces. The vertical branches of the Ts give access to platinized electrodes, which are connected to a storage battery in series with a high resistance. The horizontal branches of the Ts are used to fix the position of the testing electrodes of glass tube drawn down to long, fine points. These testing electrodes are connected in the usual way through two beakers with the Ostwald standard electrodes. The tube containing the electrolyte is immersed in a large bath containing heavy mineral oil and provided with a heating coil running the whole length, and with a shaft with screw paddles also running the whole length of the trough.

It is stated to be an essential part of the method, both that the main current is reversed after a group of observations and that several sets of comparisons are made alternately with the current, first in one direction and then in the other. The effect of taking an arithmetical mean of the results is to eliminate differences of potential difference in the single electrodes and intermediate contacts.

The paper was discussed by Prof. Burgess, who described a somewhat similar method used in his laboratory at the University of Wisconsin, also using Ostwald normal electrodes. Mr. Hering spoke of some possible sources of error in Threlfall's method, and recommended the use of gold electrodes, which he has found to be non-polarizable. Mr. Easterbrooks said that as high a degree of accuracy as is obtainable with Threlfall's method, is not necessary in the routine work of a metal refinery where quick volumetric methods are used for determining the composition of the electrolyte. An apparatus similar to that of Burgess' has its advantages in that it is simple in construction, and a curve can be plotted in less than half an hour.

Electrochemistry at St. Louis Exposition.

A paper on this subject, by Prof. C. F. Burgess, was read by title. It comprises a combination of the notes made by the group jury for electrochemistry in connection with its study of the electrochemical and electrometallurgical exhibits at the exposition.

Reversible and Irreversible Polarization.

A second paper by Prof. W. S. Franklin and L. A. Freudenberger was presented by Prof. Franklin. The first paper by the same authors on the same subject was abstracted on page 171 of our present volume. The fundamental idea of the authors is to consider the e. m. f. of an electrolytic cell as the sum of three factors:

$$E = Ri + f(i) + e,$$

where E is the total e. m. f., Ri is the e. m. f. consumed by the ohmic resistance, $f(i)$ the irreversible polarization, which is a function of the current, and e the reversible polarization, which is not a function of the current.

The authors formerly tried to measure the total e. m. f. across the cell at the instant of starting the current, so that the reversible polarization e would be zero; hence, knowing the value of Ri it would be possible to find the value of the irreversible polarization $f(i)$.

In order to find the original value of E at the instant of starting the current, the curve of growth of total voltage with the time was determined, and the intention was to exterpolate to zero time. But the curves given in the present paper show a sudden rise of voltage immeditely after the closing of the circuit, which makes it impossible to extend the curves to zero time with accuracy.

Another method was then tried by Messrs. Blume and Fleming, by which the reversible polarization e was to be determined with considerable accuracy, thus enabling the values of the irreversible polarization $f(i)$ to be inferred from the total e. m. f.

A certain value of current was started through the electrolytic cell and allowed to flow for t seconds, when the voltage between the cell terminals was measured. From the voltages so determined the calculated value of Ri is subtracted, and the result is plotted as ordinate of curves marked A in the paper. The abscissæ are values of t. The electrolyte was then vigorously stirred, the same value of current again started through the cell and allowed to flow for the same length of time, when the circuit was suddenly broken and the residual voltage between the cell terminals immediately measured. This residual voltage is assumed to be the value of the reversible polarization e at the instant t, and this residual voltage is plotted as an ordinate of curves marked B in the same diagram. The elapsed time between the breaking of the circuit and the measuring of the residual voltage was about 0.01 second.

The best that can be done to find the values of the irreversible polarization from these curves is to take the mean difference of the ordinates of the A and B curves.

The A and B curves were determined for up-flowing current as well as for down-flowing current. Horizontal parallel electrodes were used.

The paper was discussed at some length by Messrs. Burgess, Hering, Bancroft and Betts. Prof. Burgess suggested to use alternating current and to investigate the distortion of the wave form. He also doubted whether the transfer resistance between the electrode and electrolyte had been taken into account by the authors. Mr. Hering doubted whether the subject could be treated by means of such a simple formula as is the foundation of Prof. Franklin's method. The latter replied at some length.

Ammeters for Electrolytic Work.

A paper by Mr. Lawrence Addicks, of the De Lamar Copper Refinery, was read, in the absence of the author, by Mr. Easterbrooks. In view of the cost of power for a large proportion of the total cost in many electrolytic operations it is important to have the switchboard instruments correct to 99 per cent. Especially with the ammeters, certain precautions must be taken.

When several thousand amperes are to be measured, care should be taken to place both shunt and instrument in suitable positions when designing the switchboard. In the first place the shunt should be obtained from the maker with short bars inserted in the slots usually provided in the shunt terminals, these bars in turn to be bolted to the bus-bar carrying the current to be measured. In this way the shunt is calibrated with the same distribution of current through the various leaves as will be the case in after use.

Unequal heating of the two shunt terminals will introduce a constant error from the thermoelectric junctions formed where the alloy leaves are soldered onto the copper blocks. This may easily amount to 1 per cent of the full scale reading, may be either positive or negative, and will show as a zero error on shutting down the circuit, which will gradually disappear as the shunt cools off. Sometimes compensating devices are applied to avoid thermoelectric errors, but if the shunt is so placed in the bus-bar that the facilities for conducting away heat are approximately equal on both sides, there will be no appreciable error, and such devices should be unnecessary. The alloy leaves of a shunt would be red-hot at full load were it not for the heat dissipating qualities of the large copper terminals. There should be practically equal radiating surfaces of bus-bar for 3 or 4 feet each side of the shunt. If placed close to a switch on one side, there is sure to be a thermoelectric error in the shunt, as high capacity switches frequently run hot. The best plan is not to attempt to place the shunt back of the switchboard at all.

The temperature at which a shunt runs will repend upon how heavily it is loaded and the opportunity afforded to dissipate the heat generated. The temperature indicated by a thermometer and well placed upon one of the terminal blocks should not be allowed to exceed 200° F., owing to the danger of starting the solder and consequent failure.

The power wasted by a 5000-amp. shunt at full load is about 300 watts. It is perfectly practicable to build a shunt for 25,000 amps.

The author then passed over to a discussion of the instrument itself.

One of the most important sources of error is the magnetic effect of stray field in the immediate vicinity of a bar carrying several thousand amperes. Instruments should never be placed within 2 feet of such bars. The iron is not only likely to be faulty as a filter for lines of force of such density, but consequent poles are formed in the case itself. The author has found instruments placed in the vicinity of a stray field of such strength that he could hang his bunch of keys from the case.

The resistance of the leads is generally some 5 per cent of that of the instrument, and the screw connections must be kept clean. If it is found necessary to lengthen the leads, care must be taken that the resistance is kept constant by a proportionate increase in size of wire used.

The leads should be twisted and kept from swinging loosely in the presence of stray field.

When several high-capacity ammeters are in use, it is advisable to have them interchangeable.

Incandescent Lamps

A paper on the thermodynamics of the electric incandescent lamp was presented by Dr. E. F. Roeber, who also exhibited in operation the latest three types of incandescent lamps: the osmium lamp of Auer von Welsbach, the tantalum lamp of the Siemens & Halske Co. and the new graphitized filament lamp of the General Electric Co., invented in its research laboratory under the direction of Dr. Whitney.

The first part of the paper was a summary of the laws of radiation of a black body, as applied to electric incandescent lamps. A black body is one which is impartial in absorbing and emitting waves of all different wave lengths; it absorbs all, and therefore it emits all.

The author first connected the electrical energy supplied to the lamp with the total energy of radiation given out by the lamp according to the Stefan-Boltzmann law. The total radiation increases with the fourth power of the temperature. With increasing temperature the wave length, which has the maximum of energy, is shifted toward shorter wave lengths, *i. e.*, toward the visible spectrum, according to Wien's displacement law, which states that the product of absolute temperature and wave length of maximum energy is constant. The distribution of the energy within the total spectrum is given by laws of Wien and Planck. All these laws are strictly correct for the black body. They are the foundation of optical pyrometry, which is now assuming such importance in metallurgy. The author called attention to the excellent treatise on optical pyrometry by Waidner and Burgess in No. 2 of the Bulletin of the Bureau of Standards.

A review was then given of the researches tending to determine the mechanical equivalent of light, and it was explained that the physical and the physiological aspects of the question must be clearly distinguished. The author then discussed the possibilities of improving the efficiency of incandescent lamps. This is possible by increasing the temperature of the incandescent filament, which has been indeed the fundamental idea on which the osmium lamp and tantalum lamp were developed. However, in this direction it is practically impossible to get the maximum of the energy emitted by the filament within the visible spectrum. A second possibility is to get away from the black body and to use a filament with selective radiation. The departure of a filament from black-body radiation depends essentially on the substance used; it also depends, however, on the polished condition of the surface, since the more porous the surface of a filament, the more will it behave like a black body.

The necessity of using high temperatures and of employing pure substances (since the addition of any impurity lowers the melting point) indicates the intimate relation between the invention of new incandescent lamps with electrochemical research. The author showed the rôle which electrochemical methods have played in the invention of the three lamps which were exhibited. Concerning the tantalum lamp, see our March issue, page 118; concerning the graphitized filament, our July issue, page 252. In the preparation of the osmium filament, first threads are made consisting of porous rough osmium with an organic binding material; they are dried and heated in the absence of air to carburize the binding material. The carbon is then removed by heating the thread by the electric current in an atmosphere containing much steam and larger or smaller quantities of reducing gases. The carbon is removed by the same reaction by which water-gas is made.

The paper was discussed by Messrs. Bancroft, Hering, Franklin, Richards, Brown and Thatcher. The discussion dealt mainly with problems of optical pyrometry and with the question whether the black deposit on an incandescent lamp is due to actual volatilization of carbon. Prof. Franklin pointed out that for realizing selective radiation, gases appear to be most promising at present.

Radioactivity.

Two papers on this subject were presented by Prof. **Herman Schlundt** and Prof. **Richard B. Moore**. In the absence of the authors they were read in abstract by Prof. W. D. Bancroft.

The first paper discussed the chemical separation of the excited activity of thorium. The experimental results obtained by the authors show that a separation of the products constituting the excited activity of thorium can be readily made by chemical methods.

After removing the matter deposited on a negatively charged wire exposed for a long period to thorium emanation with acids, ThB may be obtained practically free from ThA with a precipitate of ferric hydroxide by means of pyridine or fumaric acid. The other component, ThA, remains in solution, and upon evaporation it remains as a residue. Ammonia does not make a separation.

By precipitating barium sulphate (or lead sulphate) in a solution of ThA and ThB, ThA is found in the precipitate. The residue obtained by evaporating the filtrate decays in activity very rapidly, falling to half value in about 56 minutes—the characteristic rate of decay of ThB. The precipitation of silver chloride in an acid solution of ThA and ThB removes very little of either of the components from the solution.

No differences were observed in conducting the separations immediately or after a lapse of time.

The residue from the solution of ThA and ThB obtained by boiling the foils in hydrochloric acid is probably more readily volatilized than the deposit itself or the residue from the nitric acid solution.

A partial separation of the two components of the excited activity may be made by direct treatment of the deposit with an alcohol-water solution of fumaric acid.

The results of the experiments are readily explained by the theory of Rutherford as recently altered by him and Miss Slater. The results of the present authors clearly show that the first stage of the matter constituting the excited activity of thorium is an inactive product, ThA, with the slow rate of change, and that this produces the radio-active product ThB, which changes comparatively rapidly into the final product ThC.

The results harmonize with the conclusion of Rutherford and Soddy, that in the separation of ThX from ordinary thorium compounds, both ThA and ThB are precipitated with the thorium, and the result of the authors, that in the fumaric acid and pyridine separation of ThX the component ThB of the excited activity is simply present in the thorium precipitate.

The second paper gave the results of tests of about twenty samples of mineral waters of Missouri for their radioactive properties. They are mostly mineral well waters, but some deep-well waters, used as water supplies, are also included. The results tabulated by the authors show that marked variations exist in the activity of these cold waters. As a rule, the activity of deep-well waters falls below that possessed by the spring waters.

Iodoform from Acetone.

A paper by Dr. **G. A. Roush**, on the electrolytic preparation of iodoform from acetone, was then presented. Abbot was the first to get a current efficiency of 60 per cent of iodoform from acetone by electrolysis in an alkaline solution, using a current density of 1.35 amps. per square decimeter and a temperature of 75° C. The catholyte was a 10 per cent solution of sodium carbonate. The anolyte contained 6 grams of sodium carbonate, 10 grams of potassium iodide in 100 cc. water; 5.5 cc. acetone were added at the rate of 0.5 cc. every 10 minutes during the electrolysis.

In 1903 Teeple worked out conditions by which he obtained a current efficiency calculated to be 94.5 per cent. He used no diaphragm, a low anode density and high cathode density, kept the solution cooled, well stirred and slightly acid in reaction. Since the reaction continually sets free alkali he neutralized it by adding gradually iodine during the electrolysis in just sufficient quantity to keep the solution slightly colored. Since this iodine, when added to the solution, forms iodoform, the method is really a combination of a chemical with an electrochemical process, although Teeple calculated his yield as a purely electrolytic yield.

The present author points out that the electrolysis of potassium iodide and acetone without a diaphragm gives two molecules of alkali for each molecule of iodoform produced. On the other hand, electrolysis with a diaphragm yields four molecules of acid for each molecule of iodoform produced. He, therefore, combines both reactions as follows:

The electrolyzing vessel was a crystallizing dish of 350 to 400 cc. capacity. The anode was a platinum crucible, presenting about 15 to 20 square centimeters of electrode surface, fixed on a shaft and rotated at a moderate speed. The rotating anode was used because it was the most efficient means of keeping the solution well stirred. Two cathodes were used. One was a platinum wire about 15 cm. long, 10 cm. of which were immersed in the electrolyte. The other cathode was of copper gauze of about 100 square centimeters surface and was placed in a porous cup. These are designated as the "regular" and "auxiliary" cathodes, respectively. Each cathode was connected to a separate resistance, so that the current could be regulated in one, independent of the other.

In this way the current entering the solution at the anode is divided into two parts, the first passing from the anode to the regular cathode, the second from the anode to the auxiliary cathode in the porous cup. Theoretically, if the current in the auxiliary cathode is one-half the current in the regular cathode, the solution will be kept neutral. In practice, however, the solution is to be kept slightly acid, and the process does not give absolutely theoretical yields; hence the current in the auxiliary cathode is about two-thirds that in the regular cathode, varying with the conditions.

The conditions which, according to the experiments of the author, give at the same time the highest current efficiency and the highest acetone yield, are as follows: Electrolyte, 20 grams KI; 300 cc. water; 1.5 cc. acetone; current, regular cathode, 1.5 amp.; auxiliary cathode, 1.0 amp.; temperature, 15 to 18° C.

Prof. W. D. Bancroft, in discussing this paper briefly, pointed out how much valuable work has been done in connection with this and kindred subjects in recent years—and all this work has been done in this country.

Resistivity of Iron and Steel at High Temperatures.

A note by Mr. **Gustave Gin** on the electric resistivity of iron and steel at high temperatures, was presented, in the absence of the author, by Prof. J. W. Richards. The author describes the arrangement of his tests and gives the result of the first measurements which he has made with cast iron containing 93.032 Fe, 2.752 Mn, 3.337 C, 0.783 Si, 0.061 P, 0.035 S. The resistivity of this iron was found to be 0.00016 microhm-centimeters for temperatures between 1.280° C. and 1.340° C.

Prof Richards welcomed this paper as one furnishing empirical data of value for metallurgical calculations. He hoped that a great many similar papers would follow.

A vote of thanks, moved by Prof. C. F. Burgess, and seconded by Mr. Carl Hering, and extending the thanks of the society to all those who had made this meeting so successful, especially to the local committee, to the authorities of Lehigh University and to the officers of the different plants visited, was carried enthusiastically.

Before the meeting adjourned, the Society was honored by the presence of Mr. **John Fritz**, the famous and venerable steel metallurgist, who came to greet the members of the Society.

The excursion to Mauch Chunk for the afternoon had to be given up on account of the threatening weather. Those who remained for the afternoon in Bethlehem attended the impressive opening services in the University Chapel of Lehigh University. Others paid a visit to the plant of the Dexter Portland Cement Company, of Nazareth, Pa. The company met the guests with a special train and conveyed them to the works, some little distance from Nazareth. Dr. R. K. Meade, the chemist of the company, took charge of the guests and showed them around the plant. They started in at the quarry from which the raw material is taken, and followed the same through the plant, first through the grinding department and then the kiln, and finally the final grinding and packing rooms. In all it was a most pleasurable visit, many of the members seeing the manufacture of cement for the first time.

In the following we give an alphabetical list of all gentlemen who registered at the meeting:

C. E. Acker, G. P. Adamson, W. D. Bancroft, F. M. Becket, A. G. Betts, G. Boericke, C. S. Bradley, Wm. Hand Browne, Jr., C. F. Burgess, C. F. Carrier, H. B. Coho, F. Conlin, C. A. Doremus, G. Drobegg, F. D. Easterbrooks, William Esty, Ernest Fahrig, W. S. Franklin, L. A. Freudenberger, W. C. Geer, J. H. Granbery, E. I. Harrington, E. Hart, W. Homer Hendricks, Carl Hering, H. C. Huettig, W. S. Landis, A. B. Larchar, H. T. Matthew, B. MacNutt, John Meyer, J. Y. McConnell, H. Phillip, A. G. Rau, J. W. Richards, E. F. Roeber, C. J. Russell, T. Rynard, S. S. Sadtler, J. W. Schade, F. F. Schuetz, William Smith, C. C. Speiden, M. Toch, F. J. Tone, R. von Foregger, A. von Isakovics, Leonard Waldo, E. H. Whitlock, W. E. Winship.

Lehigh University.—The twenty-sixth celebration of Founder's Day will take place on Oct. 12. The Founder's Day address will be delivered by the Hon. Hampton L. Carson, Attorney-General of Pennsylvania. In connection with these exercises will occur the inauguration of Mr. Henry Sturgis Drinker as president of the University. The installation address will be delivered by Mr. Robert H. Sayre, president of the Board of Trustees, while Mr. Frank P. Howe, Lehigh '78, will [illegible] for the alumni. These exercises will be followed by the turning of the sod for Drown Hall.

Action of Acids on Iron and the Use of the Acid Pickle.

By Charles F. Burgess.

(Concluded from page 335.)

Hydrochloric and Sulphuric Acids.

Hydrochloric and sulphuric acids are the most important materials used in pickling, the latter being far in the lead so far as the total quantity consumed is concerned. The pickler has his choice limited, practically, to these two acids, though he has a wide range of densities and purities to select from. The determination of what is the best composition for solution is influenced by a number of factors, including the relative costs of the acids, the nature of the iron or steel to be treated, the quantity and thickness of the scale, the purpose for which the cleaning is to be done, the cost of labor, the size of plant and the utilization or disposal of waste.

So far as cost is concerned, sulphuric acid is ordinarily considerably cheaper than hydrochloric. The former is obtainable at a density which closely approaches 100 per cent acid, while the hydrochloric acid solution contains about 40 per cent of dry HCl gas. On the basis of pure acid content, 1 pound of pure sulphuric acid has the capacity of dissolving .57 of a pound of iron, while a pound of pure hydrochloric acid will dissolve .77 of a pound. One pound of commercial hydrochloric acid has the dissolving power, as compared with sulphuric acid, of about 10 to 19. There are certain localities where muriatic acid can be produced as a by-product and supplied at a much lower price than that asked for sulphuric acid, but where transportation charges must be reckoned upon as a factor, the price of hydrochloric acid increases much more rapidly than does that of sulphuric acid, since it must be transported in glass-containing vessels, while the other may be shipped in large iron tank cars.

The determination of the strength of acid solutions to be employed is a most important factor, since upon it depends the rapidity with which the work may be done, and, therefore, the size of the plant which becomes necessary. It is generally assumed that the rapidity with which a solution will act will be somewhere nearly proportional to the percentage of acid in aqueous solution. That this rule can be applied only with reservations, at least so far as sulphuric acid is concerned, is shown by some important data which R. Kneitsch has given in a paper on "Sulphuric Acid and Its Manufacture" in Vol. X. of *The Mineral Industry.* He studied the action of various strengths of acids upon wrought iron and cast iron, and gives data and curves showing in what manner the rate of corrosion varies with the density. The curve showing the rate of corrosion of sulphuric acid in progressively increasing strengths is not a simple one, possessing a number of maxima and minima.

Some interesting comments are made by the same author upon the action of the acids upon cast iron. He states that while cast iron vessels are well suited to using for watery sulphuric acid, they cannot be used with fuming acids, for though cast iron is attacked but little, what is worse, it explodes. This remarkable property is based upon the fact that the fuming acid, diffusing in the pores of the cast iron is reduced to sulphur dioxide and hydrogen sulphide, sometimes even generating carbon dioxide from the carbon of the iron. All of these are gases, the critical temperatures of which are low, and which, as a result, cause high tensions within the iron.

A curve which would show the action of various strengths of hydrochloric acid would not be so complicated as the one given for sulphuric acid. This is probably due to the fact that the strongest liquid hydrochloric acid that is available contains only 40 per cent of HCl, while sulphuric acid is readily obtainable up to 100 per cent strength, and is even then capable of dissolving a considerable amount of additional SO_3. At no point in the concentration of HCl does iron become passive, and it is for this reason that iron vessels cannot be used for its storage and transportation as is done with sulphuric acid.

Practice shows that hydrochloric acid can be used in a higher degree of dilution than can sulphuric acid, and this constitutes a marked advantage in its favor. The action of a less than 10 per cent solution of sulphuric acid becomes so slow as to be prohibitive in most cases, and the pickle, therefore, becomes "spent" while there is yet a considerable percentage of free acid in the solution. Consequently, the available part of the acid is that which lies above the percentage at which the action on the iron becomes so slow as to prohibit its use. The amount of free acid which goes to waste in the spent pickling liquor constitutes, on this account, a large item in the total consumption of acid. Such waste may be decreased somewhat by heating the solution. With hydrochloric acid, on the other hand, a more dilute solution retains its activity, and consequently the acid may be used to a higher degree of efficiency.

It is a noticeable fact that most of the authorities, while stating what strength their pickling solutions should have, neglect to give the degree to which it may be exhausted. This is a most important factor in calculating for acid consumption, and it is a point upon which additional information is very desirable. While investigations have been made to determine the rate at which freshly prepared acid solutions will dissolve iron, the writer has been unable to find any data showing the influence which dissolved iron salts in a solution may have in affecting this rate. As pointed out heretofore, it is a well-known fact that the presence of metallic salts in an acid solution will have a marked effect on the activity of the solution, but an investigation of the effect of the dissolved iron salts is a subject worthy of further effort.

To increase the percentage consumption of the acid, the freshly prepared solution should be as concentrated as is possible and compatible with vigorous and effective corrosion, for in this manner the percentage of acid which is lost in the waste liquor bears its smallest ratio to the total amount of acid employed. A limitation, however, is placed upon the density available by the solubility of the iron salts which are formed, since, if this reaches the saturation point, the action of the acid is greatly diminished or ceases, this being due to the formation of insoluble iron salts on the surface of the metal. The chlorides of iron being much more soluble than the sulphates, the hydrochloric acid solution may have a higher concentration.

The use of temperatures considerably above ordinary room temperatures, which is recommended by many picklers, is due to the well-known fact that the activity of an acid is increased at higher temperatures, and also because the products of corrosion have their solubility increased in a like degree. Heating, then, enables the acid to be used more efficiently and effects a saving in the size of the plant necessary, but it is not always employed, on account of the complications arising in the construction of the plant by the introduction of heating coils or the changes in construction necessary to make the containing vessels withstand the action of the acid. The best way to effect such a heating is to inject steam directly into the solution, but this is not always permissible on account of the diluting effect of the condensed steam.

The purity of the acid used is a matter to which, of late, picklers have given more attention than formerly, as it is now recognized that this detail plays a most important part in the economy of the process. For a long time it has been known that the acids obtained from some manufacturers are more satisfactory than those made by others, though the reason for this difference in quality has not always been understood by the users. The impurity which seems to affect in the greatest degree the usefulness of sulphuric acid is arsenic, and frequently picklers now specify that the acid they purchase shall be free from this impurity. Such acid, however, usually commands a higher price, which in certain cases is quite justifiable. As previously pointed out, the effect of arsenic in an acid solution is to decrease the rate at which it will attack iron, and since upon the attack on the iron depends the removal of the mill scale the presence of arsenic greatly interferes with such removal. But while arsenic is decidedly disadvantageous in such connection, it may play a useful part under other conditions which will be dwelt with later. Arsenic is an impurity which is found to greater or less degree in acids made from pyrites, and is not usually present in that made from burning sulphur. This accounts for the preference which is commonly given to brimstone acid.

In addition to decreasing the activity of the acid, the arsenic is by some supposed to be responsible for the black stains which remain on the iron after the scale has been removed. Just why the presence of arsenic interferes with the action of the acid, or just what is the nature of the black deposit which it forms, are matters that do not seem to be well understood, and which are, consequently, fruitful subjects for further investigation. If it could be shown that the addition of arsenic prevents the corrosion of the iron, while at the same time it does not diminish the dissolution of the oxide, we would have a possible method pointed out for obtaining a pickling solution closely approaching the ideal. There seems to be, however, no available data showing the relative influence of arsenic on the solubility of iron and its oxides. As to the black stains for which it is supposed to be responsible, it is probable that remedies could be discovered as soon as their nature and causes are better understood. Some investigations which are now under way in the writer's own laboratory indicate that certain substances other than arsenic may be productive of all the advantages exercised by arsenic with entire freedom from its disadvantages.

Most of the other impurities commonly found in acids do not play such important parts. The presence of copper, silver and platinum, it is well known, increase the rate of attack on iron, but this increase does not seem to be advantageous in pickling, since it increases the rate of corrosion on the iron which has been cleaned of scale, while it does not increase the rate at which such scale is removed. Iron will readily dissolve in a copper chloride solution, but without any appreciable removal of scale, since there is no liberation of hydrogen and the mechanical action of hydrogen in forcing the scale away from the iron is an essential in the pickling process.

The process for pickling iron and steel presents, as has been indicated, various problems of both interest and importance, the solution of which will contribute to the improvement of methods now in vogue. One of the most important ways in which present methods could be changed for the better would be in a reduction of acid consumption. An ideal pickle would be one in which all of the acid is utilized by being consumed in the dissolution of rust and scale without dissolving any of the iron. This seems to be impossible of attainment, since we know of no solvent for iron oxide which is not a better solvent for the iron itself. But laying aside this factor, material improvement in the manner of operation now followed could be effected by more attention being paid to the choice of a suitable density of solution and to its heating. One of the ways in which acid is needlessly wasted is in allowing the portions of undissolved scale which become detached to fall to the bottom of the tank and remain there, the slow action of the acid continuing to dissolve the material and the acid thereby being wasted. Cowper-Cowles has devised an ingenious arrangement for diminishing this loss by providing a magnetic scale collector which removes the black magnetic iron oxide from time to time as it settles to the bottom of the tank (Electrochemical Industry, March, 1903, p. 263).

The reduction of labor is another important desideratum, since the frequent handling of the material, both in the cleaning operation and in its subsequent brushing and drying, involves a large item of expense. Automatic machinery is being introduced for this purpose to some extent, but under present [illegible] it does not seem possible to greatly decrease the amount of labor required. The greatest trouble appears to lie in connection with the washing of the iron after pickling, the neutralization of the remaining traces of the acid and the scouring necessary to remove the adhering particles of insoluble material.

Although little attention has been given here to the operations following the actual pickling process, they must be carefully studied. It is known that with a given grade of iron one acid will leave a cleaner surface than another acid, and the reason back of this demands a study of the action of acids upon the various constituents of iron and steel. Even after the removal of the oxide from the surface a coating of more or less adherent particles will be found, which have a black, brown or grayish appearance, varying with the character of the metal. This deposit is commonly supposed to be due to the carbon which was originally contained in the iron.

Within recent years some important work has been done along the line of investigating the action of various acids upon the innumerable constituents of iron and steel. While the result of this research has been to throw additional light upon the physical and chemical composition of iron, it also furnishes information of use to the pickler in studying the various problems which present themselves. An excellent summary of this work is given by Jüptner in "Siderology: The Constitution of Iron Alloys and Slag."

The technical varieties of iron contain a number of other elements, the chief ones being carbon, silicon, sulphur, phosphorus and manganese. These substances may be either physically or chemically combined with iron, and among the properties which they influence may be included that of corrodibility by acids. Carbon exists in various forms, including graphite, graphitic tempering carbon, hardening carbon and carbide carbon of various kinds. Since graphite is unattacked even by certain boiling concentrated acid, this material, when present in iron, is left as a black deposit upon corrosion by such acids. Graphitic tempering carbon behaves as does graphite in the presence of acids. Carbide carbon in which the carbon is in chemical union with the iron, and which exists in all kinds of commercial iron, when heated with strong acids disengages the carbon as a hydrocarbon. If a specimen of iron containing it is dissolved in very dilute hydrochloric or sulphuric acid at ordinary temperatures, with exclusion of air, the carbide is left behind in a brown or gray mass, which, according to Müller (*Stahl und Eisen*, 1888, p. 292), consists of granules of silvery luster, igniting spontaneously at a relatively low temperature on drying. On dissolving the sample of iron in cold dilute nitric acid of 1.2 s. g., a flocculent brown residue forms. Hardening carbon on dissolving in dilute hydrochloric or sulphuric acid gives off a strong-smelling hydrocarbon gas, and in cold nitric acid leaves a deep black residue.

The amount of residue which is left on dissolving by acids decreases with the concentration of the acid used. Nitric acid leaves more residue than do the other acids, as there is no liberation of hydrogen with which the combined carbon can unite to form hydrocarbon gas. Acids of different strengths leave cast iron with various appearances, since each form of carbon requires a special strength of acid to liberate it from its [illegible]

From observations such as these we may draw explanation for many peculiarities noted in the operation of pickling processes. It can be seen why a piece of iron before and after being hardened is affected differently by the acid, why one acid will leave the iron perfectly clean, while another will leave it covered with a black, powdery deposit, and why strength of [illegible] ns additional argument for the [illegible]e of hydrochloric rather than sulphuric acid, since it is seen [illegible]

If we may judge from the patent literature and from various articles in the technical press, various attempts have been made to employ the electric current as an aid in pickling. By using the article from which the scale is to be removed as the anode in a solution of sodium chloride, sodium sulphate or other suitable electrolyte, the iron may be corroded, but the fatal defect in this process is that the scale is not thereby loosened, since there is no liberation of hydrogen and consequently no tendency towards lifting off the scale. If the current is made to take a reverse direction and hydrogen is liberated upon the iron, serving as a cathode, the object sought is not attained, since the hydrogen is liberated, to a large extent, on the outside of the scale, which is a good conducting medium. The current is thus used to little advantage. The action of the ordinary pickling solutions may be facilitated by the action of the electric current, but the expense of the electrical energy and the difficulty of connecting the material to be treated as an electrode are such as to make the use of the current impracticable.

When a pickle has been used long enough to become spent, or exhausted, the question arises as to what shall be done with it. If it is to be simply thrown away, the locality determines whether the free acid be neutralized or not. Neutralization can be most cheaply effected by lime, which gives an insoluble precipitate with the sulphate solution and a soluble salt with the chloride. The spent solution contains, however, materials of value, and much attention has been given to methods of utilizing them. With sulphate solutions, the free acid may be neutralized by scrap iron and the resultant solution evaporated down until the ferrous sulphate crystallizes. This method furnishes most of the ferrous sulphate now upon the market. The supply of this material, however, exceeds the demand, so that the revenue derived from its sale does not pay for the acid consumed, it becoming frequently a doubtful matter as to whether the additional cost of plant and labor is warranted by the results obtained. Only the larger plants find the method profitable.

Many plans have been proposed for making a profit from this waste material, but the cost of purification, evaporation, crystalization, filtration and the like is usually such as to justify the pickler in adopting the cheapest and least objectionable method of disposal.

From the many difficulties encountered in the process of acid pickling comes the incentive to depart completely from the chemical method and to employ the sand blast instead. The operation of this method consists in forcing particles of sand, by means of an air blast, against the surface to be cleaned. The sand quickly breaks up and removes the scale and leaves clean the underlying surface. The only objections to its use is that the expense is greater on account of the increased cost of labor which it involves. However, the use of the sand blast is increasing, and with the improvements in its operation which may confidently be looked for, it is likely to become before long a pronounced competitor of the chemical methods.

It has been known for a long time that hydrogen is readily absorbed by iron under certain circumstances, and that this element has a remarkable influence upon the properties of the iron, which it renders brittle, reducing its strength and elasticity. The hydrogen which is liberated when an iron article is pickled is absorbed to a certain extent by the iron, and this constitutes another one of the many difficulties attendant upon a thoroughly successful pickling operation. An iron or steel wire of high grade may be rendered so brittle by a few moments' immersion in an acid solution as to make it worthless for the purpose for which it was originally intended, so that on account of this "rotting" it is impracticable to pickle certain grades of iron. An investigation of this problem will constitute the subject matter of a subsequent article.

Chemical Engineering Laboratories,
[illegible]
Madison, Wis.

Metallurgical Calculations.—VIII.

By J. W. Richards, Ph. D.

Professor of Metallurgy in Lehigh University.

THERMO-PHYSICS OF COMPOUNDS.

(*Continued.*)

There are a few other compounds than those previously mentioned whose thermophysics has been investigated, but their number is in reality infinitesimal compared with the number of those which have not been touched. There are many compounds of great importance in metallurgy whose specific heats are not known, to say nothing of their latent heats of fusion, etc. The wide introduction of the electric furnace has rendered desirable the latent heats of vaporization and specific heats at high temperatures, but these are altogether lacking; we can only estimate their values. It is to be hoped that many metallurgical laboratories may be incited to take up this very neglected field, and thus bring forward numerical data which would be of the greatest theoretical and practical value. One example of such work, which deserves special commendation, is the quite recent publication of Prof. J. H. L. Vogt, of Christiania, on the "Silikatschmelzlösungen," *i. e.*, on "Melted Silicate Solutions," in which determinations of the melting points and latent heats of fusion of many simple and complex silicates are given for the first time; we venture to predict that the data and conclusions of Prof. Vogt will be of great value to the physicist, chemist, metallurgist and geologist, in both a practical as well as a theoretical sense.

In collating the remaining available data, it is rather difficult to decide as to what has immediate metallurgical interest and what has not. Electrometallurgy, in particular, is busying itself with the treatment and decomposition of so many compounds heretofore considered outside of the metallurgists sphere of interest, that almost all of the common chemical compounds now have either a present or a prospective interest to the metallurgist.

Thermophysics of Chlorides.

Hydrogen—HCl (gas) Sm per kilo. (22° — 214°)
= 0.1867 (Regnault)
per m^3 (22° — 214°)
= 0.3067 (Regnault)
Lithium—LiCl (solid) Sm per kilo. (13° — 97°)
= 0.2821 Cal. (Regnault)
Carbon—CCl^4 (liq.) L. H. vaporization at 0°
= 52.0 Cal. (Regnault)
Ammonium—NH^4Cl (solid) Sm per kilo. (23° — 100°)
= 0.3908 Cal. (Neumann)
Sodium—NaCl (solid) Sm per kilo. (15° — 98°)
= 0.2140 Cal. (Regnault)
Magnesium—$MgCl^2$ (solid) Sm per kilo. (24° — 100°)
= 0.1946 Cal. (Regnault)
Silicon—$SiCl^4$ (liq.) Sm per kilo. (10° — 15°)
= 0.1904 Cal. (Regnault)
$SiCl^4$ (gas) Sm per kilo.^{3} (90° —234°)
= 0.1322 Cal. (Regnault)
$SiCl^4$ (gas) Sm per m^3 (90° — 234°)
= 1.0113 Cal. (Regnault)
Potassium—KCl (solid) Sm per kilo. (14° — 99°)
= 0.1730 Cal. (Regnault)
Calcium—$CaCl^2$ (solid) Sm per kilo. (23° — 99°)
= 0.1642 Cal. (Regnault)
Titanium—$TiCl^4$ (solid) Sm per kilo. (13° — 99°)
= 0.1881 Cal. (Regnault)
$TiCl^4$ (gas) Sm per kilo. (163° — 271°)
= 0.1290 Cal. (Regnault)
Chromium—$CrCl^2$ (solid) Sm per kilo. (?)
= 0.1430 Cal. (Kopp)
Manganese—$MnCl^2$ (solid) Sm per kilo. (?)
= 0.1425 Cal. (Regnault)
L. H. vaporization
= 49.37 Cal. (Ogier)
Copper—CuCl (solid) Sm per kilo. (17° — 98°)
= 0.1383 Cal. (Regnault)
Zinc—$ZnCl^2$ (solid) Sm per kilo. (21° — 99°)
= 0.1362 Cal. (Regnault)
Arsenic—$AsCl^3$ (solid) Sm per kilo. (14° — 98°)
= 0.0896 Cal. (Regnault)
(gas) Sm per kilo. (159° — 268°)
= 0.1122 Cal. (Regnault)
L. H. vaporization
= 69.74 Cal. (Regnault)
Strontium—$SrCl^2$ (solid) Sm per kilo. (13° — 98°)
= 0.1199 Cal. (Regnault)
Silver—AgCl (solid) Sm per kilo. (15° — 98°)
= 0.0911 Cal. (Regnault)
(160 °— 380°)
= 0.0978 Cal. (Ehrhardt)
Tin (ous)—$SnCl^2$ (solid) Sm per kilo. (20° — 99°)
= 0.1016 Cal. (Regnault)
Tin (ic)—$SnCl^4$ (liq.) Sm per kilo. (14° — 98°)
= 0.1476 Cal. (Regnault)
(gas) Sm per kilo. (149° — 273°)
= 0.0939 Cal. (Regnault)
L. H. vaporization at 112°
= 46.84 Cal. (Regnault)
Barium—$BaCl^2$ (solid) Sm per kilo. (14° — 98°)
= 0.0896 Cal. (Regnault)
Mercurous—HgCl (solid) Sm per kilo. (7° — 99°)
= 0.0521 Cal. (Regnault)
Mercuric—$HgCl^2$ (solid) Sm per kilo. (13° — 98°)
= 0.0689 Cal. (Regnault)
Lead—$PbCl^2$ (solid) Sm per kilo. (20° — 100°)
= 0.0651 Cal. (Luginin)
(160° — 380°)
= 0.0707 Cal. (Ehrhardt)
(liq.) Sm per kilo. above 485°
= 0.1035 Cal. (Ehrhardt).
L. H. fusion at 485°
= 20.90 Cal. (Ehrhardt)

Thermophysics of Bromides.

Hydrogen—HBr (gas) Sm per kilo. (11° — 100°)
= 0.0820 Cal. (Strecker)
Sm per m^3 (11° — 100°)
= 0.2989 Cal. (Strecker)
Sodium—NaBr Sm per kilo. (?)
= 0.1384 Cal. (Regnault)
Potassium—KBr (solid) Sm per kilo. (16° — 98°)
= 0.1132 Cal. (Regnault)
Silver—AgBr (solid) Sm per kilo. (15° — 98°)
= 0.0739 Cal. (Regnault)
Lead—$PbBr^2$ (solid) Sm per kilo. (16° — 98°)
= 0.0532 Cal. (Regnault)
(190° — 430°)
= 0.0532 Cal. (Ehrhardt)
L. H. fusion at 490°
= 12.34 Cal. (Ehrhardt)

Thermophysics of Iodides.

Hydrogen—HI (gas) Sm per kilo. (21° — 100°)
= 0.0550 Cal. (Strecker)
Sm per m^3 (21° — 100°)
= 0.3168 Cal. (Strecker)
Sodium—NaI (solid) Sm per kilo. (16° — 99°)
= 0.0868 Cal. (Regnault)
Potassium—KI (solid) Sm per kilo. (20° — 99°)
= 0.0819 Cal. (Regnault)
Cuprous—CuI (solid) Sm per kilo. (18° — 99°)
= 0.0687 Cal. (Regnault)

Silver—AgI (solid) Sm per kilo. (15° — 204°)
= 0.0577 Cal. (Bellati and Romanese)

Mercurous—HgI (solid) Sm per kilo. (17° — 99°)
= 0.0395 Cal. (Regnault)

Mercuric—HgI^2 (solid) Sm per kilo. (18° — 99°)
= 0.0420 Cal. (Regnault)

Lead—PbI^2 (solid) Sm per kilo. (14° — 98°)
= 0.0427 (Cal. (Regnault)
(160° — 315°) = 0.0430 (Cal. (Ehrhardt)
(liq.) above 375° = 0.0645 Cal. (Ehrhardt)
L. H. fusion at 375° = 11.50 Cal. (Ehrhardt)

Thermophysics of Fluorides.

Sodium-Aluminium—$3NaF. AlF.^3$ Sm per kilo. (16° — 99°)
(Cryolite) = 0.2522 (Oeberg)

Calcium (Fluorspar)—$CaF.^2$ Sm per kilo. (15° — 99°)
= 0.2154 (Regnault)

Thermophysics of Sulphate.

Hydrogen—H^2SO^4 Sm per kilo. (5° — 22°)
= 0.332 (Cattanes)
L. H. vaporization at 326°
= 122.1 Cal. (Person)

Sodium—Na^2SO^4 Sm per kilo. (17° — 98°)
= 0.2312 (Regnault)

Magnesium—$MgSO^4$ Sm per kilo. (25° — 100°)
= 0.2250 (Pape)

Potassium—K^2SO^4 Sm per kilo. (15° — 98°)
= 0.1901 (Regnault)
$KHSO^4$ Sm per kilo. (19° — 51°)
= 0.2440 (Kapp)

Calcium—$CaSO^4$ Sm per kilo. (13° — 98°)
= 0.1965 (Regnault)

Manganese—$MnSO^4$ Sm per kilo. (21° — 100°)
= 0.1820 (Pape)

Nickel—$NiSO^4$ Sm per kilo. (15° — 100°)
= 0.2160 (Pape)

Copper—$CuSO^4$ Sm per kilo. (23° — 100°)
= 0.1840 (Pape)

Zinc—$ZnSO^4$ Sm per kilo. (22° — 100°)
= 0.1740 (Pape)

Strontium—$SrSO^4$ Sm per kilo. (2° — 99°)
= 0.1428 (Regnault)

Barium—$BaSO^4$ Sm per kilo. (10° — 98°)
= 0.1128 (Regnault)

Lead—$PbSO^4$ Sm per kilo. (20° — 99°)
= 0.0872 (Regnault)

Thermophysics of Nitrates.

Hydrogen—HNO^3 L. H. vaporization at 326°
= 115.1 Cal. (Person)

Ammonium—NH^4NO^3 Sm per kilo. (14° — 31°)
= 0.4550 (Kopp)

Sodium—$NaNO^3$ Sm per kilo. (14° — 98°)
= 0.2782 (Regnault)
$NaNO^3$ (fluid) Sm per kilo. (320° — 430°)
= 0.4130 (Person)
$NaNO^3$ L. H. fusion at 305.5°
= 64.87 Cal. (Person)

Potassium—KNO^3 Sm per kilo. (13° — 98°)
= 0.2387 (Regnault)
KNO^3 (fluid) Sm per kilo. (350° — 435°)
= 0.3319 (Person)
KNO^3 L. H. fusion at 333.5°
= 48.90 Cal. (Person)

Sodium-potassium—$(K:Na)NO^3$ Sm per kilo. (15° — 100°)
= 0.2350 (Person)

Strontium—$Sr(NO^3)^2$ Sm per kilo. (17° — 47°)
= 0.1810 (Kopp)

Silver—$AgNO^3$ Sm per kilo. (16° — 99°)
= 0.1435 (Regnault)

Barium—$Ba(NO^3)^2$ Sm per kilo. (13° — 98°)
= 0.1523 (Regnault)

Lead—$Pb(NO^3)^2$ Sm per kilo. (17° — 100°)
= 0.1173 (Neumann)

Thermophysics of Carbonates.

Sodium—Na^2CO^3 Sm per kilo. (16° — 98°)
= 0.2728 (Regnault)

Potassium—K^2CO^3 Sm per kilo. (23° — 99°)
= 0.2162 (Regnault)

Calcium (calcite)—$CaCO^3$ Sm per kilo. (20° — 100°)
= 0.2086 (Regnault)

Calcium (aragonite)—$CaCO^3$ Sm per kilo. (18° — 99°)
= 0.2085 (Regnault)

Calcium (marble)—$CaCO^3$ Sm per kilo. (23° — 98°)
= 0.2099 (Regnault)

Calcium-magnesium—$CaMg(CO^3)^2$ Sm per kilo. (20° — 100°)
= 0.2179 (Regnault)

Magnesium-iron—$Mg^7Fe^2(CO^3)^9$ Sm per kilo. (17° — 100°)
= 0.2270 (Neumann)

Iron (siderite)—$FeCO^3$ Sm per kilo. (9° 98°)
= 0.1935 (Regnault)

Strontium—$SrCO^3$ Sm per kilo. (8° — 98°)
= 0.1475 (Regnault)

Barium—$BaCO^3$ Sm per kilo. (11° — 99°)
= 0.1104 (Regnault)

Lead (cerussite)—$PbCO^3$ Sm per kilo. (16° — 47°)
= 0.0791 (Kopp)

Thermophysics of Chromates.

Potassium—K^2CrO^4 Sm per kilo. (19° — 98°)
= 0.1851 (Regnault)

Potassium—$K^2Cr^2O^7$ Sm per kilo. (16° — 98°)
= 0.1894 (Regnault)

Lead—$PbCrO^4$ Sm per kilo. (19° — 50°)
= 0.0900 (Kopp)

(Chromite)—$FeCrO^4$ Sm per kilo. (19° — 50°)
= 0.1590 (Kopp)

Thermophysics of Borates.

Sodium—$Na^2B^2O^4$ Sm per kilo. (17° — 97°)
= 0.2571 (Regnault)
$Na^2B^4O^7$ Sm per kilo. (16° — 98°)
= 0.2382 (Regnault)

Potassium—$K^2B^2O^4$ Sm per kilo. (16° — 98°)
= 0.2048 (Regnault)
$K^2B^4O^7$ Sm per kilo. (18° — 99°)
= 0.2198 (Regnault)

Lead—PbB^2O^4 Sm per kilo. (15° — 98°)
= 0.0905 (Regnault)
PbB^4O^7 Sm per kilo. (16° — 98°)
= 0.1141 (Regnault)

Thermophysics of Phosphates.

Sodium—$Na^4P^2O^7$ Sm per kilo. (17° — 98°)
= 0.2283 (Regnault)

Potassium—$K^4P^2O^7$ Sm per kilo. (17° — 98°)
= 0.1901 (Regnault)

Calcium—CaP^2O^6 Sm per kilo. (15° — 98°)
= 0.1992 (Regnault)

(Apatete)—$3Ca^3P^2O^8.CaF^2$ Sm per kilo. (15° — 99°)
= 0.1903 (Oeberg)

Silver—Ag^3PO^4 Sm per kilo. (19° — 50°)
= 0.0898 (Kopp)

Lead—$Pb^2P^2O^7$ Sm per kilo. (11° — 98°)
= 0.0821 (Regnault)

Lead—$Pb^3P^2O^8$ Sm per kilo. (11° — 98°)
= 0.0798 (Regnault)

Thermophysics of Aluminates, etc.

Spinell—$MgAl^2O^4$ Sm per kilo. (15° — 47°)
= 0.1940 (Kopp)

Chrysoberyl—$BeAl^2O^4$ Sm per kilo. (0° — 100°)
= 0.2004 (Nillson and Pettersson)

Ilmenite—$FeTiO^3$ Sm per kilo. (15° — 50°)
= 0.177 (Kopp)

Wulfenite—$PbMoO^4$ Sm per kilo. (15° — 50°)
= 0.083 (Kopp)

Scheelite—$CaWO^4$ Sm per kilo. (15° — 50°)
= 0.097 (Kopp)

Wolframite—$Fe(Mn)WO^4$ Sm per kilo. (15° — 50°)
= 0.098 (Kopp)

$KMnO^4$ Sm per kilo. (15° — 50°)
= 0.179 (Kopp)

$KClO^3$ Sm per kilo. (10° — 100°)
= 0.210 (Regnault)

Miscellaneous.

Glass—Ca. K Silicate Sm per kilo. (14° — 99°)
= 0.1977 (Regnault)
Sm per kilo. (0° — 300°)
= 0.190 (Dulong and Petit)

Flint Glass—Sm per kilo. (10° — 50°)
= 0.117 (H. Meyer)

Crown Glass—Sm per kilo. (10° — 50°)
= 0.161 (H. Meyer)

Paraffin—(solid) Sm per kilo. (10° — 15°)
= 0.562 (Battelli)
Sm per kilo. (35° — 40°)
= 0.622 (Battelli)
(fluid) Sm per kilo. (52° — 63°)
= 0.706 (Battelli)

Beeswax—(solid) Sm per kilo. (26° — 42°)
= 0.820 (Person)
Sm per kilo. (42° — 58°)
= 1.720 (Person)
(fluid) Sm per kilo. (65° — 100°)
= 0.499 (Person)

Vulcanite—Sm per kilo. (20° — 100°)
= 0.331 (A. M. Mayer)

Soft Para Rubber—Sm per kilo. (0° — 100°)
= 0.481 (H. Gee and Terry)

Thermophysics of Sulphides.

Hydrogen—H^2S (gas) Sm per kilo. (20° — 206°)
= 0.2451 (Regnault)
Sm per m^3 (20° — 206°)
= 0.3750 Cal. (Regnault)

Carbon—CS^2 (liq.) Sm per kilo. (14° — 29°)
= 0.2468 (Person)
(gas) Sm per kilo. (86° — 190°)
= 0.1596 (Regnault)
Sm per m^3 (86° — 190°)
= 0.5458 (Regnault)
L. H. vaporization at 46°
= 83.8 Cal. (Wirtz)

Manganese—MnS Sm per kilo. (10° — 100°)
= 0.1392 (Sella)

Iron—FeS Sm per kilo. (17° — 98°)
= 0.1357 (Regnault)
(Pyrrhotite) Fe^7S^8 Sm per kilo. (20° — 100°)
= 0.1602 (Regnault)
(Pyrites) FeS^2 Sm per kilo. (19° 98°)
= 0.1301 (Regnault)

Nickel—NiS Sm per kilo. (15° — 98°)
= 0.1281 (Regnault)

Cobalt—CoS Sm per kilo. (15° — 98°)
= 0.1251 (Regnault)

Copper—Cu^2S Sm per kilo. (9° — 97°)
= 0.1212 (Regnault)
Sm per kilo. (0° — t°) = 0.1126 + 0.00009t
(Bellati and Lussana)

Zinc—ZnS Sm per kilo. (15° — 98°)
= 0.1230 (Regnault)

Arsenic—AsS Sm per kilo. (20° — 100°)
= 0.1111 (Neumann)
As^2S^3 Sm per kilo. (20° — 100°)
= 0.1132 (Naumann)

Molybdenum—MoS^2 Sm per kilo. (20° — 100°)
= 0.1233 (Regnault)

Silver—Ag^2S Sm per kilo. (7° — 98°)
= 0.0746 (Regnault & Sella)
Sm per kilo. (0° — t°)
= 0.0685 + 0.00005t
(Bellati and Lussana)

Tin—SnS Sm per kilo. (13° — 98°)
= 0.0837 (Regnault)
SnS^2 Sm per kilo. (12° — 95°)
= 0.1193 (Regnault)

Antimony—Sb^2S^3 Sm per kilo. (23° — 99°)
= 0.0840 (Regnault)

Mercury—HgS Sm per kilo. (14° — 98°)
= 0.0512 (Regnault)

Lead—PbS Sm per kilo. (16° — 98°)
= 0.0509 (Regnault)

Bismuth—Bi^2S^3 Sm per kilo. (11° — 99°)
= 0.0600 (Regnault)

Thermophysics of Compound Sulphides.

Bornite—Cu^3FeS^3 Sm per kilo. (10° — 100°)
= 0.1177 (Sella)

Chalcopyrite—$CuFeS^2$ Sm per kilo. (14° — 98°)
= 0.1310 (Kopp)

Tetrahedrite—$Cu^4Sb^2S^7$ Sm per kilo. (10° — 100°)
= 0.0987 (Sella)

Bournonite—$PbCuSbS^3$ Sm per kilo. (10° — 100°)
= 0.0730 (Sella)

Mispickel—FeAsS Sm per kilo. (10° — 100°)
= 0.1030 (Sella)

Cobaltite—CoAsS Sm per kilo. (15° — 99°)
= 0.0991 (Sella)

Proustite—Ag^3AsS^3 Sm per kilo. (10° — 100°)
= 0.0807 (Sella)

Pyrargyrite—Ag^3SbS^3 Sm per kilo. (10° — 100°)
= 0.0757 (Sella)

Thermophysics of Arsenides and Antimonides.

Löllingite—$FeAs^2$ Sm per kilo. (10° — 100°)
= 0.864 (Sella)

Smaltite—$CoAs^2$ Sm per kilo. (10° — 100°)
= 0.0830 (Sella)

Domeykite—Cu^3As Sm per kilo. (10° — 100°)
= 0.0949 (Sella)

Dyscrasite—Ag^2Sb Sm per kilo. (10° — 100°)
= 0.0558 (Sella)

Thermophysics of Silicates.

Magnesium (Olivin) Mg^2SiO^4
Sm (100° — 0°) = 0.2200 (Vogt)
Melting point = 1400° (Vogt)
L. H. fusion = 130 Cal. (Vogt)
Q (solid) (0° — 1400°) = 520 Cal. (Vogt)

Magnesium (Enstatite) $MgSiO^3$ (with a little Ca replacing Mg)
Sm (0° — 100°) = 0.206 (Vogt)
Sm (0° — 1200°) = 0.301 (Vogt)
Melting point = 1300 (Vogt)
Q (solid) (0° — 1300°) = 403 Cal. (Vogt)
L. H. fusion = 125 Cal. (Vogt)

Aluminium (Topaz) $Al^2Si(F)O^5$
Sm (12° — 100°) = 0.1997 (Joly)

Aluminium-Beryllium (Beryl) $BeAl^2Si^2O^8$
Sm (12° — 100°) = 0.2066 (Joly)

Potassium-Aluminium (Microcline) $K^2Al^2Si^6O^{16}$ — Triclinic
Sm (20° — 100°) = 0.197 (Bogarawlensky)
Melting point = 1170° (Vogt)
L. H. fusion = 83 Cal. (Vogt)

Potassium-Aluminium (Orthoclase) $K^2Al^2Si^6O^{16}$ — Monoclinic
Sm (20° — 100°) = 0.1877 (Oeberg)
Melting point = 1200° (Vogt)
L. H. fusion = 100 Cal. (Vogt)

Calcium-Aluminium (Anorthite) $CaAl^2Si^2O^8$
Sm (100° — 0°) = 0.189 (Vogt)
Sm (1200° — 0°) = 0.294 (Vogt)
Melting point = 1220° (Vogt)
Q (solid) (0° — 1220°) = 358 Cal. (Vogt)
L. H. fusion = 100 Cal. (Vogt)

Calcium (Wollastonite) $CaSiO^3$
Sm (0° — 100°) = 0.179 (Vogt)
Sm (0° — 1200°) = 0.288 (Vogt)
Melting point = 1250° (Vogt)
Q (solid) (0° — 1250°) = 360 Cal. (Vogt)
L. H. fusion = 100 Cal. (Vogt)

Calcium-Magnesium (Malacolite) $Ca^3MgSi^4O^{12}$
Sm (0° — 100°) = 0.186 (Vogt)
Sm (0° — 1200°) = 0.264 (Vogt)
Melting point = 1200° (Vogt)
Q (solid) (0° — 1200°) = 319 Cal. (Vogt)
L. H. fusion = 94 Cal. (Vogt)

Calcium-Magnesium (Diopside) $CaMgSi^2O^6$
Sm (0° — 100°) = 0.194 (Vogt)
Sm (0° — 1200°) = 0.281 (Vogt)
Melting point = 1225° (Vogt)
Q (solid) (0° — 1225°) = 344 Cal. (Vogt)
L. H. fusion = 100 Cal. (Vogt)

Iron (Fayalite) Fe^2SiO^4
Sm (0° — 100°) = 0.1700 (Vogt)
Q (solid) (0° — 1040°) = 310 Cal. (Vogt)
L. H. fusion = 85 Cal. (Vogt)

Iron-Aluminium (Garnet) $Fe^3Al^2Si^3O^{12}$
Sm (16° — 100°) = 0.1758 (Oeberg)
Melting point = 1145° (Joly)

Zirconium (Zircon) $ZrSiO^4$
Sm (15° — 100°) = 0.1456 (Regnault)
Melting point = 1760° (Joly)

(Pumice) Sm (10° — 100°) = 0.240 (Dunn)

(Lava from Ætna, 1669)
Sm (0° — 500°) = 0.182 + 0.000155t (Bartoli)
Sm (500° — 800°) = 0.270 (Bartoli)

(Crystalline Slag)
Sm (14° — 99°) = 0.1888 (Oeberg)

(Enamel Slag)
Sm (15° — 99°) = 0.1865 (Oeberg)

(Bessemer Slag)
Sm (14° — 99°) = 0.1691 (Oeberg)

(Basalt) Sm (20° — 470°) = 0.1990 (Roberts-Austen)
(Granite) Sm (20° — 524°) = 0.2290 (Bartoli)

For silicates in general, the specific heat is found to agree fairly well with what would be calculated from their percentage composition, giving each oxide constituent its proper specific heat. Those oxides whose specific heat at 0° are known are as follows:

SiO^2 = [illegible] (Pionchon)
Al^2O^3 = 0.2081 (Richards)
Fe^2O^3 = 0.1456 (Richards)
MgO = [illegible] (Regnault)
CaO = 0.1779 (calculated)

Those oxides not known give good results if assumed to be as follows:

FeO = 0.1490 (Vogt)
MnO = [illegible] (Vogt)
K^2O = 0.1390 (Vogt)
Na^2O = 0.2250 (Vogt)
Li^2O = 0.4430 (Vogt)

Using these data, S at 0° is calculated from the percentage composition of the silicate. For S at higher temperatures it can be assumed with considerable approximation to the truth, that S increases 0.078 per cent for each degree, and so, calling S_0 the specific heat at zero, we would have

$$S = S_0 (1 + 0.00078t)$$
$$Sm = S_0 (1 + 0.00039t)$$
$$Q (0° — t°) = Sm \times t$$

Besides the above generalizations, which enable one to calculate the heat in the solid silicate at the melting point (when the latter is known), Vogt has shown that if the heat necessary to heat the silicate from — 273° to the melting point is calculated, the latent heat of fusion may be taken as being 20 to 25 per cent of this quantity, say an average of 22.5 per cent, and thus the heat required for fusion may be approximately calculated.

Akerman has determined for many metallurgical silicate slags the total heat contained per kilogram of melted slag at the melting point. These quantities vary from 347 to 530 Calories, depending principally on the elevation of the melting point of the slag. A brief tabulation of Akerman's results are as follows, arranged according to the amount of heat in the just-melted slag:

Calories.	% SiO^2	% CaO	% Al^2O^3
347	59	36	5
	39	42	19
350	63	35	2
	58	35	7
	58	37	5
	53	37	10
	41	42	17
	38	47	15
	39	43	19
	37	40	23
360	66	32	2
	59	38	3
	48	42	10
	40	48	12
	34	48	18
	31	37	32
	46	37	17
	58	32	10
380	58	27	15
	62	37	1
	38	52	10
	25	34	41
	44	33	23
400	60	20	20
	65	35	0
	41	52	7
	37	53	10
	21	32	47
	43	30	27

The above includes most varieties of acid and basic iron blast-furnace slags. Data for other slags made in the metallurgy of iron and of other metals, are almost altogether lacking. A wide field is here open for metallurgical experiments: data thus obtained would be immediately useful in practical calculations.

[On account of absence of the writer from the United States the foregoing instalment is shorter than usual and problems are lacking. The brevity and omissions will be compensated for in following instalments.]

ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS.

Electric Furnaces.

Incandescent Electric Furnace.—F. J. Tone, 800,515, Sept. 26. Application filed Dec. 7, 1904.

In intermittently-operating resistance furnaces, as used for the production of carborundum, siloxicon and graphite and for heating carbon articles, the charge is piled up within walls of loose bricks. These walls serve merely to keep the charge in position, and being protected from the heat by the charge material they need not be highly refractory. After each furnace run the walls are removed, either wholly or in part, and up to the present the walls have been torn down by hand-labor, one brick at a time, and built up in the same manner. To save this expense, the inventor uses permanent end walls, which support the electrodes and make the side walls of one or more sections composed of reinforcing frames of iron, in which the walls are assembled. This renders the walls permanent in character and movable by means of cranes. The frames are constructed with downwardly-converging sides. This is important in a carborundum furnace, because it saves charge mixture.

Revolving Electric Furnace.—E. Stassano, 799,105, Sept. 12. Application filed April 9, 1902.

Claim 1 reads: "In an electric furnace, the combination with a circular closed chamber having its axis inclined at an angle to the vertical, of means for rotating such chamber, means for feeding an anode and a cathode toward one another within the chamber, and means for cooling the outer ends of the anode and cathode." Two vertical sections through the furnace are shown in Fig. 1.

This is the steel furnace used in the Government Gun Foundry in Turin, Italy, described in the Canadian Commission report (page 11) as follows: "The rotation of the furnace during the operation produces a proper intermixture of the melting mass, which, according to the inventor, results in accelerating reduction 'with great advantages from the point of view of the utilization of the heat and the preservation of the fireproof lining of the furnace.' * * * The furnace is of the arc type and consists of a cylindrical outer casing of iron, surmounted by a conical roof. The furnace is lined with magnesite brick. The axis of the furnace about which it rotates during the operation, includes an angle of 7° with the vertical. A three-phase alternating current of 90 volts between the phases and 400 amps. is distributed to the three electrodes, which nearly meet in the center of the interior of the furnace. Their distance apart is varied by a hydraulic regulator." Data on cost of furnace and operation are also given in the Canadian Commission report.

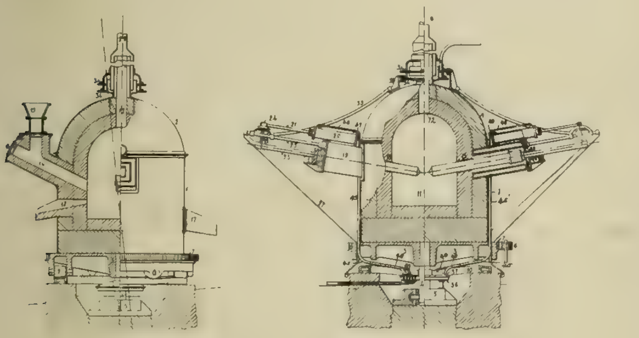

FIG. 1.—STASSANO STEEL FURNACE.

According to recent reports the Forni Termoelettrici Stassano Co. has been formed in Turin for the manufacture of steel in the Stassano furnace. Two furnaces, one of 1000 hp. and one of 200 hp., are to be installed.

Purifying Tantalum.—Werner von Bolton. (Assigned to Siemens & Halske, A. G.) 799,441, Sept. 12. Application filed Oct. 12, 1904.

Tantalum is freed from impurities, especially from tantalum oxide, by heating in an electric furnace in a vacuum. The oxide is sooner volatilized than the tantalum metal itself. The process is possibly aided by an electrolytic or thermic decomposition of the oxide into metal and oxygen. A vacuum is required, since at the very high temperature the tantalum metal is liable to enter into reaction with nearly all known elements. An arc furnace is used with electrodes of tantalum; or the arc is produced between the mass, to be purified, and one electrode of tantalum.

Producing Alkali and Alkaline-Earth Metal Hydrides.—F. J. Machalske, 800,380, Sept. 26. Application filed July 12.

A compound of an alkali or an alkaline-earth metal; for example, an oxide, carbonate, chloride, sulphate, sulphide, carbide or cyanamide, or a mixture of two or more of these compounds is heated to a high temperature in an electric furnace and is then subjected to the action of a saturated hydrocarbon, specifically methane or natural gas, thereby producing a hydride of the metal and by-products dependent on the compound employed. For instance, when calcium oxide and methane are the reacting compounds the products are calcium hydride, carbon monoxide and hydrogen. When chlorides of the metals are used, the chlorine combines with the carbon of the methane to produce chlorides of carbon.

Metallurgical Furnace.—G. H. Benjamin, 798,258, Aug. 29. Application filed April 25, 1903.

Details of construction of a tank-furnace, specially adapted to glass manufacture. ("It may, however, be used in smelting ores to obtain matte or pig metal and in other corresponding metallurgical operations.") Fig. 2 shows the furnace with the circular tank at the bottom. Extending from one side of the tank, and in an upward direction, is a flue in form of a series of communicating inclined and vertical passages. Each vertical passage is divided into two compartments, the outer one forming a by-path for the products of combustion, which flow to the chimney flue, while the inner compartment forms the connection between two succeeding inclined passages. This inner compartment contains the electrodes for the arcs (indicated in Fig. 2 in cross-section by small circles.) The glass-making materials drop down through the arcs, and are acted upon by the same. Sand is fed into the furnace through the hopper at the top and passes downward through the flue. Just before it reaches a vertical passage, fluxes, lime, etc., are fed through the hoppers, specially provided for this purpose, as shown in the illustration. The mixture then drops downward through the

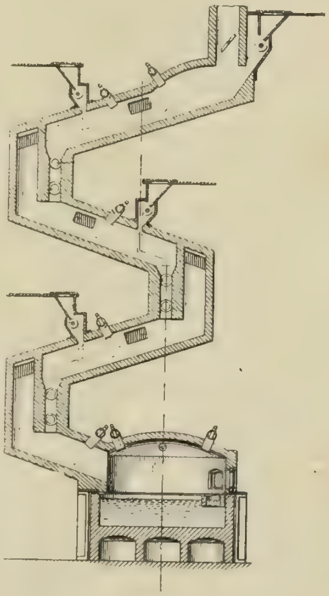

FIG. 2.—GLASS FURNACE.

etc. On account of the provision of the outer compartments at the vertical passages, which form a by path for the products of combustion, the light materials in the form of fluxes are not subjected at the time of their introduction into the furnace to the carrying action of flames or air currents.

Electrolysis.

Electrolytic Process.—Carl Hering, 798,790, Sept. 5. Application filed July 13, 1901.

Any electrolytic action disturbs the equilibrium which existed before in the electrolyte. In many cases the products of electrolytic action pass into or combine with the electrolyte and then participate in the conduction of the current; this may result in undesired reactions. To prevent such "objectionable" ingredients from accumulating in the electrolyte, the inventor catches the ions which would produce them at the electrodes in form of an insoluble compound. It may be that the ingredients would be objectionable in the electrolyte, but otherwise valuable. In such cases the electrode, which has now on its surface the insoluble compound, is removed from the electrolytic cell and placed into another cell, and the current is now passed through the electrode in the reverse direction; *i. e.*, if the electrode was before anode, it is now made cathode, and reversely. The result is, of course, that the insoluble compound is decomposed and the ingredient is obtained in the new cell in which it is no longer objectionable. The process will be rendered clear by two examples:

If zinc is extracted from an aqueous solution of zinc sulphate, with an inert anode, the anodic reaction is the evolution of oxygen and formation of sulphuric acid. Here the accumulation of sulphuric acid becomes objectionable, since it participates in the conduction of the current with resulting evolutions of hydrogen at the cathode and a correspondingly decreased efficiency of zinc deposition. Therefore, the SO_4 anion is not allowed to form sulphuric acid at the anode, but is caught in form of an insoluble sulphate. This may be done by means of a spongy lead plate (as used in storage batteries), where lead sulphate is formed. Under such conditions zinc may be deposited on the cathode without evolution of hydrogen, *i. e.*, with a high efficiency. If the lead plate with its coating of lead sulphate is then removed from the cell and used in another cell as cathode, it changes back into spongy lead, and gives off sulphuric acid.

Another example is the recovery of both zinc and acid from discharged primary-battery cells. First, the free acid is extracted by electrolyzing the solution between a spongy lead anode and lead peroxide cathode; both electrodes then receive a coating of lead sulphate; they are then removed and changed back in another cell to recover the acid. The solution from the first cell is then electrolyzed between a cathode of mercury, zinc amalgam, etc., and a spongy lead anode, as in the first example.

Ferric Oxide Electrodes for Chloride Electrolysis.—H. Specketer, 800,181, Sept. 26. (Assigned to Chem. Fabrik Griesheim Elektron.) Application filed Feb. 18, 1903.

A method of producing "in any large quantity good and useful magnetic oxide-of-iron electrodes of good electrical conductivity and great resistance against chemical action." The inventor uses "ordinary oxide of iron, such as is found, for example, in the decopperized pyritic waste (purple ore), in unlimited quantities and in the necessary purity at an exceedingly low price." The description of the method is very meager. The bottom of the electric furnace for smelting the mass is connected with the positive pole of the electric supply circuit. Above it is provided a movable vertical rod of carbon, connected to the negative pole of the supply circuit. The molten mass flows from the furnace through a channel in the wall into a properly shaped mould.

The properties claimed for this electrode are of considerable interest. They are "particularly suitable for the electrolysis of chlorides of alkalies, and present the following advantages over the hitherto-used anodes of carbon and platinum: First, they are considerably cheaper, their price being about a fifth of that of the carbon anodes, and they have an unlimited life in comparison with the latter; second, they furnish the chlorine free of carbonic acid, which is of great value for the manufacture of bleaching powder, for the volatilization of the chlorine, and for other uses of the chlorine, and, third, they require a considerably lower tension of the current than the electrodes of platinum."

Electrolytic Apparatus and Electrodes Therefor.—Carl Kellner, 799,061, Sept. 12. Application filed July 7, 1896.

In conducting an electrolytic process at high-current density with electrodes of an expansive material, like platinum, the cost sometimes becomes prohibitive with solid-plate electrodes. "A current from solid-plate electrodes will either be of insufficient density for the purpose or uncontrollable." Moreover, the large surface of a solid-plate electrode may have the disadvantage of unduly accelerating undesirable secondary reactions (for instance, in the electrolysis of sodium chloride for the production of bleaching liquor, those particles of hypochloride formed which come into contact with the cathode, are reduced again to chloride). The inventor uses for such purposes an electrode, consisting of a plate of non-conducting material, like glass or ebonite, around which is wound a helix of wire of platinum, etc. This may be used as a bipolar electrode, so that any number of such plates may be placed one after the other in a cell.

Electrolytic Reduction of Oxalic Acid.—E. von Portheim, 798,920, Sept. 5. Application filed Jan. 29, 1904.

A method of obtaining glyoxalic acid, its esters and anids, from compounds containing the radical oxayl (C_2O_2), consisting in dissolving the compound to be treated in dilute sulphuric acid in the cathode compartment of an electrolytic cell, and then electrolyzing the solution at a low or moderate temperature. For instance, in a diaphragm cell containing 2½ per cent sulphuric acid, oxalic acid is dissolved in the solution in the cathode compartment. Electrolysis is started, and by titration with a solution of phenylhydrazine the increment of glyoxalic acid is ascertained. When the maximum is reached, or just overpassed, the electrolysis is interrupted. The solution of glyoxalic acid is freed from the remaining small quantities of sulphuric acid by barium hydrate, and retains no impurities, with the exception of a slight trace of glycolic acid.

Electrolytic Production of Metallic Wire, Strip, etc.—S. O. Cowper-Coles, 799,634, Sept. 19. Application filed Feb. 6, 1905.

Around the periphery of a mandrel or cathode, which is preferably cylindrical, a fine groove or indentation of V-shape is described in the form of a helix or spiral, the pitch of which depends upon the width of the rod or tape to be produced. The groove must be very fine. The mandrel is rotated at a high speed. When the desired thickness of metal has been deposited upon the mandrel in the usual way, the latter is removed from the vat, when it is found that the metal sheet can be wound off from the mandrel with facility in the form of a spiral, the metal parting along the line of the groove or indentation."

Electrotyping.—C. F. Blackledge, 799,218, Sept. 12. Application filed Jan. 14, 1905.

The object is to quickly precipitate a compact and uniform surface of metallic silver upon a wax mould, so as to make it a conductor for subsequent deposition of nickel for electrotyping purposes. Twenty grains of nitrate of silver are added to 1 ounce of water, and liquid ammonia is added until the precipitate is redissolved. A reducing agent is then prepared, consisting of 40 per cent formaldehyde diluted with water in the proportion of 1 ounce of the former to 3 ounces of the latter. An alcoholic substratum of collodion is then applied to the wax mould with a brush or by flowing over it; it is made by dissolving 1 grain of gun-cotton in ¼ ounce of sulphuric ether and ¾ ounce of 95 per cent alcohol. Now, the above reducing agent is added to the silver solution in the proportion of 6 drams of the former to 2 ounces of the latter, and immediately afterwards the mixture is poured on the face of the mould, and the mould is manipulated so that the mixture is spread evenly over the face. In a very short time the mould gets a compact silver surface, on which nickel may be easily precipitated.

Electroplating Apparatus.—Louis Potthoff, 799,402, Sept. 12. Application filed April 30, 1904.

Details of apparatus for galvanizing bars, tubes and other pieces too large to be shaken up, as is done with small nails, etc. The inventor provides a traveling band with a series of cathode terminals adapted to move the articles to be galvanized; he also provides means which will automatically change the point of contact between the articles, the supporting means and the cathode terminals.

Galvanizing Wire.—G. L. Meaker, 799,860 and 799,861, Sept. 19. (Assigned to American Steel & Wire Co., of New Jersey.) Application filed Feb. 1, 1905.

A method and details of apparatus for coating a wire with zinc. The wires are laid in place from above, both the anode and the cathode contacts being slotted, or opened upwardly to receive the wire. The inventor also has devised a contact for the wire comprising a receptacle containing a series of loose balls, among which the wire may be readily placed, and, while making a good contact with the wire, the balls will yield to allow a joint in the wire to freely pass through without breaking the electrical connection. A cap or casing of insulating material is provided for each anode, so as to guide the wire and prevent it from short circuiting by coming into contact with the anode.

Electrode of Electrolytic Apparatus.—G. J. Atkins, 798,314, Aug. 29. Application filed Nov. 22, 1904.

Details of construction of a drum, almost completely closed, with the exception of a slit in the top; its inner surface is lined with carbon, which forms the anode; it is constructed in two symmetrical parts so that the drum may be opened and the cathode inserted or removed. The cathode is of cylinder form, concentric with the drum, and provision is made for revolving the cathode.

Batteries.

Storage Battery.—A. E. Knight, 798,906, Sept. 5. Application filed Nov. 16, 1904.

Details of construction for the purpose of preventing the buckling of the positive plates of storage batteries. The plate is provided with a plurality of slots, which extend from one side toward but not to the opposite side, so that the latter is composed of an unslotted portion from which extend a plurality of arms. These arms have coöperating with them a guide extended transversely of the arms and causing the arms to expand in a straight path.

Grid.—W. Gardiner, 800,128, Sept. 19. (Assigned to Universal Electric Storage Battery Co.) Application filed Aug. 22, 1904.

A sheet of pure lead is rolled into very thin sheets and closely perforated; it is then folded transversely into a plurality of folds. The outer edges are forced close together, leaving the central portion of the fold curved outwardly. The plate is then formed electrochemically.

Miscellaneous.

Electrolyzing Air.—A. Johnson, 798,511, Aug. 29. (Assigned to Barnard & Leas Mfg. Co.) Application filed Jan. 5, 1905.

The object is to convert air into a gaseous medium for bleaching flour by electric arc discharges. The apparatus consists of a number of tubular "generating chambers" in which the arc discharges take place between an upper cylindrical negative and a lower globular positive electrode. The position of the upper electrode can be adjusted so as to regulate the arc discharge. The amount of air passed through these chambers is controlled by valves. The outlet valve is automatically closed when the electrodes are in contact and is opened when the electrodes are separated.

Liquid Heater and Strainer.—W. J. Morrison, 799,798, Sept. 19. Application filed Nov. 7, 1904.

In obtaining certain liquids it is desirable that they be heated to a certain temperature as they are passing through the strainer. The inventor provides a form of strainer made up of a series of contiguously arranged sections of non-absorbent material, in which a suitably controlled electrically-heated conductor is embedded.

RECENT METALLURGICAL PATENTS.

On account of space limitations, due to the report of the American Electrochemical Society meeting, a large number of abstracts of metallurgical patents had to be reserved for our next issue.

Antimony.

J. S. MacArthur (796,849, Aug. 8) patents a process for the recovery of antimony compounds from ores containing them (especially stibnite Sb_2S_3) and the regeneration of the chemicals employed. A solution of 2 per cent of caustic soda is prepared in a causticizing vat provided with an agitator and a steam-jacket or coil, and is then delivered to a storage vat, also provided with a steam-jacket or coil, in which its temperature is maintained at about 100° C. The heated solution is then led to extraction vats provided with agitators, and it may be with steam-jackets or coils. The antimonial ore in a state of comparatively fine division—such, say, as to pass through a sieve of twenty meshes per linear inch—is introduced into the extraction vats, and the heated alkali solution is then allowed to act upon it, either at one or in successive operations, taking into solution practically its whole antimonial contents. The solution is then pumped from the extraction vats to a filter press, the effluent from which passes to precipitation vats. In the precipitation vats the antimony is precipitated from the solution and carbonate of soda formed in it by carbonic acid gas passed into the solution by distributing devices, the carbonic acid gas being preferably obtained from lime kiln or furnace gases. The solution is cooled while in the precipitation vats. The antimonial magma from the precipitation vats is discharged to a filter press, the effluent carbonate of soda solution from which is delivered to sumps, while the recovered antimony compound is removed from the filter press in the usual cake form. The carbonate of soda solution is delivered from the sumps back to the causticizing vat, and caustic soda is reformed by a suitable causticizer, preferably lime obtained from the kiln supplying the carbonic acid gas. The process thus proceeds in a cycle, the alkaline solution first acting on the ore, then being formed into a carbonate and then recausticized repeatedly.

Filter.

G. A. Duncan patents two filter constructions. The essential feature of one patent (796,198, Aug. 1) is a filtering bag of canvas, stretched upon and enclosing a peripheral frame made of perforated pipe. The pipe frame has connections extending fluid-tight through the web for exterior communication. The second patent (796,503, Aug. 8) relates to a filtering cell with its length and breadth respectively horizontal and vertical, while its thickness is small compared with the other dimensions. This filtering cell is suspended in a tank of similar proportions, there being relatively small space at opposite sides of the cell. Suction pipes intrude into the cell from the top and terminate open near the bottom. The slime-bearing liquor is supplied to the tank and aerated from the bottom.

Furnace Construction.

The ore roasting furnace of H. C. Holthoff (797,003, Aug. 15) has already been introduced into metallurgical practice by the Power & Mining Machinery Co., and was described in detail on page 87 of our February issue.

A patent of W. T. Rushton (797,915, Aug. 22) relates to mechanical details of a roasting furnace comprising one or more cylindrical muffles with internal spiral flanges, whereby the material under treatment is conveyed from end to end of the cylinder as the latter rotates. Cross projections, or inclined baffles, are arranged between adjacent flanges, and project laterally therefrom, so as to assist in moving the material.

J. A. Anker, J. H. Watson and P. Evans (794,837, July 18) patent a sinuous vertical roasting chamber; from the convexed portions of the walls project retaining shoes which catch a falling stream of ore, and retain the same in a mass and then delivers the mass gradually in a thin stream. These shoes are arranged in two sets, and automatically one set is brought to a horizontal position while simultaneously the shoes of the other set are inclined. Oil burners project their flame underneath the shoes, and independently of, but in proximity to burner, an air blast is introduced into the chamber.

A patent of C. E. Keating (797,584, Aug. 22) refers to details of construction of conveyor mechanism by which ore is moved along the hearth of a roasting furnace. A conveyor extends along the hearth from end to end and carries a succession of alternate sets of relatively fixed and rotating stirring devices. The fixed devices are adapted to enter the mass of material resting on the hearth and turn the same into longitudinal furrows, while the rotating devices have arms, entering the mass of material and rolling in a longitudinal direction, for preventing a crust from forming on the surface of the material.

F. Baldt (797,883 and 797,932, Aug. 22) patents details of construction by which open-hearth furnaces may be conveniently cleaned of any deposit which settles therein.

C. Ellis (795,259, July 18) points out that in present gas regenerative furnaces employing the usual type of reversing regenerator, the sensible heat of the producer gas is from the nature of the system employed wholly wasted. His invention consists of a furnace having an auxiliary or third regenerator, through which the air for combustion travels before entering the air regenerator proper, and which on the reversal is so constructed that the products of combustion from both the air and producer-gas regenerators combine and pass through this auxiliary checker-work, thereby giving up an additional portion of their sensible heat before departing to the stack passage. As this auxiliary regenerator has been cooled by the entering air to a low temperature much lower than that of the producer gas, it follows that the sensible heat of the producer gas, or a large portion thereof, is conserved and utilized.

J. Kirby (793,938, July 4) patents details of construction of a furnace in which the ore is to be smelted and refined by one and the same fire. It is a reverberatory furnace with an inclined smelting hearth. The molten mass then passes into a basin where it is refined.

A. Skoog (783,903, February 28) patents details of construction of an upright furnace for roasting quicksilver ores. By means of a plurality of fire and ore chambers arranged alternately and having them comparatively long and flat, with sinuous windings of the heat and vapors, the heat acts on the ore from all sides.

SYNOPSIS OF PERIODICAL LITERATURE.

A Summary of Articles Appearing in American and Foreign Periodicals.

Industrial Electrochemistry.

Aluminium Industry.—The *United States Geological Survey* has just published the statistics of the production of aluminium and bauxite in 1904. The production in this country has increased about tenfold in as many years. The following table shows the development in the United States since 1883, the output being given in pounds:

1883, 83 pounds; 1886, 3000; 1892, 259,885; 1896, 1,300,000; 1900, 7,150,000; 1904, 8,600,000.

The world's production in 1903 was as follows:

	Metric Tons.	
United States	3,402	$2,325,000
Switzerland	2,500	1,413,125
France	1,700	849,025
United Kingdom	650	328,250
Total	8,250	$4,915,400

A new development of some interest is the attempt to utilize the extensive deposits of bauxite at Leece, Italy, and the waterfalls of Pescara to generate electric power for the purpose of manufacturing aluminium. The prices per pound of aluminium and its alloys in 1901, 1902, 1903 and 1904 were as follows:

	Small lots. [illegible]	100 lb. Lots.	1000 lb. Lots.	2000 lb. Lots.
No. 1, aluminium, 99.75%	37	35	34	33
[illegible]	[illegible]	[illegible]	32	31
[illegible]	39	35	34	33
Special casting alloy (80% Al)	35	30	29	27

Aside from the electrical industry in which aluminium is gaining favor as a substitute for copper conductors for transmission lines, there has been expansion in other directions. "The steel industry has become an important consumer of aluminium. Usually from 2 to 5 ounces of aluminium are employed per ton of open-hearth steel made, and from 6 to 8 ounces for Bessemer steel." If every ton of steel manufactured in the United States in 1904 had been subjected to this treatment, there would have been something like 5,000,000 pounds of aluminium consumed. "The consumption of thermit shows marked expansion. * * * For welding iron and steel and for reducing refractory oxides thermit has no equal." Aluminium alloys represent a fertile field for inventors. The production of bauxite in the United States in 1904 amounted to 47,661 long tons, valued at $235,704.

Sterilization of Water by Ozone.—The August issue of the *Bulletin Mensuel* of the Société Belge d'Electriciens contains an illustrated paper by L. Gerard on the use of electric power in connection with the water-works of Breda, Netherlands. Alternating current is taken from the lighting network, which is operated by the three-wire system of 100 volts between each outer and the neutral wire. The water-works treat 20 cubic meters of water per hour. The water first undergoes rapid sand filtration and is then sterilized by ozone. For the operation of the ozonizers the voltage is raised from 100 to 65,000. The Schueller ozonizer is used, which is very briefly described. The average production of ozone is 7 grams per kw-hour. Considerable data are given as to the sterilizing effect of ozone upon water and the cost of operation. The cost of the production of the ozone and of its use for the sterilization of water, together with the cost of the previous filtration, is stated to be 1 cent per cubic meter; of this amount 0.234 cent is the cost of the ozone and 0.76 general expenses, interest on capital and amortization. The paper contains considerable statistical information.

Electric Steel Furnaces.—The *Oest. Zeit. f. Berg u. Hüttenwesen,* No. 31 to 36 (Aug. 5 to Sept. 9), contains a long illustrated serial, by Viktor Engelhardt, on the use of the electric furnace in the metallurgy of iron and steel. He first gives a general review of the subject and then discusses in detail the Kjellin furnace; the information appears to be the same as in his former paper (see page 294 of our August issue).

Theoretical and Experimental.

Tantalum and Columbium.—In view of the increased importance of tantalum, due to the new tantalum lamp, two papers from the John Harrison Laboratory of Chemistry of the University of Pennsylvania are quite interesting, giving new information on the chemistry of these elements. Both papers are printed in the *Proc. Am. Phil. Soc'y,* Vol. 44, 1905, on pages 151 and 177, respectively; Edgar F. Smith being the author of one paper, and the same and Roy D. Hall joint authors of the other. Without going into details it may be said that they succeeded in preparing large quantities of the double fluoride of tantalum and potassium, and had no difficulty in eliminating from it every trace of what was supposed to be the titanium double fluoride. Having thus at their disposal really pure tantalic oxide, it was determined to make a new study of the double fluorides of tantalum with the alkali metals and organic bases. One interesting result is that there are two cæsium tantalum fluorides, two rubidium tantalum fluorides, two sodium tantalum fluorides, two ammonium tantalum fluorides, etc.

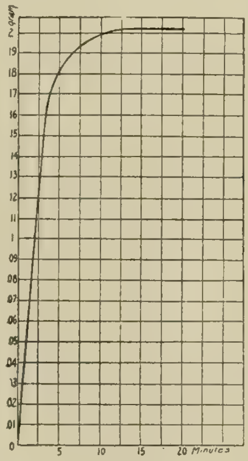

FIG. I.—ELECTROANALYSIS.

Electroanalysis.—A paper by Lily G. Kollock and Edgar F. Smith, in Vol. XLIV. (1905), p. 137, of the *Proc. Am. Phil. Soc'y,* recommends the combined cury cathode for electroanalytic use of a rotary anode and a merwork. Not only is the time reduced for the precipitation of the metals which were studied, but "the plan of combining a mercury cathode with the rotating anode gives an inexpensive form of apparatus which will eliminate the platinum dish, cone or cylinder from electroanalysis, and thus remove an expensive factor." Fig. 1 gives the results of the precipitation of zinc from zinc sulphate (containing 0.2025 gr. zinc with some free sulphuric acid); the current was 2 amps., the voltage 7, the speed of anode 500 r. p. m. The deposition was complete in 15 minutes. The absissas in Fig. 1 represent the time, the ordinates the weight of zinc deposited at the corresponding moment. Other metals studied were copper, nickel, cobalt, chromium and iron.

Physico-Chemical Research in Japan.—Vol. I., No. 2 of the *Memoirs* of the College of Science and Engineering of Kyoto Imperial University, contains three papers by Yukichi Osaka; the first relating to the distribution of iodine between two solvents, the second to the equilibrium of the electrolytic dissociation of partial neutralized acids and gases, and the third to the reaction between silver nitrate and disodium phosphate. The same journal contains an article by Sadajiro Okada, on the action of phosphorus pentachloride upon dextro-tartarin.

Electrolytic Analysis of Antimony.—This is the subject of a paper by H. D. Low and F. Mollwo Perkin, presented before the Faraday Society on July 3. Owing to the trouble of preparing the sodium sulphide solutions necessary for electrolysing antimony solutions by the method of Classen, the authors recommend the use of a solution containing ammonium tartrate, which they have found to give very satisfactory results. J. H. Belcher said that he had used the ammonium tartrate solution with success.

Hydrate Theory of Solutions.—A very interesting paper on an application to electrolytes of the hydrate theory of solutions was presented on May 18 before the *Faraday Society,* by T. Martin Lowry. The object of the paper is to consider the possibility of extending the hydrate theory to electrolytes in such a way as to take account of the observations which form the experimental basis of the theory of electrolytic dissociation. The hydrate theory postulates that an aqueous salt solution consists of a mixture of hydrates in equilibrium with the solvent and with one another. But it must be supposed that even in solution there is a limit to the possibility of hydrate formation, so that ultimately a stage will be reached at which the molecule as such will be unable to combine with any further quantity of water. The ionization of an aqueous electrolyte consists essentially in a further process of hydration, whereby the fully hydrated molecule combines with an additional quantity of water to form two or more hydrated ions. The hydration of the ions is thus conceived to be the primary cause of the ionization of aqueous electrolytes. It is believed that this extension of the hydrate theory to the phenomena of electrolysis may help to remove the fundamental difficulty of Arrhenius's theory, namely, the absence of a motive for electrolytic dissociation. The author discusses in detail various facts which are in favor of his hypothesis, and thinks the latter is quite as simple as the dissociation theory. For any given solution the chief constants are: (1) The total hydration H, of the solution, which expresses the total number of molecules of water present per molecule of salt. (2) The coefficient of combination b which expresses the fraction of the total quantity of water that is actually combined with the salt to form hydrates. (3) The product of these two quantities will give the average composition of the hydrates in solution; this may be termed the "molecular hydration" of the solution, and may be represented by the symbol h. Thus $h = b\,H$ may be regarded as the fundamental equation of the hydrate theory.

Dissociation.—Two papers on the dissociation of ternary electrolytes, by K. Drucker and G. Kuemmell, are published in *Zeit. f. Elektrochemie,* April 7 and June 2, respectively. The dissociation of potassium iodine is a subject of a discussion by J. W. McBain in *Zeit. f. Elektrochemie,* April 7. This paper is interesting, since it refers to the former method of procedure of theorists in the dissociation theory. Formerly one used to assume that the molecule dissociates into certain ions, and to test whether the conclusions are correct. The present author uses all the facts established experimentlly concerning transport numbers, electric conductivity and depression of the freezing point, to determine the concentrations of all the ions and complex ions which are theoretically possible. C. Liebenow, in *Zeit. f. Elektrochemie,* May 26, publishes a highly theoretical and mathematical paper, in which he develops a new formula for the dissociation of electrolytes, which he shows to be in agreement with Kohlrausch's measurements.

Equilibrium Conditions.—A paper by R. Luther and G. V. Sammet, in *Zeit. f. Elektrochemie,* May 19, deals with the conditions of equilibrium of the following two reactions:

$$HIO_3 + 5HI \rightleftarrows 3I_2 + 3H_2O$$
$$HBrO_3 + 5HBr \rightleftarrows 3Br_2 + 3H_2O$$

as determined chemically and by e.m.f's.

Tantalum.—F. Streintz in *Zeit. f. Elektrochemie* makes some remarks concerning Bolton's determinations of some properties of metallic tantalum. He compares the temperature coefficient as determined by Bolton with those of some other metals, and finds the rule confirmed, that with increasing one weight the temperature coefficient of the resistance increases.

Silicon.—Some rather peculiar notes on silicon are given by T. Gross in *Elektrochem. Zeit.*, June. The author considers that very poor electric conductivity of a substance indicates it to be a compound and not an element. To show that silicon is a compound he treated an alkaline solution of potassium silicate with high-tension alternating current, and found that he had at the end of the experiment less SiO_2 than at the beginning; he concludes some silicon was decomposed.

Alloys of Copper and Bismuth.—A paper on this subject was presented on April 4 before the *Faraday Society* by A. H. Hiorns, giving results of further research on copper alloys carried out in a similar manner to that on the copper-arsenic series (our Vol. II., p. 170). Arnold has investigated the effect of bismuth, from 0.1 to 0.5 per cent, on copper, and found that the investing membranes surrounding the grains of copper appeared to be split down the center, presenting a definite plane of cleavage. Gautier obtained a freezing-point curve similar to the author's, but his temperatures are generally higher.

The microscopic evidence mainly confirms the records of the freezing-point curves, of which there are four branches. We have pure copper and pure bismuth at the extremes, and solid solutions of copper in bismuth or bismuth in copper at intermediate points. The types of pattern on the etched surfaces depends on the position of the freezing points on the curve. The eutectic points occur at 243° C. and 1020° C., with a transition point at 858° C. From this point, with increase of bismuth in alloys containing 30 to 43 per cent of copper, the curve is practically a horizontal line, showing little differences in the freezing points. In this region two possible chemical compounds may be present, viz., Cu_3Bi_2 and Cu_3Bi, but there are no well-marked indications of such, either in the freezing-point curve or in the microsections. Such compounds can, therefore, only exist in contact with solutions containing excess of bismuth. The above-mentioned transition point is the transition point between solutions of copper in bismuth and bismuth in copper, and at this point the two solutions have probably the same freezing points, each being lowered by the addition of the other, thus forming a conglomerate of the two mixed crystals, which here exist in equilibrium.

Copper, with small percentages of bismuth, consists of polygonal grains, the boundaries of which contain a thin line of bismuthide. When such copper is broken the line of fracture follows this line. With 2 per cent bismuth the surface shows a characteristic eutectic structure. With increase of bismuth gradually widening boundaries to the grains appear and the interior is of a eutectic nature, which also increases with the addition of bismuth. With 57 per cent bismuth the whole surface is composed of grains of the two solid solutions, one of a light-gray and the other of a light-red color. With 60 to 67 per cent bismuth the alloys are composed of the two solid solutions mentioned above, with the addition of dark-blue grains, probably the compound Cu_3Bi. With 70 to 90 per cent of bismuth there are two main constituents, consisting of the red solid solution in a eutectic mixture. With 97.2 per cent of bismuth the whole is decidedly a eutectic mixture. The structure of alloys containing 98 per cent bismuth and upwards resembles pure bismuth.

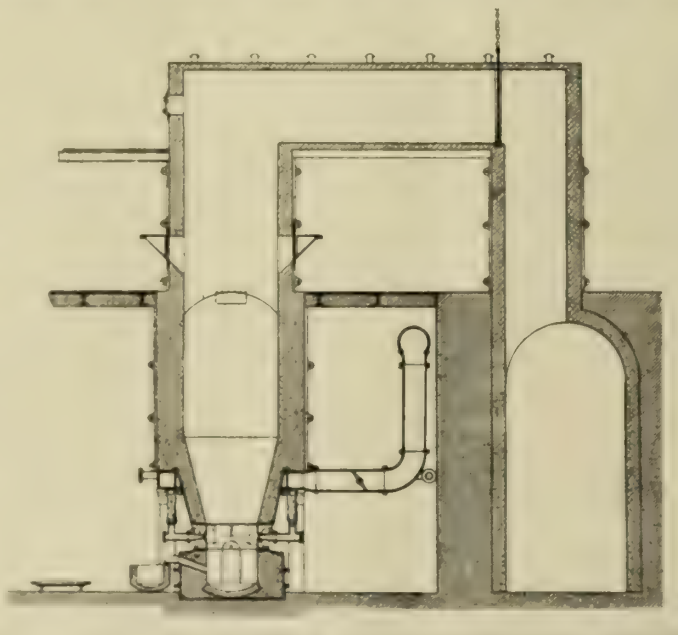

Fig 3

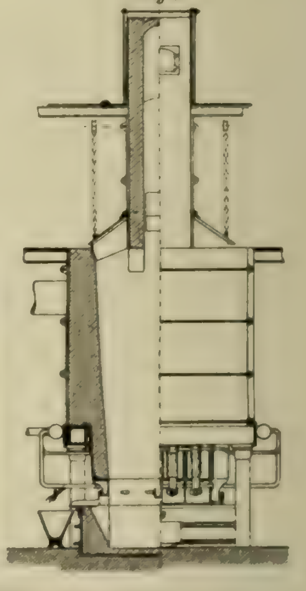

FIG. 2—LEAD BLAST FURNACE AT MONTEPONI.

Radioactivity as an Atomic Property.—It is now generally considered that the radioactivity of any element is independent of its form of chemical combination; that, in fact, radioactivity is a property of the atom rather than of the molecule. There is already much evidence in favor of this view. Thus, it is well established that the rate of decay of activity of temporarily radioactive products is wholly independent of the chemical as well as physical conditions. Further, all compounds of radium and uranium are active, as are also, probably, all those of thorium. The radioactivity of pure uranium compounds is roughly proportional to the percentage of uranium. The latter point has been studied in detail by Herbert N. McCoy, who gives the results of his investigation in the April issue of the *Jour. Am. Chem. Soc'y.* His experiments show that the effective (or directly observed) activity of the layers of uranium compounds, sufficiently thick to show the maximum alpha-ray activity, depends not only on the uranium content but also on the coefficient of absorption of the rays by the radioactive substance itself. When this absorption is taken into account the total alpha-ray activity of any uranium compound is strictly proportional to its percentage of uranium. This is a direct confirmation of the theory that radioactivity is an atomic property.

METALLURGY OF LEAD.

Lead Smelting Works at Monteponi, Sardinia.—The Monteponi smelter is the largest zinc-lead producer in Italy, and treats the complex local zinc lead ores. A brief illustrated description of the method of treatment adopted at the works is given in the *Oester. Zeitsch. für Berg. u. Hüttener*, Sept. 2, by E. Ferraris, the manager. The smelter receives: 1. Lead carbonate ores with some zinc oxide, which are sieved through a 10-millimeter screen, the part which remains on the sieve going directly to the blast furnace charge, while the part which passes through the sieve is roasted with lead sulphide ores or sintered by itself; 2, lead ores consisting of quartz with 10 to 15 per cent lead, which are smelted directly in addition to the lead carbonates, and 3, lead sulphides, which are ground fine and roasted. During the roasting there is added quartz sand, for the purpose of driving out the sulphuric acid and obtaining an easily fusible lead silicate. The roasting is carried out in two hand roasting furnaces, which are 18 meters long, and have a capacity of 12 tons of roasted product in 24 hours, with a consumption of 1800 kg, or 15 per cent of English bituminous coal, or 20 per cent of Sardinian brown coal. The smelting furnace charge consists, as a rule, of 50 per cent ores, raw and roasted, 20 per cent fluxes and 30 per cent slag from the works. The fluxes are limestone, with 98 per cent $CalO_3$ and limonite. The latter ore contains 48 per cent Fe, not over 4 per cent Zn, some lead and traces of copper and silver. One blast furnace smelts in 24 hours 60 tons of charge, or 30 tons of ores, with a coke consumption of 12 per cent of the charge and a wind pressure of 30 per cent to 50 millimeters mercury. The furnace shown in Fig. 1 consists of a firebrick structure, which in place of ordinary water jackets is provided with cooling plates in the smelting zone. These plates are 0.50 meters high, and can be cooled as desired by means of sprays of water. This arrangement is stated to entail a lesser consumption of water for cooling purposes than would be the case with cooling jackets, inasmuch as a part of the water evaporates and the danger of leakage is prevented. The crucible is provided with a cast-iron tube, 0.10 millimeters wide, for the purpose of conducting the lead to the outer kettle; this tube is slightly inclined in a downward direction, in order to prevent the escape of slag. The lead is tapped every 20 minutes. The condensing arrangement for the flue dust consists of a chamber, 3.40 meters wide and 1.60 meters long, which is divided by one longitudinal and five cross walls into twelve compartments. The gases change their direction seven times, and pass thereby six times over the longitudinal wall, in doing which they are met by fine sprays of water. The latter use 1.5 liters per minute each, of which four-fifths evaporate and one-fifth flows into a collecting reservoir. By this means 10 to 15 per cent of the entire flue dust are precipitated in the condensing chamber itself, and are removed from time to time in the shape of a thick liquid through openings provided for this purpose.

Metallurgy of Iron and Steel.

Bertrand-Thiel Process in the Manufacture of Steel.—In a paper presented before the Iron and Steel Institute in London, on May 11, by John H. Darby and George Hatton, attention is called to the many attempts which have been made to accelerate the open-hearth process of making acid or basic steel by increasing the size of furnaces, perfecting the machinery for charging the furnace, using molten iron from the blast furnace in conjunction with a mixer, etc. In all of these it has been found essential to use pig metal of special composition—not always easy to obtain, and often difficult to manufacture. The Bertrand-Thiel process is really a two-stage cleansing process. In the first furnace the bulk of the phosphorus, the whole of the silicon, and part of the manganese and carbon are removed; while in the second furnace scrap, ore and lime are added to the already fluid metal. The slag from the primary furnace of such a combination would often yield from 25 per cent to 28 per cent of phosphoric acid. As this process was worked by Bertrand, six to seven 20-ton charges of soft steel were secured from a pair of furnaces in twenty-four hours. At Brymbo, with a more highly phosphoric pig iron, this rate had been maintained. At the Hoesch works, in Dortmund, ten charges have been regularly produced per day. In the case of two works employing the Bertrand-Thiel process, the pig iron containing a high percentage of phosphorus, it was found advisable to employ a gas-heated mixer, in fact, a large tipping gas-heated furnace. In it the molten metal from the blast furnace is poured and retained for use as required, week-end metal melted, desulphurization takes place, and the percentage of silicon is reduced. If a temperature of about 1500° C. is maintained in the furnace, cold pig iron is readily melted, and the wear and tear of the furnace is slight.

Analyses of the metal finally yielded showed that the carbon largely remained, the silicon was almost all eliminated, the sulphur reduced, and 61.5 per cent of phosphorus removed. Counting the mixer as a primary furnace, three furnaces were engaged, the two secondary furnaces charged with 25 per cent scrap working up the output of the mixer with forty-two 20-ton charges between three furnaces, or fourteen charges per week per furnace. In conclusion, the authors state that by partial preliminary refining in one furnace, pig iron of almost any ordinary composition can be commercially treated.

Continuous Steel Process in Fixed Furnaces.—This is the subject of a paper read by S. Surzycki, on May 5, before the Iron and Steel Institute in London. At the Czenstochowa Works there is at work a 25-ton furnace of the following dimensions: The length between blocks, 26 feet 3 inches; width, 8 feet 7 inches, and depth of hearth from 1 foot 8 inches to 2 feet. During its first campaign it made 574 charges by the continuous process, whereupon, whilst still in quite good condition, it was stopped for certain local reasons. After repairing the ports and the roof and repacking the chambers, the furnace was again put into work, and since then has already made over 690 charges, and will, it is hoped, keep at work up to the 1000 charges.

The process is based on the idea and the principle of the Talbot process, with the essential difference that it can be carried out in any fixed furnace of not less than 25 tons capacity. This practice is mainly based on the arrangement of two or more tapping-holes placed over each other, but not arranged in a vertical line, such tapping-holes leading into a double launder by which the whole of the contents of the furnace, or only a part of the contents, may be easily emptied at any time. The tapping-holes are arranged in a cast-iron plate, lined up with magnesite bricks and stamped dolomite; they give no trouble, since they can neither slip nor sink, nor yet be corroded. The hearth of the furnace stands perfectly, and remains sound for months without any repairs. As a rule, only the slag line, and to a less extent the blocks, require repair.

The chief material required for the continuous process is molten pig iron, which, in the case under review, is taken from a 70-ton mixer. Unfortunately, the chemical composition of the pig iron varies very greatly, and varying percentages of iron ore have to be added to secure the conversion of the graphite into combined carbon and the oxidation of the silicon and manganese. The percentage of phosphorus is often rather high; with more than 1 per cent of phosphorus the slag must be drawn off as quickly as possible after the end of the refining period.

The furnace makes usually three charges in the twenty-four hours, each of 25 tons; when the blast furnace is working well, and a pig iron with low silicon and manganese is obtainable, four charges can be made. Unfortunately, the author is not able to give exact figures as to coal consumption, since the producer plant is common to the whole of the furnaces, and it will only be when the new continuous furnace, now building, which will have separate producers, is working that it will be possible to give the coal consumption exactly. Of course, it is considerably less than the ordinary process, as the furnace when continuously worked stores up an immense amount of heat, and much less fresh gas than in the ordinarily worked furnaces is required. It can, however, be stated that the coal consumption for the whole steel plant (which consists of five furnaces, four of which are ordinarily in operation) since the introduction of the continuous process, has fallen about 9 or 10 per cent (reckoned on the production of good ingots).

The final making ready of the charge at Czenstochowa takes place partly in the steel ladle. The test taken from the furnace before tapping, and before the addition of ferromanganese, must be capable of being hammered out without any fault, must possess the wished-for hardness, must be perfectly de-

dephosphorized must show a perfectly sound fracture, and must remain quiet in the small test ingot mould. This being so, the charge can then be tapped, and at the same moment a regulated amount of finely ground ferromanganese is thrown against the stream of steel as it falls into the ladle. A fairly active reaction is thereby produced, which, after one to one and a half minutes, quiets down completely, and when the metal has stood for a short time teeming can be commenced.

The author carried out a parallel series of experiments on ingots made by the continuous and ordinary processes. In both cases the differences in the composition were very nearly the same; in the continuous process the differences as regards the carbon and manganese are rather less, and as regards the phosphorus rather greater.

These experiments demonstrated that metal made (*a*) by the ordinary process and (*b*) by the continuous process are similar to each other, and made it clear that the former fears as to the metal deoxidized in the ladle being less homogeneous than that produced by the ordinary process were perfectly groundless.

The continuous process can naturally be still further developed, but in any case it has shown, during the last two years of practical work, its capabilities and advantages. These advantages do not consist solely in the continuity of the process itself, but in the longer life of the furnace, the higher production and yield, the lessened fuel consumption, and also in the simplicity of the whole plant and of the whole practice. There may be here and there some imperfections in connection with it, as is the case with everything new, but these will, it is to be hoped, be overcome with time.

Automatic Stock Line Recorder for Iron Blast Furnaces.—In a paper published in the May issue of the *Bi-monthly Bulletin* of the American Institute Mining Engineers, J. E. Johnson, Jr., describes in detail an automatic recorder of the status of the stock in a blast furnace. The device consists essentially of a test rod suspended at its top by a chain, but normally resting on the stock and descending with it, except when the bell is open; a simple mechanism is provided for lifting this rod out of the way on the incoming stock at the latter period. An attachment records its motion on a reduced scale.

Metallurgy of Gold.

Some Suggestions on the Cyanidation of the Tailings.—In the *Journal* of the Chemical, Metallurgical and Mining Society of South Africa, June, 1905, Prof. Prister outlines a method for cyaniding tailings which differs from that usually adopted. His discussion is based upon results which he obtained by leaching batches of 280 tons of sands with a preliminary wash of 0.057 per cent KCy, followed by a strong solution of 0.25 per cent KCy, which in turn was succeeded by a medium solution of 0.17 per cent KCy and a final wash of 0.05 per cent KCy. He ascertained that of the 388 tons of cyanide solution which were used for treatment, and which were made up of 20 tons of weak solution as a preliminary wash, 51 tons of strong solution remained in contact with the sands, 50 tons of strong solution for leaching, 153 tons of medium solution for leaching, and 134 tons of weak solution for leaching, about 208 tons contained more than 2 dwt. of gold in solution, while the remainder contained less. He therefore raises the question whether it is economical and necessary to pass all the solutions through the boxes, as is the usual practice, or whether it would not be better to pass only the first two-thirds of the solution through the boxes and run the last third directly to the sump. The first two-thirds of the solution leaching from the cyaniding vats are then rich in gold, and contain the highest percentage of KCy. Consequently, a good precipitation is assured, when the strong solution goes to its corresponding precipitation box and then to the strong solution sump, while the medium solution passes through its corresponding box and sump. The strong and medium solutions will, therefore, have passed through their boxes, and have lost all their gold and a part of their KCy, while the weak solution still contains all of its gold and KCy, as it has passed directly to the sump. For the next charge, therefore, the strong solution is made up with KCy to the required strength and used as above. For the medium wash, however, not the medium solution from the medium sump is taken, but in its place there is used the solution contained in the weak solution sump, which latter still contains its gold and all of its cyanide, as outlined above. This solution will dissolve more gold, and will then come from the leaching tanks partly as medium and partly as strong solution. For the weak wash of this second charge there is used the solution from the medium solution sump, which is pumped on the tank and drawn off into the weak solution pump, to be used as medium solution for the third charge. The subsequent charges will be treated in the same manner. The author describes a calorimetric method by which the control of the solution, as far as their classification into strong, medium and weak solution is concerned, may be effected with sufficient practical accuracy. It consists in adding a few grains of zinc dust or zinc fumes to 100 cc. of the solution, and strengthening it further with KCy by the addition of a few drops of strong KCy solution. The solution is then warmed gently for about 5 minutes and filtered, the excess of zinc dissolved with dilute sulphuric acid and filtered again, and the residue dissolved in about 10 cc. of aqua regia, made up of 3 parts conc. HCl, 2 parts conc. HNO_3, and 2 parts H_2O. A few drops of a solution of stannous chloride are then added, and the color obtained will indicate the richness of the solution. The advantages of the method thus proposed consist in the using up of a lesser quantity of zinc as well as of KCy in the boxes, the possibility of getting along with a lesser number of precipitation boxes, and therefore a smaller plant and the production of zinc slimes richer in gold, inasmuch as the solutions which are passed through the boxes are richer in gold. As the largest quantity of white precipitate is also formed in the weak boxes, and as in the outlined method weak solutions are not passed through the boxes, the latter will remain in a better working condition.

Cyaniding of Concentrates by Percolation.—A brief account of the method in which some 210 tons of accumulated concentrates, assaying 60 dwt. of gold to the ton, were treated by percolation at the Clutha gold mines at Barberton, Transvaal, is given by A. L. Edwards in the June, 1905, issue of the *Journal* of the Chemical, Metallurgical and Mining Society of South Africa. The concentrates were ground in a small ball mill and passed through a 1600-mesh standard screen, sampled, taken to the treatment vats, where they were first washed with water and 4 pounds of lime per ton of concentrates added. A first weak solution of 0.05 per cent KCy was then run on, and leached off, which was followed by a medium solution of 0.1 per cent of KCy. This was leached off and the concentrates were turned over. A strong solution of KCy (0.75 per cent) was then run on and allowed to remain in contact for 12 hours, after which the solution was drawn off and all the dissolved gold washed out as rapidly as possible. This treatment was repeated on three subsequent occasions, with the result that the extraction was brought up to 93 per cent. The time occupied in treatment was forty days, which could have been reduced by 25 per cent if time had been allowed for grinding the concentrates finer. The consumption of cyanide was rather high, owing to the concentrates not having been roasted. The leaching was good and the purification was excellent, the action being so strong as to necessitate the flow being reduced to 2 tons per hour. Two-thirds of the gold, amounting to 390 ounces, was precipitated in the first three weeks. In the strong box only clean zinc was used, whilst a zinc-lead couple was used with good effect in the weak box, the solution leaving the box with from 6 to 8 grains of gold. The strongest solution used was 0.75 per cent, and the weakest 0.08 per cent, with a final water wash. Towards the end of the treatment period the white precipitate made its appearance in the weak box.

Methods of Charging a Tavener Pan Furnace.—In a paper before the Chemical, Metallurgical and Mining Society of South Africa, Mr. H. Rusden gave a brief outline of the way in which the Ferreira pan furnace for smelting clean-up material from the cyanide works is charged. The fire is lighted about ten hours before the cyanide clean-up is ready, by which time the whole brick work is thoroughly hot. Any by-product lead that happens to be on hand is then melted in the furnace in order to form a bath. A little litharge may be reduced to serve the same purpose. The furnace damper is then closed tightly and a few shovelfulls of roughly broken slag is thrown on the lead bath. The wet clean-up mixed with the necessary amount of fluxes is then charged upon the bath. This clean-up in its turn is covered with crushed pan-slag from previous meltings, which pan slag is broken to about ⅜-inch tubes. The furnace damper is not opened until about twenty minutes later, and during this time the radiated heat from the furnace renders the slag cover pasty. This method of procedure prevents, to a great extent, any dusting. The fire is then urged on and the smelting is continued as rapidly as possible. When the first charge is smelted down level the second charge is put in under the same precaution as the first.

Tube Mills for Fine Grinding of Gold Ores.—Attention has been called frequently in the pages of Electrochemical and Metallurgical Industry to the importance which the tube mill, as an instrument for fine grinding of gold ores as an auxiliary to stamps, has already acquired up to this time, and that it is likely to play a still more important rôle in the future. Various attempts have been made to gain some idea as to the theoretical manner in which it performed its duty, yet the subject is so much in its infancy that there is still a large field open to research. Some further contribution to the theory of the tube mill has lately been made by H. A. White, in a paper read before the Chemical, Metallurgical and Mining Society of South Africa, and published in the May issue of the *Journal* of that society. In a general way the author concurs with the views of Herman Fischer, published in the *Zeit. Bes Vereines Deutscher Ingenieure*, namely, that the pulverising action is due almost entirely to the actual impact of the falling balls when the mill is crushed dry. However, when the mill is half full of water Mr. White thinks that that conclusion will require some modification. As a ball falling into 2 or 3 feet of water will not strike the bottom with enough force to do much crushing, he thinks that in this case it is probable that the grinding action between the balls will do a great proportion of the work. Thus the speed of the revolution would require to be so adjusted as to avoid an unnecessary amount of practically wasted fall, and would be best regulated so that the angle of repose of the crushing material is alone considered. Mr. White goes into a very extended mathematical calculation of the action of the balls in the tube mill, calculating a number of quite intricate formulæ. He also gives a table of diameters of the tube mills in inches and meters and the theoretical speed of revolution calculated in order to obtain the best results. As one of the salient features of the results of his experiments he mentions that as far as the running of the tube mill was concerned, a fairly efficient fall would be obtained if the tube were filled to the extent of two-thirds (when at rest) with balls. While relying on the fall for the crushing part it is, however, obvious that in all cases it cannot pay to waste energy in splashing water—the autlet must, therefore, be arranged so that the depth of the water does not much exceed the depth of the balls on the bottom of the tube mill, as shown in any diagram showing the mill in motion. In concluding his remarks about the subject Mr. White is of the opinion, however, that the question of the best diameter and length of the tube mill and the material and size of balls seems to be so essentially one to be determined by experiments on a working scale that he does not venture to offer any remarks upon it. In his opinion a practical method of determining the best speed at which any tube mill should be run when fitted up for working would be by indicator diagram or measurements of current, so as to determine when the power absorbed, divided by the number of revolutions per minute, became a maximum.

Milling in India.—The Champion Reef Mine is one of the most important of the gold mines in India on the Kolar gold field, and has yielded already an enormous amount of gold. A description of a method of treatment of the rock by R. F. J. Weeks is given in *Mines and Minerals*, Aug., 1905. The ore is a free milling one, and is treated in three mills of 120, 50 and 60 heads respectively. In the first of these mills the stamps are arranged sixty in a row, face to face, the ore bins behind each row. The stamps have a falling weight of 1250 pounds, comprising the stem, 16 feet x 3½ inches, 518 pounds ;the head, 2 feet x 9 inches, 368 pounds; the shoe, 12 inches x 9 inches, 237 pounds, and the tappet, 12 inches x 9 inches, 127 pounds. The stamps drop 8 inches at about ninety-four blows per minute. The mortar boxes are of the heavy Homestake pattern, 4 feet 8 inches high by 4 feet 9 inches long by 13 inches wide, inside measurements without liners. The order of drop of the stamps is 1, 3, 5, 4. The amalgamating tables are silver-plated copper, with an inclination of 1⅛ inches in 1 foot. They are of the step pattern, that is, instead of one continuous length of plate there are two sections, each 5 feet long with a drop of 2 inches between the top and bottom sections. The tables are not of uniform width, as the top section is 5 feet square, but the bottom section is tapered at the discharge end to 3 feet. No traps ar eused, but rifles are provided at the bottom of the tables for catching coarse amalgam. There is one table to each battery of five stamps. No inside plate is used, but chuck blocks, 6 inches in average width, are employed. Woven wire screening of 30-mesh is used. The mill is driven by electric power derived from a main station at a place some 92 miles from the field, known as the Canvery Falls, the Canvery being one of the principal rivers of Southern India. The mill is supplied with a three-phase alternating current of 25 periods per second, the supply voltage being 2200. The tailings flow through vats, where the separation of slimes and sand is effected preparatory to their being treated by the cyanide process. The average assay value of the ore crushed was 1 ounce 2 dwt., seven grains of fine gold per ton yielding an extraction of 18 dwts., one grain of fine gold per ton.

Bisulphate of Sodium in the Cyanide Clean-Up.—Owing to the fact of its having been observed that the rate of solution of zinc in bisulphate solution was greater than that in sulphuric acid of equivalent strength, J. E. Thomas and G. P. Williams carried out a series of laboratory experiments with a view of determining the following points: 1. Comparative rate of solution of zinc in equivalent solutions of sulphuric acid and acid sodium sulphate (bisulphate). 2. Maximum working strength of the bisulphate. 3. Maximum strength of the stock solution. 4. Effect of temperature, electrolytic couples, etc. The results of these experiments are described by them in a paper read before the Chemical, Metallurgical and Mining Society of South Africa, and published in the *Journal* of that society June, 1905. The paper also contains a detailed description of a series of trials on a large scale, which proved quite satisfactory. For a trial at the cyanide works of the Simmer and Jack mine, the solution of bisulphate was prepared as follows two days before the clean-up: A stirring gear was fitted in one of the lower acid vats, 10 feet in diameter and 4 feet 7 inches deep, and a wooden platform of 2-inch slats 2 inches apart was suspended from the top. The sodium bisulphate, previously broken to about 4-inch cubes, was placed on this platform, the vat was then filled with hot water until the platform was submerged and the stirring gear was started. Altogether 7556 pounds in all were dissolved, the total volume of water being 9 tons, so as to make a solution of approximately 11 gms. H_2SO_4 per 100 c. c. The whole amount of bisulphate dissolved in about five hours. The solution stood for about forty hours before use, and showed no sign of crystallization. In order to make as severe a test as possible the gold-carrying zinc from one of the weak cyanide boxes was taken without any cleaning whatsoever. On the morning of the clean-up, part of the solution was elevated to the zinc dissolving vats by means of a monteju, and the zinc from the weak box was fed into the solution. At first the action was not considered satisfactory, and hot water was added, and the strength of the solution brought down to about 9 gms. H_2SO_4 per 100 c. c.

This increased the action considerably, and the zinc dissolved briskly until the acid was used up, when all action ceased. The contents of the tubs were then transferred to the settling vats for washing, and the settlement obtained was good. More acid was then elevated to the tubs, diluted with hot water to about the same strength as before, and more zinc added until the action ceased. The liquid was then lowered again, and the process repeated until the acid solution had all been used up. The total moist weight of zinc from the weak box dissolved was 2201 pounds, *i. e.*, 1 pound of free acid in the bisulphate dissolved approximately 1 pound of moist zinc (unwashed). A noticeable feature during the solution of the zinc was the entire absence of that extremely unpleasant odor usually associated with a zinc plant clean-up, which the author contributed to the freedom of the gases from arsine, usually formed when zinc is dissolved in sulphuric acid which contains more or less arsenic in solution. The sulphuric acid used in the preparation of bisulphate is manufactured from pure silicon sulphur and free from arsenic. This point is considered by the authors of the paper to be of great importance in view of the extremely dangerous effects of arsine poisoning, as a man once overcome by gas at the zinc dissolving vats is always very liable to be overcome by fumes later on, and the effect is then much more severe. Another feature of importance was that although a most unusual quantity of zinc was fed into the tubs there was not the slightest tendency to boil over. The washing and filter pressing of the gold precipitates was carried out in the usual manner and the cakes calcined, placed, fluxed and smelted in clay liners in a reverberatory furnace. The solution when being run to the settling vats analyzed as follows: Free acid, nil; zinc sulphate, 13.32 per cent; sodium sulphate, 20.82 per cent; free acid originally present, 8,23 per cent. The acid plant at the Simmer and Jack works is now being reconstructed to allow of the use of sodium bisulphate throughout the clean-up, and it is expected that a saving of well over 55 per cent will be effected in the cost of the acid. With a properly designed plant the labor required would be no more than when using commercial sulphuric acid, as the soultion of the bisulphate could be performed by the usual labor on the works, and the coal required would be little more than is used at present.

MERCURY

Mercury-Ore Roasting Furnaces.—The large roasting furnaces for mecury ore at the mercury works of Idira, Austria, were provided with an open throat, which is covered to a depth of about 80 cm. with a layer of ore 3 to 20-millimeter size. This layer was not sufficient to entirely prevent the egress of the roasting furnace gases, which contained mercury, and therefore poisoned the workmen to a considerable extent. In order to prevent this occurrence, L. Buchal, the superintendent of the works, introduced an arrangement which he briefly describes in the *Oest. Zeit. f. Behg., p. Hüettenwesen*, July 15. In principle, it consists in drawing away the gases which enter the layer or ore mentioned above, for which purpose canals are formed in this layer by means of cast-iron roof-[illegible] [illegible]s. There are six of these canals in each shaft of the Czermak-Spirek roasting furnace, and twenty-four in the whole furnace, inasmuch as the latter has four shafts. The canals collect the gases and conduct them to longitudinal canals, in which there is maintained a comparatively high depression. A suction action is therefore also exerted in the gas-collecting canals in the cover of ore, and thus the gases are drawn away. This device, the author states, has proven entirely satisfactory. This action does not disturb the roasting furnace gases, which pass from the furnace through a pipe condenser, the chambers and flues. The canals for drawing away the gases in the ore cover do not lead to a pipe condenser, but the gases are carried directly into the main flue.

BOOK REVIEWS.

TRANSACTIONS OF THE AMERICAN ELECTROCHEMICAL SOCIETY. Vol. VII. Philadelphia: American Electrochemical Society.

This volume covers the proceedings at the seventh general meeting, held in Boston, Mass., from April 25 to 27. All the papers contained in the volume were abstracted in the report of the meeting in our May issue.

It seems that the statement in our report of a remark, made by Dr. Theo. W. Richards, in the duscussion of Dr. H. W. Patten's paper, has given rise to some misunderstanding. Our report stated that "Dr. Theo. W. Richards objected to Patten's statement that he has applied Helmholtz' formula while he used apparently Thomson's rule, thus using the total energy instead of the free energy. A complete treatment of the case would involve far more experimental work."

According to the Transactions, Dr. T. W. Richards said: "In this paper Dr. Patten speaks of having used Helmholtz' formula in calculating his results. I ask information. It seems to me he has rather used the equation of Berthelot or Thomson. That is to say, has confounded total energy and free energy. Now, if it is true that in this particular case the total energy is equal to the free energy charge, that is, that the cell has no temperature coefficient, it is an interesting case, and one which is well worth while studying further on this account. On the other hand, in view of the fact that Dr. Patten acknowledges decomposition at the electrodes, the conditions are uncertain, and the investigation, as presented to us, is an interesting beginning rather than a complete treatment of the subject. Being given as a complete treatment, a portion of it appears to be distinctly misleading."

In a reply, communicated for publication in the Transactions after the meeting, Dr. Patten states that it seems to him that our review gives a false impression both of Prof. Richards' attitude toward his paper and of the paper itself; that he has not yet determined the temperature coefficient; that in his paper he referred to the use of Helmholtz' formula, because he had been thinking in terms of that formula and omitted the temperature coefficient only until it should be determined experimentally; that he used the words "rough" and "tentative" in the two critical places, and that he did not consider the problem as finished. By the application of Helmholtz' formula Dr. Patten calculates the probable magnitude and sign of the temperature coefficient, and finds it would be + 0.000443 volt per 1° C., "a value rather high but not impossible." (But it remains to be proven experimentally that the total reaction in the cell is reversible—otherwise Helmholtz' formula cannot be applied.)

The frontispiece of the volume is a portrait of the president of the society, Dr. Wilder D. Bancroft, but scarcely does his normal smiling countenance full justice. The editing of the papers and discussions appears to have been carefully done.

Hot-Blast Smelting.

Hot-blast smelting is now engaging the serious attention of many intelligent metallurgists. In iron smelting the hot blast has made high blast pressure possible, without which furnaces of large cross-section could not be blown. There is no condition of blast temperature applying in iron smelting that does not apply with equal force and effect in lead smelting, for gold and silver, in copper matting and pyritic smelting—although, of course, each case has its own peculiarities.

In its recently issued very interesting book on "Some Details as to Smelting Practice and Equipment," the Colorado Iron Works Co. devote considerable space to the thorough discussion of the problems of hot-blast smelting. We believe the following abstracts from this book will be of interest to our readers:

In blast furnace smelting of gold, silver, lead and copper ores, conditions at the incandescent zone above the tuyeres for oxidation and reduction are controlled by volume, temperature and pressure of blast with reference to cross-section of the furnace at the tuyeres, character of the material being operated upon and end results accomplished. In the use of hot air provision must be in tuyere area to accommodate and allow the passage of its vastly expanded volume.

Heating air blast by reason of its vastly increased volume tends to facility of distribution and not to localization of combustion. Thereby uniform action is rendered possible by elimination of much of the cold otherwise carried in by the air blast and circulated or distributed irregularly and without control. In the exact proportion that a given quantity of air is increased in volume by heat, the practicability of its thorough distribution and contact with every atom of exposed material, and hence of uniformity in degree of combustion at the zone of fusion and in the desired conditions throughout the other auxiliary zones respectively, is realized.

There are large proportional areas, especially in the neighborhood of the tuyeres, in every cold-blast furnace, in which the desired reaction cannot take place because the blast of cold air keeps these areas too cool to admit of them. Proportionately as the air is sent hot into the furnace these cool areas are reduced in size and in far greater proportions are the capacity and efficiency of the furnace increased. Eliminating the element of cold from the air blast removes the one unmanageable factor in blast furnace smelting; it is unmanageable because the course or direction of the air when it enters the furnace is largely determined by the dark areas and patches of cold material, rendered so dark and cold by the cold of the air blast and against which it impinges.

In cold-blast furnaces there is a constant tendency to burn too high up above the tuyere zone, which is caused by the cold air cooling relatively all material at the tuyeres, while the air is being heated up to the temperature at which it is possible for it to become a factor in the reactions to which it is necessary.

Hot blast contributes to support the combustion of carbonaceous and other fuel, as sulphur, etc., with a minimum of oxygen applied, *i. e.*, the hot blast and fuel are drawn upon most for the necessary smelting reactions, and least possible for heating *per se*.

Of greatest importance is the proportionate quantity of hot air as compared with cold air involved in smelting. Let us assume a furnace, 42-inch x 168-inch cross-section at the tuyeres, into which 10,000 cubic feet of cold air is blown per minute, and 300 tons per 24-hour day of sulphide and other ores, fluxes and fuel are charged in, the percentage of good coke being 11 per cent of the charge.

The 10,000 cubic feet of air blown in each minute weigh 761 pounds at sea level, with normal temperature 62° F.; 300 tons charge per day is 417 pounds per minute; the coke, being 11 per cent of this, is 46 pounds per minute. To burn pure carbon two and two-thirds its weight in oxygen is required. Assume 10 per cent off, which must be accounted for as silica in the charge of coke, for ash and other waste, and there is left 41.4 pounds of carbon, which, multiplied by two and two-thirds, gives 110.4 pounds of oxygen, or 480 pounds of air, the extreme minimum possible. This, from 761 pounds of air blown in, leaves 281 pounds of air per minute absorbed in burning sulphur and iron in other reactions, and some passing through the furnace unchanged. An ore charge that has enough sulphur in it to run cold-blast on 11 per cent fuel charge, will always run on 3 per cent and less with 800° F. hot-blast.

By calculation as before, but now applied to hot-blast, 3 per cent of the charge being fuel would absorb 131 pounds of of air, which added to 281 pounds blown in the cold blast and not accounted for as combined with carbon, assumed again as excess in the case of hot-blast, makes a total of air required to drive the hot-blast furnace 412 pounds, or 5400 cubic feet of air, required in hot-blast furnace for a given duty, as against 10,000 cubic feet in the cold-blast furnace. Hence only 5400 cubic feet of air must be pre-heated in the stove for this purpose, and that with a cheaper fuel, as against 10,000 cubic feet to be heated at the tuyere zone of the cold-blast furnace with expensive coke. The stove thus proves an economy in heating the necessary air alone, and the fact is brought out that, by heating the air, the smelting capacity of any given size of furnace is very greatly increased—nearly doubled, in fact—under average conditions and proportions of material in the ore charge. The furnace that used 10,000 cubic feet of cold air per minute, blown in at the tuyeres, has capacity for a like 10,000 cubic feet of cold air heated up to, say, 900° in a stove, while, as shown above, the heated air in the latter case is equal to serve in smelting capacity to nearly double that of the like amount of cold air blown directly into the furnace.

The cold or deficiency temperature carried into the tuyere zone of a cold-blast furnace by the air necessary to smelt a ton of ore, is more than one and a half times the cold or deficiency temperature incident to the ton of ore charged in, and it takes more than one and a half the number of heat units to heat this air to the smelting temperature than it does to to heat the ton of ore to the melting temperature. So great a proportion of cold introduced at the tuyeres is certainly an active disturbing factor, and must be prolific of furnace irregularities.

In hot-blast smelting of sulphide ores, but one-half, or sometimes three-fifths, as much air is necessary as in cold-blast smelting of the same ores, and, therefore, but a proportional cost is involved in heating the air used; that is, but one-half or three-fifths as much in value of fuel is expended in bringing the air necessary to drive a hot-blast furnace up to the ultimate smelting temperature as is required for a cold-blast furnace.

The difference in the amount of air required is owing to the smaller quantity of carbonaceous fuel necessary to be burned in the hot-blast furnace. Every pound of carbon saved saves also 140 cubic feet of air, for it takes 140 cubic feet of air to burn a pound of carbon. It is true that oxygen is necessary to the burning of sulphur for the production of heat, but it is also true that a cold-blast furnace has to burn the sulphur, and uses as much air in burning that sulphur as it is burned there, as a hot-blast uses in bringing the sulphur to complete combustion, and utilizes it as an available fuel in the smelting operation. The sulphur in the charge has to be gotten rid of—burned up—except that retained as a constituent of the matter. It will not burn when exposed to a blast of cold air; therefore, in a cold-blast furnace a lot of coke has to be charged in and the air heated thus to burn the sulphur, for the sulphur must be burned to dispose of it, so excess of air over that needed for burning the coke, must be blown in and heated by the coke to burn the sulphur. Thus it is evident that it takes as much heated air to destroy and so get rid of the sulphur in a cold-blast furnace as it does to utilize it as an available fuel when smelting with hot-air blast.

In the cold-blast furnace the cold air is blown in at the tuyere zone, and a part of it is there absorbed in keeping up the coke fire, while the excess over that so used is heated at that point, and passes thus hot into the zones of the furnace above, and there contributes the necessary oxygen to burn the sulphur. For this reason most of the sulphur is burned high up in the cold-blast furnace instead of being burned at the smelting zone, where its calorific effect can be of any value in the smelting operation. Sulphur will not burn in a blast of cold air without the addition of carbon. Sulphur does burn without the addition of carbon in a heated air blast.

When sulphur is burned in the higher zones of the furnace its tendency is to carry the smelting zone bodily high up, and when this occurs the furnace soon freezes up. Burning much sulphur above, as must always be done in a cold-blast furnace, contributes to, and usually produces, hot top; and hence cold-blast furnaces running on a high sulphur charge nearly, or quite, always run with hot top, and the heat wasted in a hot top is usually enough to run the whole furnace if applied in the right place—the tuyere zone. With the air blast heated to 800° or 900° F. all sulphur is burned at and near the tuyeres, and thus the smelting is done by the aid, chiefly, of burning sulphur instead of coke, as is the case in the cold-blast furnace

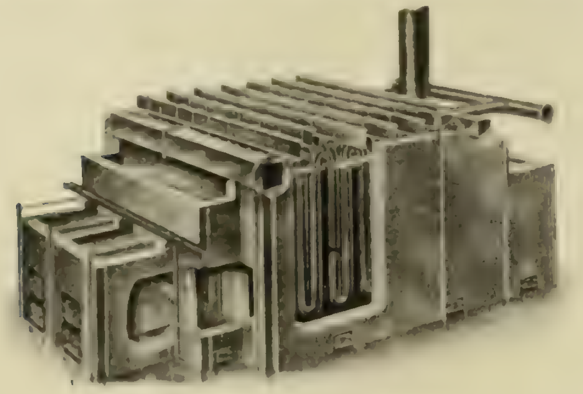

FIG. 1.—HOT-BLAST U-PIPE STOVE.

In iron smelting the air blast is heated by the waste gases, containing carbon monoxide. Little or no such gases escape from furnaces smelting the ores of copper, lead, silver and gold. In the latter case the Siemens regenerative or brick checker-work stove cannot, therefore, be used as it is in iron smelting. In its place, the U-type stove is very largely used. Fig. 1 shows the exterior and Fig. 2 a longitudinal vertical section of a hot-blast U-pipe stove, as made by the Colorado Iron Works Co.

The U-pipes are made of cast iron, and are so constructed that they will stand month after month at a low red heat without distortion or other damage. In order to provide ample heating surfaces the pipes are cast with longitudinal ribs on the inside.

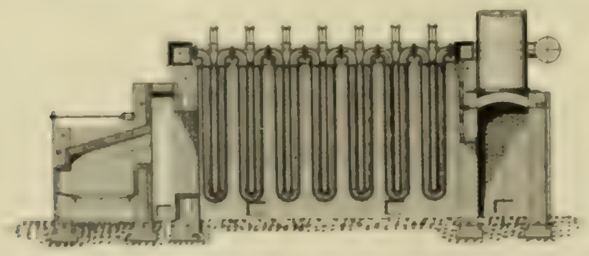

FIG. 2.—LONGITUDINAL VERTICAL SECTION OF HOT-BLAST U-PIPE STOVE.

The heating surface necessary for heating an air blast to 600° F. may be taken of as 0.4, and to 800° F. as 0.5 square foot for each cubic foot of air to be heated per minute. The extreme ultimate velocity of heated air on leaving the stove and in the pipes to the furnace should not exceed 5000 feet per minute. Air expands 0.002036 of its volume for each Fahrenheit degree added; therefore, when heated to 600° F. from 60° normal atmosphere, its volume has become 2.1 times its original volume, and hence all pipes and tuyeres must have more than double the area required for cold air of given amount in weight.

The Colorado Iron Works also have devised an oil stove hot-blast system for copper matting and pyritic furnaces. It comprises essentially a suitable chamber through which the air passes on its way from the blower to the furnace, and within this chamber in the atmosphere of the air blast an open fire of gas or oil is kept burning, the heat from which is absorbed by the passing air blast and carried thence into the furnace.

All the different advantages which are claimed for hot-blast smelting are summarized in the statement that when the air blast is heated, all matte smelting may be more satisfactorily done with less cost for fuel, with a cleaner slag, with higher tenor of matte, with less furnace irregularities, with increased furnace capacity and with less flue dust.

Metallurgy at the Lewis and Clark Exposition.

During the last session of Congress an appropriation was made to enable the United States Geological Survey to investigate the black sands of placer mines. Under the supervision of Dr. David T. Day, chief of the division of Mining and Mineral Resources, samples of black sands have been collected from placer mines of the United States, British Columbia, Central America and Mexico, and concentration experiments have been carried on all summer at the Lewis and Clark Centennial Exposition at Portland, Ore., in connection with the exhibits of mining machinery. A preliminary report on the progress of the investigation has recently been submitted by Dr. Day to the director of the Survey.

By the courtesy of the Lewis and Clark Exposition Co. a pavilion adjoining the Mines Building has been provided, 100 feet long by 50 feet wide, for the installation of concentrating machinery to carry on full-scale experiments as to the best methods of separating the useful minerals in various specimens of sand collected.

Prof. Robert H. Richards, of the Massachusetts Institute of Technology, is in charge of the concentration experiments. Invitations were sent out several months ago to all manufacturers of concentrating machinery in the United States, to send full-sized concentrating machines to this exposition for testing sands. As a result of this correspondence the following machines have been installed:

The Mine & Smelter Supply Co., of Denver, Col., has erected at its own expense a full-sized Wilfley concentrator, and has detailed Mr. A. W. Park to carry out any and all experiments in the concentration of sands that the Government shall direct.

The Joshua Hendy Machine Works, of San Francisco, Cal., have similarly installed the Pinder concentrator, which is under the personal direction of Capt. J. W. Pinder, the inventor. This company also operates a Hendy challenge ore feeder, which is in the charge of Messrs. J. Lee Anderson and Spencer Thompson.

The Woodbury concentrating table has been installed and is operated by the inventor, Mr. George E. Woodbury, of San Francisco, Cal.

Mr. C. Christonsen, of Oretown, Ore., has installed a Christonsen concentrator, and is operating it himself with Mr. Chas. D. Walcott, Jr., as assistant.

A new style of ore muller has been installed, and is operated by the inventor, Mr. I. J. Merrill.

A Wetherill magnetic separator, type E, of full size, has been loaned by the Wetherill Magnetic Separator Co., of New York. This machine is arranged to make separation of minerals at greater or less magnetic strength, ranging from a magnetic strength furnished by a current of 0.02 amp. to that furnished by a current of 3.5 amps. This is under the direction of Mr. Harmon V. Morse, of Johns Hopkins University.

Placed next to the magnetic separator is a set of the well-known Imperial ore screens, loaned by Mr. John Traylor, of Denver, Col., and manipulated under the supervision of Mr A. W. Park, assisted by Mr. Clifford L. Gardiner.

Two hydraulic classifiers, devised by Prof. R. H. Richards, are in process of erection. Prof. Richards has also installed an amalgamating table and a glass table with greased surface, such as is used in the South African diamond mines for separating special minerals.

Arrangements have been completed with the American Concentration Co., of Joplin, Mo., to procure a Knowles magnetic separator for quickly separating magnetic iron from black sands.

A rock crusher and an ore pulverizer are also installed.

With this equipment it is possible to treat as much as one carload of sand in 8 hours, although this amount has not yet been run in a single day.

Obviously the course of treatment to be used must vary greatly with the kind of sand to be treated. The sands are grouped in two classes: (1) sea sands, low-grade gravels and tailings from a number of dredges and their middlings from placer workings, and (2) heavy tailings from concentration, containing the residue from the cleanup from placer mines, etc.

The method of treatment so far developed for beach sands and tailings from placer mines has been to deliver the sand, after appropriate sampling, to the automatic feeder. It is then elevated to the roof of the building, passing over the Traylor screen, and is delivered to an automatic distributor, from which it is evenly fed by a current of water through four iron pipes to the several concentrating machines. These machines separate the sands into three portions—concentrates, middlings and tailings. The entire concentrates and middlings are collected, and samples of the tailings are taken out every 5 minutes. Samples of these concentrates, middlings and tailings are dried and then treated by the process devised by Mr. Henry E. Wood, of Denver, Col., by which the sands are first separated by the magnetic separator into six portions by five successively increasing magnetic fluxes, representing the chief minerals contained in the sands. These are finally separated by a hand batea. The end products thus separated and obtained are magnetite, chromite, garnet, olivine, monazite, zircon, quartz, gold and platinum. The amounts of these minerals are then weighed and the portions representing precious minerals are assayed.

An assay laboratory has also been installed and is now complete. This laboratory occupies a space of 20 x 20 feet in the Mines Building proper, courteously given for this purpose by the Lewis and Clark Exposition Co. It is under the immediate charge of Mr. Frederick W. Horton, of the Massachusetts Institute of Technology, chief assistant to Prof. Robert H. Richards, who directs the details of operating the concentrating plant. Assisting Mr. Horton are Mr. G. H. Shadinger, of Johns Hopkins University; Mr. Amos Loveland, Mr. Northrop Dawson, Mr. F. H. Hazard and Mr. L. G. Gillett, of Colorado Springs, Col.

Here the following apparatus has been installed, partly loaned and partly purchased of the F. W. Braun Co., of San Francisco, Cal., and Messrs. Eimer & Amend, of New York City: No. 40 Braun's combination crucible and cupelling furnace, with complete blowpipe apparatus and Cary hydrocarbon burner; the necessary crucibles, cupels, scorifiers, fluxes, etc.; and weighing devices supplied by Eimer & Amend, of New York City, including very fine pulp and button balances. This firm has also supplied a small electric furnace and electric hotplate. All the necessary chemicals and apparatus for making simple mineralogical determinations are found here.

In addition to being assayed, many specimens of concentrates from the various kinds of sands in process of treatment are run through a small glass classifier. This classifier, devised and made by Prof. Robert H. Richards, has twenty-four spigots and is capable of grading any material fed into it into as many different products, according to the settling power of each individual grain. The settling columns are of glass, allowing the operator to see just what is taking place in each column and enabling him to regulate the machine with great precision. Samples of sand which, on account of their small size, cannot be concentrated to advantage on the concentrating tables are put through this classifier and finished by the batea, and their values are thus successfully saved.

News and Notes.

Philadelphia Section American Electrochemical Society.—The first meeting of the new season is to be held on the evening of Oct. 6, at Soulas' Cafe, Fifth and Ludlow Streets.

Carbon Tetrachloride.—According to a recent consular report from Marseille, France, two manufacturers of that city have recently adopted tetrachloride of carbon for the extraction of oil, "with results satisfactory to themselves."

Hoisting Engines and Accessories.—Catalogue No. 126 of the Allis-Chalmers Co. on hoisting engines and accessories is an excellent, concisely written and profusely illustrated treatise on complete hoisting equipments made by this company. Everybody interested in hoisting problems will find this pamphlet a rich source of valuable information.

Gramme Monument.—A monument to the memory of Zenobe Gramme is to be dedicated on Oct. 7 in Liège.

Chemical Engineering Course at the University of Wisconsin.—Beginning with the fall of 1905, the College of Engineering of the University of Wisconsin offers a course in chemical engineering. It is the purpose of this course thoroughly to train students in the fundamental principles of both chemistry and engineering. For this purpose a five-year course is offered. The fifth year is not made obligatory, although, in some respects, it is the most important one, since in this year specialization and research work are provided for. Further details will be found in Bulletin 121 of the University of Wisconsin, Madison, Wis.

Electrolytic Lead Refining.—Messrs. Locke, Blackett & Co., of Newcastle-on-Tyne, England, are installing an electrolytic lead refinery under the patents of Mr. Anson G. Betts. The treatment of the lead bullion will be carried out about as described in our issue of Aug., 1903 (Vol. I., p. 407), while the slime will be treated by the new process of Mr. Betts, described in our issue of April, 1905 (Vol. III., p. 141). Dr. William Valentine is building the plant.

Artificial Graphite.—The International Acheson Graphite Co., of Niagara Falls, has doubled the capacity of its production by the large extension recently completed, and will use in future 2000 hp. instead of 1000 hp. as heretofore. According to the United States Geological Survey the value of Acheson graphite, made in the electric furnace, was $217,790 in 1904, while the value of all the graphite mined in the United States was $341,372.

The works of the Roberts Chemical Co., in Niagara Falls, were recently almost completely destroyed by fire. The company now considers rebuilding the plant. The products of the company are pure caustic potash and hydrochloric acid, the latter being obtained by combination of the hydrogen and chlorine gases evolved at the cathode and anode respectively in the electrolysis of potassium chloride.

Tungsten.—The Beaver Metallurgical Co. has just completed the installation of a large metallurgical and chemical plant at Beaver Falls, Pa., amply provided with the necessary furnaces and machinery for the manufacture of a long line of chemical and metallurgical products on a large scale, especially those of the rare metals. For the present the company will devote itself only to the production of metallic tungsten, ferro-tungsten and all other tungsten alloys and compounds in any quantity and in any degree of purity, with a view of later enlarging the scope of its activity as the conditions may warrant. Its favorable location, its close association with some of the largest tungsten ore producers, and its large and entirely modern equipment will place it in a position to meet any urgent demands upon its capacity. The company solicits correspondence with all manufacturers in the iron and steel branch, the main property of whose products consists in special hardness and resistance to abrasion, properties which can in no better way be secured than by the addition of tungsten to the raw material, no matter whether it be crucible steel, open hearth steel, cast iron or any other form of iron or steel used in the industries.

Life Saving in Mines.—The Daily Consular Reports of Sept. 25 contain an illustrated description of an apparatus of Profs. Bamberger and Boeck, for the purpose of eliminating the dangers accruing from poisonous gases generated by explosions, etc. There are two types of apparatus, one for continual use while at work and another for use in the moment of danger. Provision is made for both generating the oxygen needed for breathing and for absorbing the carbon dioxide. This is accomplished by means of sodium-potassium peroxide. The use of "oxone," now made by the Roessler & Hasslacher Chemical Co., suggests itself. This is fused sodium peroxide. Oxone, put into water, generates oxygen, while the sodium hydrate formed on the decomposition of sodium peroxide, absorbs carbon dioxide. Dr. von Foregger's Electrochemical Society paper, printed on another page of this issue, should be referred to in this connection.

Microscopes.—We have received from the Bausch & Lomb Optical Co., of Rochester, N. Y., their catalogue A of microscopes and accessory apparatus. The issue of this catalogue marks the completion of the fiftieth year of the business of this firm and of the establishment of the optical industry in America. During this half century the company has developed from a small beginning until it now employs over 1200 workmen in the production of optical instruments, which the ever-increasing demand for their products in all parts of the world requires. The catalogue is not only an extremely well illustrated price list of the instruments made by this company, but gives much information for users of optical instruments. We have also received from the same company their smaller catalogue on projection and photomicrographic apparatus, botanical and entomological equipment and supplies for elementary school laboratories, new apparatus for use in plant physiology, and microscopical stains.

Tanks.—The W. E. Caldwell Co., of Louisville, Ky., have sent us their illustrated catalogue of tanks, towers and tubs, made of steel or of wood or of wood and iron combined. This company has been building tanks and towers for twenty-seven years, and also makes tanks for special purposes, including acid tanks, mining tanks, plating tanks, etc.

Filter Papers.—We have received from Messrs. Carl Schleicher & Schuell, of Dueren, Germany, their catalogue with some samples of filter papers (in sheets, rolls, circles and folded filters). They manufacture filter papers in the most various forms and sizes and in different degrees of thickness and hardness, so as to meet any requirements of industrial or laboratory practice. These different brands are fully described in the catalogue. Messrs. Eimer & Amend, of New York City, are the sole agents for the United States.

"Gold Galvanizing."—A recent pamphlet of the United States Electrogalvanizing Co., Brooklyn, N. Y., deals with the advantage of electrogalvanizing over the hot process, and describes four special new devices which have recently been patented by Mr. Potthoff, the patents being owned by the above-named company. The first device serves for galvanizing long material from 6 inches up to bar-iron or pipes 20 feet and longer, the latter can be coated inside and outside, or outside only, according to requirements; the same device can also be used for sheet metal. A second apparatus serves for galvanizing small articles of any description, as nails, screws, small bolts, stampings, castings, etc. A third device is designed for wire of all kinds, and a fourth one for irregular shaped material which can only be handled by hand—the old method, but greatly improved upon.

Charging Apparatus.—A well written and illustrated booklet, recently issued by the Wellman-Seaver-Morgan Co., of Cleveland, deals with charging and manipulating apparatus for iron and steel works equipments. There are two sections, the first relating to charging apparatus for open-hearth furnaces, the second to charging and manipulating apparatus for reheating furnaces.

Morse Chain Co.—When the members of the American Electrochemical Society meet next spring in Ithaca, they will be able to see the new plant of the Morse Chain Co. which is now being erected there, and is to be operated in conjunction with the present factory of the company at Trumansburg, N. Y. The main building of the new plant will contain the general offices and will be three stories in height; the first floor will be used for the machine shop, and a floor space of 64 x 236 feet has been set apart for this purpose. Separate one-story buildings will be provided for the foundry, the forge shop, the pattern shop and the power house.

Mechanical Stokers.—That the mechanical stoker has reached such a state of perfection as to be considered indispensable in the equipment of modern boiler plants is indicated by the large number of orders booked by the Westinghouse Machine Company for the Roney stoker, a type of their exclusive manufacture. During the past ten years this company has developed the Roney stoker by successive improvements until it has become capable of meeting successfully all the requirements of heavy modern service. During the past month orders have been received for no less than fifty-one Roney mechanical stokers, ranging in size from 54 x 20-inch grate to 132 x 26-inch grate, the largest of the orders being that of the Pennsylvania Railroad for six 132 x 26-inch grate stokers, and five 100 x 20-inch grate stokers.

Factory Welfare Work.—We have received from Messrs. Crosfield & Sons, Ltd., The Erasmic Co., Ltd., Warrington, England, an interesting illustrated pamphlet which bears the modest title, "Notes on Some Improvements at Bank Quay Soap and Chemical Works, 1900 to 1905." It deals with the social improvement of the condition of the working people by means of recreation clubs, bowling, cricket, swimming, dancing, music, etc. clubs, fire brigade, life insurance, youth's education, etc. Other paragraphs deal with cleanliness in the works, absence of smoke, medical aid, compulsory medical examination, etc. "Every person who has been in the continuous employ of the firm for fifteen months on April 1, is allowed a week's holiday with full pay during the summer months, the choice of date being left as much as possible to the individual." "The firm has tried profit-sharing schemes in a few departments, and the success has been sufficient to induce it to extend the principle as far as possible. The schemes are adapted to the varying conditions in the different departments, but the main principle is that quality as well as quantity of output shall be considered in estimating the profits. In different departments increments from 14 to 25 per cent on the weekly wages are earned." Messrs. Joseph Crosfield & Sons have recently been favored with the Royal Warrant as Manufacturers by special appointment to his Majesty the King of England.

Alkali in China.—The Daily Consular Report of Sept. 26 contains some notes which should interest our manufacturers. The entire business in China is now, since 1901, monopolized by Brunner, Mond & Co., of Northwich, England. The products sold are 98 per cent carbonate of soda; caustic soda in three grades, 97 to 98, 90 to 92, 76 to 78 per cent; bicarbonate of soda, refined and mineral water; soda crystals; concentrated crystal soda; chloride of lime (sold at 5.50 Shanghai taels per 100-pound case; 1 tael = 52 to 58 cents); sulphate of ammonia. The business of the English company, starting from nothing, "has grown to large proportions. Their sales have reached 1500 tons of 2240 pounds a month of all products, and rarely fall below 500 tons of 2240 pounds a month, at an average price of approximately $50 a ton. In order successfully to compete with this firm it would be essential to build a warehouse on the water front at a cost of $20,000 in round numbers to carry the stock required for current business. * * * It would also be essential for a competing firm to possess a capital sufficient to withstand a cut in prices."

Ore Breakers.—The Allis-Chalmers Co. has shipped to San Francisco a consignment stock of Gates rock and ore breakers of various styles and sizes, together with standard repair parts which may be required for Gates breakers which that company has already installed in various plants in California and neighboring States. It will interest the Western customers of the Allis-Chalmers Co., who have in use any of their Gates rock and ore breakers, to know that by this arrangement they will be able to secure prompt delivery of repair parts for their machines. This venture will also enable the Allis-Chalmers Co. to make immediate shipment of complete crushers required in that part of the country, or in the islands of the Pacific. The company's San Francisco offices are located in the Rialto Building.

Dry Air for Blast Furnaces.—In view of Mr. James Gayley's revolutionary work introducing the dry blast into the metallurgy of iron, a recent folder of the De La Vergne Machine Co. is interesting. It illustrates the air-drying system of this company. The incoming undried air passes first through a fore-cooler, where it is partially cooled by water, and then through a regenerator to the first refrigerated chamber, where it is passed over coils of cold brine or direct expansion ammonia pipe, over which chloride of calcium solution is continually circulated, cooling it down to about 50° F. The cooled, moist air now passes through a series of screens, which separate out a large percentage of the moisture, after which it passess into the second refrigerating chamber, which has a temperature several degrees below that in the first chamber. Here the air is cooled to about 32° F., and then passes through a second spray catcher, and from there to the regenerator (in which the outgoing air is heated and the incoming air refrigerated). Then the air goes into the blowing tubs. Under certain conditions it may be advisable to leave out the regenerator, since the increased volume of the air heated in the regenerator decreases the efficiency of the blowing engines.

Modern Circuit-Breaker Practice.—Under this title the Cutter Electrical & Mfg. Co., of Philadelphia, has recently issued a profusely illustrated book describing, besides some of their established standards, a number of more recently developed types. The circuit breakers of this company are well known under the name of I. T. E., which stands for Inverse Time Element (the greater the need for the operation of the instrument the shorter the time within which it responds). Of the many types of circuit breakers which have been devised to meet almost all possible requirements, some of special interest for electrochemical and electrometallurgical works and for storage battery plants, are those which prevent the voltage dropping below a certain value and those which prevent a reversal of the current. To meet special requirements—such as are liable to arise in electrolytic refineries, etc.—circuit breakers of very large capacity are now available. Thus, the present book shows the illustration of 10,000-amp., 250-volt double-pole circuit breaker; it is 30 inches high, 24 inches wide and has a maximum depth when open of 20 inches from the switchboard to the most extended portions of the operating handle.

Ore Reduction Machinery.—Several new types of ore reduction machinery have been devised in recent years by Mr. William A. Merralls, of San Francisco, and are now placed on the market by the Merralls Machinery Co., 1123 Broadway, New York City, and the Merralls Engineering Co., 106 Shuter Street, Toronto, Can. Perhaps the most interesting of these new machines is the Merralls' improved stamp mill, which has the following novel features. Each stamp has its own separate mortar box, by reason of diaphragms cast in the solid mortar-box frame, and is, therefore, not burdened or interfered with in its work by the other stamps as in the open battery. Each stamp has its own ore feeder and its own water supply, and has screens on all four sides, so that the moment the material is crushed fine enough to pass the screens it goes onward, thus preventing sliming, making room for incoming ore and greatly increasing the production. The mill being practically free from the swash of the open mortar, which causes so much flowing and cutting of mercury, has great amalgamating capacity. Each stamp can be hung up for cleaning up or repairs while the balance of the battery go on crushing their full quota. The mortar permits of much heavier stamps being used, with a corresponding increase of capacity. For hard ore 1200-pound stamps are recommended. It is stated that on account of these improvements each stamp in a battery is enabled to handle from 6 to 8 tons per 24-hour day of hard quartz through a 40-mesh screen, wet, at an expenditure of about 2 hp. per 1200-pound stamp. With the four-side discharge, outside of each screen a battery plate is inserted, and it is stated that these battery plates secure from 60 to 90 per cent of the free gold right in the battery. Mr. Merralls has also constructed a new ore breaker and an improved Chili mill, and advocates the combined use of all three machines; the ore breaker for preparatory crushing, the stamp mill for further reduction to 8 or 101mesh, and the Chili mill for fine grinding.

Personal.

Mr. Walter H. Whiteside, recently elected president of the Allis-Chalmers Co., joined the organization in July, 1904, when he accepted the position of general manager of sales. He came at the time when the company, which had just taken over the Bullock Electric Manufacturing Co., needed the injection of a vigorous and energetic personality into its sales force. It was not merely that he had to become familiar with all the intricacies of the company's widely varied products, but the new interests and the old had to be consolidated, the sales organization enlarged and its efficiency increased. His promotion to the presidency indicates with what success Mr. Whiteside has met in his efforts. In taking up this higher and more responsible position, Mr. Whiteside has behind him not only the confidence of his organization but a long and varied business experience, in which he has filled many executive positions. His achievements have won him recognition as a man of marked administrative ability. Mr. Whiteside is a member of the American Institute of Electrical Engineers, of the Engineers' and Lawyers' Clubs, of New York, of the Mid-Day Club, Chicago, and of the Milwaukee Club.

WALTER H. WHITESIDE.

Dr. Paul Héroult, the distinguished French metallurgist, has returned to this country. On his present visit he is accompanied not only by Mrs. Héroult but by three of his children, since he intends to stay here for about a year to look after his interests in the development of electric furnace methods in the metallurgy of iron and steel in this country and Canada. Dr. Héroult recently expressed the conviction that not too far in the future the largest steel-making furnace will be an electric one.

Mr. W. S. Doran has just been appointed manager of the power department of Allis-Chalmers Co. In this capacity he will have complete charge of the company's commercial affairs pertaining to reciprocating steam engines, steam and hydraulic turbines, condensers, gas engines, blowing engines for iron and steel blast furnace service, and rolling mill engines, with headquarters at the general offices of Allis-Chalmers Co., Milwaukee, Wis. Mr. Doran was formerly associated with the British Westinghouse Electric & Mfg. Co., Ltd. As an evidence of the high esteem with which Mr. Doran is regarded by his business associates a banquet was tendered him on the eve of his departure from England, attended by many well-known railway officials, manufacturers, electrical engineers and representatives from practically all the large electrical manufacturing concerns in England.

The General Storage Battery Co., of New York City, have secured the services of Mr. J. B. Cowen, formerly prominently connected with the G. I. Co. for many years past, for the management of their sales department.

Mr. Eugene Holcomb has recently been appointed manager of the newly created foreign department of the Allis-Chalmers Co., to take charge of all the foreign agencies and foreign selling representatives of the company. Not only will he supervise the work of the well-established European and South African offices and all Eastern Hemisphere agencies, but he will also have direct control over the South and Central American business for the company. With the engineering possibilities and projects in South America Mr. Holcomb is thoroughly familiar, having spent a number of years in that country. The past three years he has had immediate charge of the Westinghouse companies' interests in Argentina. Mr. Holcomb's headquarters will be at the general offices of the company at Milwaukee, Wis.

Mr. W. S. Landis, of the department of metallurgy of Lehigh University, has sailed for Europe to spend a semester in Heidelberg, where he will study mineralogy with Prof. Goldschmidt. Mr. Landis' many friends will wish him bon voyage and a happy return. At the recent Bethlehem meeting of the American Electrochemical Society Mr. Landis, as secretary of the local committee, did very effective work in making the visitors feel at home and in doing everything in his power to make the meeting successful.

Digest of U. S. Patents.

Conducted by Byrnes & Townsend, Patent Lawyers, National Union Building, Washington, D. C.

Electric Smelting and Reduction Processes.

(Continued.)

No. 651,718, June 12, 1900, Henri Leleux, Paris, France.

Employs a metal, specifically iron, as an agent to reduce argentiferous galena and ores containing copper or nickel in combination with sulphur, arsenic or antimony. The iron withdraws the metalloids from the lead, silver, copper and nickel. Some arsenic, arsenic sulfide, iron arsenide, etc., are volatilized and pass off. The reduced metals collect beneath a floating layer of iron sulfide or arsenide, and are tapped into a mold. The reaction being exothermic, electric energy is only required to fuse the ore. The furnace shown comprises a vertical chamber of bauxite bricks, with feed-hoppers and gas outlets at its upper end. A massive iron electrode, to effect reduction, depends into the chamber. The hearth constitutes the other electrode and preferably consists of the metal or alloy to be produced, although carbon may be employed in recovering lead, silver or copper. In smelting nickel, the hearth preferably consists of a nickel slab, having a water-cooled terminal of cast steel.

No. 658,412, Sept. 25, 1900, Gustaf M. Westman, New York.

Ores of arsenic, as, for instance, mispickel, are reduced in an electric furnace designed particularly for the recovery of metallic arsenic and of such precious metals as may be present. The ore is charged onto a flat hearth, at opposite ends of which are depressions containing water-cooled electrodes of copper. On the passage of the current the ore is melted, and the metallic arsenic distilled into a condenser. Alternating currents are preferable, on account of the convenient control and freedom from electrolytic effects. When the ore contains precious metals a collecting layer of lead is maintained within the depressions above each of the electrodes, a single charge of lead being stated to serve for several charges of arsenical ore.

No. 659,926, Oct. 16, 1900, Charles B. Jacobs, East Orange, N. J.

An abrasive material is produced by heating bauxite or other aluminium hydrate to quiet fusion in an electric furnace, and permitting the melt to cool slowly. The walls of the furnace comprise a sheet-iron casing lined with firebrick or ground lime and interiorly faced with carbon bricks. The hearth is similarly constructed, and is arranged to be slowly lowered during the operation. A plurality of pairs of horizontal opposed electrodes enter the furnace above the hearth at its highest position, and the charge is fed upon these electrodes, the melted product collecting on the hearth. Preferably the bauxite is primarily calcined. The product is crystalline, somewhat harder than corundum, and possesses a much greater degree of toughness. The crystalline character, and hence the toughness of the product, is governed by the rate of cooling or by subjecting the mass to agitation during cooling.

No. 662,548, Nov. 27, 1900, Bernhard Scheid, Frankfort-on-the-Main, Germany.

Produces silicon containing 99 per cent Si and traces of carbon, iron and aluminium, but substantially free from carbide of silicon, by reducing quartz by carbon in the presence of acid, neutral or basic silicates of the alkali or alkali earth metals. The preferred charge consists of quartz and carbon in combining proportions, together with water glass of the composition $Na_2Si_3O_7$, the latter being added in the proportion of 5 to 30 per cent of the quartz. The current may be either alternating or direct. It is stated that the efficiency of the process is about 30 per cent. It is supposed that the melted silicate serves to instantly condense the vapors of silicon, thereby preventing the formation of silicon carbide.

No. 673,761, May 7, 1901, and No. 676,577, June 18, 1901, Alfred H. Cowles, Cleveland, Ohio.

Reduces sodium aluminate by coke or a hydrocarbon, recovering metallic sodium. The aluminate is mixed with crushed coke, or saturated with pitch, etc., the reducing agent being present in substantially equivalent proportions, and is reduced in an electric furnace provided with walls and hearth which are permeable to the sodium vapor. For this purpose they are constructed of compacted but porous carbon, or of hard carbon containing blocks or masses of porous carbon, the furnace being surrounded by a water-cooled spaced jacket, forming a condenser for the liquid sodium. By adding a volatile metal, as zinc, to the charge, the condensed product will be an alloy of sodium with this metal. Iron, copper or tin may be added to the charge, in which case the reduced product remaining within the furnace is the corresponding aluminium alloy. The furnace is arranged to be worked under considerable gas pressure, whereby the transfer of the sodium through the porous walls is aided. Describes also the reduction of mixed sodium and calcium compounds, in which case sodium is recovered in the condenser and calcium carbide produced in the furnace.

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Copper Smelting as a Process for Recovering Gold and Silver.

In the past ten years copper smelting has gained on lead smelting and the chlorination process, as a method the "rationale" of which is the recovery of the precious metals. Of course, there have been advance steps in these competing branches, but copper smelting has made the greater progress. Furnace treatment to a 50 per cent matte in the large shaft furnace and, what is having a wave of metallurgical popularity, the large reverberatory matting furnace are both very cheap concentrating processes and both entail very small losses, for copper slags often average as low as 0.36 per cent Cu. Blowing this matte to blister copper is a very cheap process, for copper Bessemerizing resembles the iron Bessemer process in putting an enormous tonnage through a small plant with a small number of laborers, and the copper process has the added advantage of the possible use of a silicious gold ore as a lining whose values are all "velvet" to the process. But the greater advantage of the copper complex treatment is found in the last stage—the electrolytic refining. Here the copper is turned out chemically pure and 98 to 99 per cent of actual values—not simply assay values, for these are usually exceeded—of gold and silver are recovered. Often nickel and cobalt are recovered and many other by-products are possible.

* * *

In competition with lead smelting as a means for collecting the precious metal, the copper process, we see, has superiority over the lead process which uses the zinc-desilverizing method—cheap and practical to be sure, but not as finished or neat as electrolytic copper refining. In the field of ore treatment, the copper furnace is cheaper in construction and uses less coke, especially if the furnace be operated so as to employ the available heat of the sulphur and iron of the pyrites. While the lead furnace must be very carefully operated, using a low temperature to prevent volatilization of lead and a very fusible slag which restricts ore purchases to certain classes of ores, the copper furnace has no such delicate apetite but takes in its voracious maw slags of a wide range of composition. Its losses are much lower for the reason that high temperature of the smelting zone keeps the slags liquid and allows perfect separation. The question of proper reduction of iron oxide and the elimination of the iron as iron matte is, of course, a problem in each furnace, but a much less difficult one with the copper furnace. The question of elimination of zinc (for all lead plants run their slags to 6 to 8 per cent ZnO, and use as much zinc ores carrying high lead gold and zinc values as possible) is less serious in the copper furnace with its short shaft than in the long shaft of the lead furnace. Here incrustations are naturally much more liable to occur. In fact, at Carson City, Col., a copper matting furnace on high zinc charge operates successfully with 18 inches between tuyeres and charging floor—a real dwarf of a furnace!

* * *

If reverberatory smelting is employed, the question of slags and degree of reduction do not enter the proposition at all, for by independent firing with soft coal any heat up to 1600° C. can be reached. The plant is not dependent on coke, but can use a poor quality of soft coal or wood in remote districts. In fact, the Argo plant of the Boston & Colorado Co., near Denver, has competed for years with the Colorado lead smelters for gold and silver ores. This plant has always used a refining process for gold and silver that is many dollars less favorable than electrolytic copper refining. Such are the inherent virtues of copper smelting when considered as a means for treating gold ores that it succeeds in poor conditions. With the great increase in consumption of copper, due to general expansion in the electrical business, the capacities of the electrolytic copper refineries near New York have gone up with leaps and bounds, and are about double what they were only six years ago. The American Smelting & Refining Co. and the De Lamar Co. are both enlarging their refineries to a capacity of 60,000 tons of copper each per year; the Raritan Copper Works within a year will be producing 24,000,000 pounds of copper monthly. The Balbach Smelting & Refining Co. is enlarging its plant, while the Nichols Chemical Co.'s monster plant is operating at almost full capacity, producing some 17,000,000 pounds per month. Considered in its entirety, the future of copper smelting, both for its main products and for the by-products, gold and silver, is most bright, and in all probability this rate of growth will increase. Its cause can be traced to metallurgical and commercial conditions for expansion.

◆

Partial Pressure, Partial Volume and the Lungwitz Zinc Process.

When certain quantities of different gases at the same pressure and the same temperature are brought together and allowed to intermingle, then—if there is no chemical reaction but simply diffusion—we know from experience that the volume of the gas mixture will be equal to the sum of the volumes of the constituent gases if the pressure and temperature are maintained at their original values. Diffusion will, of course, take some time, but finally a state of equilibrium will be reached in which the gas mixture is of homogeneous composition throughout the volume. *A priori* it would be possible to form two different and equally probable conceptions concerning this homogeneous composition. The first conception is to assume that each of the constituent gases maintains its individuality, that it simply disintegrates (mechanically) into a very large number of minute particles, each particle having the same volume and pressure as before. The gas mixture then is an agglomeration of minute particles of the different constituent gases side by side. In each particle there is the same pressure as before. The pressure is common to all constituent gases. But the total volume is not common to all; each infinitely small space is filled with one and only one gas; the total volume is the sum of the "partial volumes" of the different constituent gases, each "partial volume" being equal to the original volume which that gas had before diffusion. The second conception is that the different gases penetrate each other during diffusion, and that each gas finally fills the total volume. Since the volume of each gas has been increased, the pressure to be attributed to it must be smaller according to the Boyle-Mariotte law. Thus we have to attribute a "partial" pressure to each gas, and the total pressure of the gas mixture is equal to the sum of the partial pressures of the different gases (Dalton's law). Hence, according to the first conception, the pressure is common to all constituent gases, but the partial volumes are different, and the total volume is the sum of the partial volumes; according to the second conception the volume is common to all constituent gases, but the partial pressures are different and the total pressure is the sum of the partial pressure.

* * *

For many purposes of calculation the one conception will be just as good as the other, and whether to prefer the one or the other will depend on the conditions of the special case—just as in electrical engineering one will calculate with resistances if the different parts of the system are in series, or with conductances if they are in parallel. But with partial pressures and partial volumes the situation is somewhat different. The two conceptions are not perfectly equivalent. By putting the question before Nature in the right way we can get a decision as to which conception is the correct one. As Planck points out in his classical lectures on thermodynamics, a distinct answer to our question may be obtained from the consideration of a vapor differing widely from an ideal gas. Take, for instance, atmospheric air and water vapor at 0° C. under atmospheric pressure. Here the water vapor cannot be supposed to be subject to a pressure of one atmosphere, since at 0° C. no water vapor exists at this pressure. The only choice remaining is to assign to the air and water vapor a common volume—that of the mixture—and different pressures—partial pressures. Then the water vapor is under much less than atmospheric pressure, and this explains its existence in vapor form.

* * *

This question is of importance for the Lungwitz zinc process which has recently received, or is receiving, a trial on a large scale at the plant of the Warren Separating Co., Warren, N. H. The peculiarity of zinc metallurgy is, of course, due to the fact that the reduction temperature of zinc oxide is above the melting point of metallic zinc at atmospheric pressure. Dr. Lungwitz' process is based on the general consideration that sufficient pressure will prevent ebullition of any liquid, even if its temperature be raised considerably beyond its boiling point under ordinary pressure. He, therefore, proposes to treat zinc oxide in a blast furnace under pressure. Dr. R. C. Schuepphaus has made some experiments in this direction, and was successful in obtaining ingots of zinc, as described in a paper before the Society of Chemical Industry in 1899, but his experiments were made in a closed electric furnace in which a comparatively high pressure could be maintained without much difficulty. With a blast furnace the difficulties will be considerably greater. But, above all, it would seem that it has not always been kept clearly in mind that it is the partial pressure of the zinc vapor—not the total pressure in the furnace—which is the decisive element in getting liquid zinc. The case is exactly analogous to that of water vapor in air at 0° C. under atmospheric pressure, mentioned in the last paragraph. Let us assume, for the sake of argument, that if we have nothing but zinc, a pressure of, say, three atmospheres will be sufficient to get metallic zinc in liquid form at the reduction temperature of zinc oxide. Then it would be very wrong to assume that a pressure of three atmospheres, when applied to the blast furnace, would be sufficient. In the blast furnace we have a mixture of zinc and other vapors, and, as may be easily found by calculation, the total pressure will be about seven or nine times the partial pressure of the zinc. Hence to get liquid zinc it will, under these suppositions, be necessary to apply a total pressure of twenty-one to twenty-seven atmospheres. We do not know what results have been or are being obtained with the Lungwitz process on a large scale at Warren, but in view of the above consideration it would be very interesting if such information would be published.

◆

Silicon.

According to the estimate of Dr. F. W. Clarke, of the Geological Survey, silica forms 58.3 per cent (or silicon 27.4 per cent) of the contents of the solid crust of the earth. Only very recently has it become possible to produce silicon by a sufficiently cheap process in the electric furnace, this being another example of the general rule that the conquest of the fixed, very stable and difficulty-transposable compounds of the strong metals is a special domain of electrochemistry. Aluminium is another example; and as was the case with aluminium some ten years ago the chief problem is now to find uses for silicon. So far its main use has been in the metallurgy of iron and copper, and silicon has been applied there in the form of ferro-silicon or copper-silicon on account of the lower melting points. Both ferro-silicon and copper-silicon are also products of the electric furnace. Copper-silicon is produced on a large scale in the Cowles works in Lockport, N. Y., and is also imported and is very largely used to get sound copper castings in order to combine with silicon the oxygen of the cuprous oxide dissolved in the copper. Ferro-silicon is still very largely imported from Europe. The reason given by American manufacturers of other ferro-alloys, is that in the special case of ferro-silicon the cost of power is such a large item of the total cost that with the power prices charged in general in this country it is impossible to compete with manufacturers in mountain regions in France, where power may be had at extremely low prices. The main action of ferro-silicon is assumed to be the same as that of copper-silicon. When used in steel manufacture, the first effect of the silicon is to combine with any oxide of iron remaining in solution in the bath, and thus to prevent the formation of "red short" metal.

* * *

However, it has often been pointed out that this cannot be the only action of the silicon, and that another effect is to increase the solvent power of the steel for the gases—since by direct chemical combination silicon evidently cannot remove hydrogen or nitrogen. The retention or occlusion of gases by iron is well known, but little exact information is available as to the influence of occluded gases on the physical properties of steel, apart, perhaps, from the hardness of electrolytically refined iron, due to occluded hydrogen, and apart from the mechanical defects, such as blow-holes produced by the gases. There is much more research work necessary and it is necessary to approach the problem with as broad a viewpoint as possible. Chemical research cannot reveal everything. It is important to use all methods of physics and physical chemistry. In this respect Mr. Albro's American Electrochemical Society paper, the first part of which is published in this issue, should be welcomed as an important addition to our knowledge on the way silicon influences the microstructure of steel and copper. Some points in this paper are also extremely interesting with respect to the chemistry of silicon.

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A Tree Never Quite Grows to Heaven.

Of the men associated with the group of capitalists known as the Standard Oil People in the lane that re-echoes Trinity's chimes, none seems to the public to do his work better than Frank A. Vanderlip, vice-president of the National City Bank of New York. This accomplished gentleman gives out at different times to the public utterances of a judicial character, evincing a practical and theoretical knowledge of political economy that is representatively American. In fact, we would compare his broadness of view in commercial affairs to that of our great electrical engineers, like Steinmetz, who unite a knowledge of "fourth-dimension mathematics" with the ability to build actual machines. Mr. Vanderlip likewise has the gift for expressing his thoughts in clear-cut crystalline English as do our highest type of technical men. In his recent address at Washington before the American Bankers' Association, he sounds a warning against speculation on the street and repeats the sentence, "A tree never quite grows to Heaven" like the ever-recurrent Leitmotiv of a Wagner opera. This warning should be seriously considered in the metal business, for the great increase in metal production, which is so closely intertwined with our country's prosperity, cannot in the nature of things keep on advancing perpetually. The setback is sure to come, and then those concerns that have not progressed conservatively will go to the wall, because of industrial weakness. Another line of thought that appealed to us was his eulogy of the savings and economies effected by the broad management of our trusts. The large corporations are doing in most cases exactly right, trying to keep their business on the basis of large scale production, to be seen in the cost sheet—the supreme court of commerce. It is satisfactory to Americans to see the Teutonic spirit of philosophy applied in a Caledonian practical manner by a representative man of affairs to the actual problems of the day. It makes us think that with all the weaknesses and vices of our crude, virile youth, this country possesses in the American type, seen best in our athletic President, sane and safe qualities for enduring noble life.

William B. Rankine.

Niagara Falls has become the center of electrochemical industries of the world on the basis of its power development, and the Niagara power development on large scale became a fact largely through the enthusiasm, energy and foresighted policy of Mr. William B. Rankine, whose loss we now mourn.

Mr. Rankine died on Sept. 30, at Franconia, N. H., to which place he went for the benefit of his health Aug. 12. The cause of death was congestion of the lungs, superinduced by heart trouble.

Mr. Rankine was born at Owego, Jan. 4, 1858. He was interested in the organization of the Niagara Falls Power Co., and at the time of his death was vice-president of that company and of the Canadian Niagara Power Co., and was in a large number of allied concerns. Besides his wife and mother, he is survived by three brothers, Harold, Delancey and Richard.

Mr. Rankine's father was the Rev. Dr. James Rankine, an Episcopalian minister, at one time president of Hobart College. He entered Hobart in 1873, and left it to go to Union College two years later. While at college he became one of the corporators of the Alpha Delta Phi fraternity. He was graduated in 1877.

For the next four years he lived in Niagara Falls. He then came to New York City and entered the law firm of Vanderpoel, Green & Cummings. Three years later he struck out for himself.

It was while living at Niagara Falls that he first conceived the idea of utilizing the great natural forces in the Niagara River and Falls for the production of power for commercial purposes. While in New York City he met prominent financiers and interested them in his plans. His leading associate and confident was Edward D. Adams, to whom and himself, more than any other men, the modern utilization of Niagara on a grand scale is due. He succeeded in forming a company, called the Niagara Power Co., and became its secretary and treasurer. Later he was made a vice-president.

Mr. Rankine always acted as the chief executive of the company, and spent his time between the works and the main office in New York City. Eventually he went again to live in Niagara Falls.

For many years he was a member of the committee on admissions of the University Club. He also belonged to the Metropolitan Club and the Bar Association. Besides the Niagara Power Co., the Natural Food Co., one of the largest concerns in the city, paid him a salary as a member of its executive committee.

In Feb., 1904, he married Miss Annette Norton, of Detroit, Mich.

The funeral was held at the Episcopal Church, in Niagara Falls. Mr. Rankine had a fine personality and made many strong and warm friendships. His hospitality at Niagara Falls will be remembered by many readers of this tribute to his worth and memory. What he has done for Niagara Falls is history.

Notes on Electrochemistry and Metallurgy in Great Britain.

(From Our Special Correspondent.)

[illegible] Electrical Exhibition at Olympia.

From the electrochemical or electrometallurgical standpoint this exhibition is distinctly disappointing, particularly to any who may read the preface to the catalogue of the exhibition, which contains a concise summary of the history of electrochemistry from the pen of Mr. K. R. Swan, a son of Sir Joseph Swan. From the electrochemical point of view the [illegible] are serious. There are no electric furnaces, no tanks for electrodeposition, and no electrometallurgical products, if one excepts copper, which appears in the form of cables or dynamo windings. No aluminium, no electric steel, no electrically produced chloride of lime or caustic soda or clorates, no calcium carbide, no samples of galvanized articles or deposited copper. There are, however, exhibits of secondary batteries (other than alkaline) on a scale never before attempted, and several pyrometers. In regard to the latter, Messrs. Siemens Bros. & Co., Ltd., exhibited a platinum resistance pyrometer, with a differential galvanometer and direct reading scale. A thermoelectric recording pyrometer with a platinum and platinum-rhodium couple, with a range of 1600° C., was also shown. For the purpose of exhibiting the delicacy of these instruments, the ends of the pyrometer tubes were placed in a small Bunsen furnace. In connection with the recording thermoelectric pyrometer, Messrs. Siemens Bros. supply an automatic commutator which will connect in turn different thermo-couples in different parts of a large factory, each circuit being made for the space of 2 minutes, and an interval of 1 minute being allowed between the records from each couple. Mr. R. W. Paul also exhibited his "universal pyrometer" for measuring temperature of from 100° to 1400° C., in connection with various interchangeable thermocouples. A smaller pyrometer for steam temperatures was also shown, together with an alarm attachment to ring a bell when any given temperature is reached or exceeded.

Gold from Sea Water.

Among the papers read before the recent South African meeting of the British Association for the Advancement of Science—to give that body its official title—special reference must be made to Mr. G. T. Beilby's paper on the subject. For a time, at any rate, Mr. Beilby has given a quietus to schemes for recovering gold from sea water, which contains on the average one grain of gold to the ton, by pointing out that on existing gold fields, with highly trained chemists in charge of works, the cyanide liquor which contains 100 grains or more of gold per ton, and the slimes which contain only eighteen grains or so, when run through the zinc precipitating boxes, issue from those boxes still containing 1.5 grain per ton; and the extraction is not more complete because the chemist cannot make it so, or because a more exhaustive treatment than that which leaves two or three grains per ton behind would not pay. Thus we have the gold seeker intentionally throwing away from his plant as of no further value, although it has already paid its own costs of collection, preparation and passage over the zinc, a liquor which is appreciably richer in gold than the ocean it is proposed to treat. The sea contains only half the metal present in the final effluent of the gold works, and yet it would have to be impounded and handled in a manner that must prove more expensive in labor and material than the simple process of zinc-box precipitation.

The Autumn Meeting of the Iron and Steel Institute.

First Day's Proceedings.

Last year the Autumn meeting of the Iron and Steel Institute was held in the United States, but for the September meeting of this year Sheffield was selected as the place of assembly. A gathering at a center like Sheffield has one advantage over the early summer meetings in London, and that is that, in addition to the reading of papers, various works are open to visits by members of the Institute. As a result, the Sheffield meeting, which was held in the last week in September, was well attended, and Mr. Bennett Brough, the popular and indefatigable secretary, had good cause to be pleased with the success of the meetings.

The papers read on the first day began with Prof. Arnold's account of the "Department of Iron and Steel Metallurgy at the University of Sheffield," narrating the equipment which, in the practical laboratories, contains appliances for producing steel on a manufacturing scale by the three standard methods of manufacture—crucible, Bessemer and Siemens steels. The crucible plant consists of a 6-foot new form Siemens gas furnace, capable of making heats up to 2½ tons. The Bessemer plant consists of a 1-ton vessel, fitted so as to take a very shallow bottom-blown bath, a side-blown heat by the Roberts modification, or a surface-blown heat by the Tropenas method. The praise which Prof. Turner, of Birmingham; Prof. Wedding, of Charlottenburg, and Prof. Gowland, of London, bestowed upon the equipment is well merited. There seems to be one salient omission, and that is that there are no electric furnaces of any type available for the production of special steels, but that all the alloys produced for investigation are the yield of the crucible furnace.

Thermal Transformations of Carbon Steel.

The second paper, by Prof. Arnold and Mr. McWilliam, on "The Thermal Transformations of Carbon Steel," is of considerable length and rather difficult of concise abstraction. Suffice to say, that steel was considered from the three points of view of saturated, super-saturated and unsaturated steel, and the appearances of ferrite, pearlite, hardenite, cementite, graphite, sorbite, etc., at various temperatures chronicled. A concluding paragraph on micrographic nomenclature may be quoted in extenso, because of a plea for a clearing of the air in regard to this matter.

"The authors earnestly advise students of metallography to keep as closely as possible to the well-established terminology of mineralogists and petrologists. If specific names terminating in 'ite' are indiscriminately given to indefinite transition mixtures of true constituents, the utmost confusion is certain to ensue. It becomes more and more evident that metals are igneous rocks (usually crystalline), and in more senses than one the nomenclature of metallography should be founded on a rock, otherwise the subject will never become a science. The constituents of iron and carbon steels must always be the types with which to compare the micrographic variations caused by the influences of elements other than carbon."

Finally, the authors quote the following words of Prof. Grenville Cole on the subject:

"The study of the igneous rocks has become, unfortunately, so involved in the question of their nomenclature that it is impossible to give an outline of the characters employed in their discrimination without a statement of the sense in which each particular name is used. All these names represent groups of rocks graduating into one another, but certain types can be kept clearly before the mind. When a rock is on the border line between two groups, whether in structure or mineral constitution, we must be content to say so without attempting to disguise natural facts by our classification. Petrography has of late suffered from the introduction of an abundance of new terms, and, what is far worse, of old terms defined in new senses; but the majority of these can be avoided by the use of familiar adjectives or mineral prefixes, to the great lightening of the science."

As might have been anticipated the discussion hinged largely on this virtual question of standardizing scientific terms. For instance, Mr. J. E. Stead advocated the use of the term "austenite," also preferred the words euctectoid, hypo-euctectoid and hyper-euctectoid, to the expression saturated steel used in the paper, and also had a hankering after martensite. Prof. Bauerman appealed for a simplification of the nomenclature used in this work, and urged the abandonment of the insincere use of mineralogical names. Troostite, for example, is the name of a well-known mineral, and now is also applied to some structure about the character of which everybody differed. Prof. Turner, Birmingham, had searched for sorbite in iron, and had not found it. Mr. Lange sympathized with the simplification movement as a practical man as well as from the point of view of the investigator. Mr. McWilliam pointed out that the object of the paper was not to dogmatise, as had been suggested, but to clear matters. Prof. Arnold, in reply, supported his contentions, and said that the misuse of names was really a breach of scientific etiquette. The name sorbite had been adopted with Dr. Sorby's permission to a substance discovered by Prof. Howe, and nobody should appropriate that name for another purpose without permission of Prof. Howe or Dr. Sorby.

Overheated Steel.

The third paper, by Mr. Arthur Windsor Richards and Mr. J. E. Stead, dealt with "Overheated Steel," and contained a detailed examination of a wagon axle (which broke at a flaw after twenty years of use), and was submitted to heat treatment. Subsequently, certain hypothetical conclusions were recited on which the author suggests that an ideal condition would be obtained if free ferrite were absent in carburized steels which have to be subjected to severe vibratory stresses. Ordinary structural steels in this condition can be obtained by heating to a suitable temperature, quenching and reheating at a lower temperature. One of the samples described as sorbitic steel was prepared in that way. It was devoid of free ferrite, and it will be seen that it has double the resisting power of the normal forged bars.

In concluding the authors laid special emphasis upon the fact that they "do not maintain that steel initially bad, brittle and dangerous, owing to irregularity in the distribution of the elements, or from other causes which have not yet been explained, can be made good by any kind of heat treatment. What we believe has been proved conclusively is that good steels which have been heated to any point short of incipient disintegration, and made excessively brittle by such treatment, can be completely restored to perfectly sound and reliable material. Also that it is safest to heat to a temperature about 50° above the critical points to ensure the complete change of every portion of the steel, excepting in the case of the purest and most homogeneous steels, when the temperature of the upper critical points need not be greatly exceeded." A short discussion followed, which does not call for comment.

Vanadium.

The fourth and fifth papers, by Dr. Leon Guillet, respectively, entitled "The Use of Vanadium in Metallurgy," and "Steel Used for Motor Car Construction in France," were then read in abstract by Mr. Brough. The following are the main conclusions respecting the former paper: (1) On normal steels it produces a very distinct increase in the tensile strength and elastic limit, and has no influence, or an insignificant one only, on elongation and contraction and upon resistance to shock. It slightly increases the hardness. (2) On quenched steel, vanadium considerably increases the tensile strength and elastic limit; it acts in this way, with almost as great an effect as carbon, yet notwithstanding this it does not increase the brittleness. The influence of vanadium in metallurgy is thus, in my opinion, of considerable importance. It is undoubtedly the element which, together with carbon, acts with the greatest intensity in the way of improving alloys of iron—that is to say, in very small percentages. It is to be specially noted, however, that alloys of iron, carbon and vanadium are more sensitive to heat treatment and mechanical handling than ordinary steels, but this does not appear to be any longer the case in more complex alloys, particularly in nickel vanadium steels. It remains to consider the influence of the addition of vanadium upon the cost per cent of the vanadium contained. The cost of production of ferro-vanadium is such as readily to allow of its addition, and if, at the moment of writing, the price of ferro-vanadium is still high—about £1 per pound of vanadium contained—this must be attributed to the scanty demand, which is altogether inadequate, and, consequently, entails expenses of manufacture which are spread over but a very small output, thus considerably increasing the price. This state of affairs will disappear when the use of vanadium becomes more widespread. It may be concluded that the employment of vanadium in the manufacture of special steels is distinctly indicated, particularly in the manufacture of quarternary alloys such as iron-nickel-carbon vanadium.

Steel for Motor Cars

With regard to Mr. Brough's paper the following summary is all that calls for quotation:

(1) Steels with low percentages of carbon and nickel—pearlitic steels—which are used for parts which require cementing and quenching, *i. e.*, shafts, gears which engage directly, etc. (2) Steels with medium percentages of carbon and low percentages of nickel used, after quenching and reheating, for a large number of parts, shafts, gearing, pinions, etc. (3) Steels low in carbon and with high percentages of nickel, used for valves. (4) Chromium steels, with high carbon and low chromium percentages, used for bearings. (5) Silicon steels, used for springs and for gearing. (6) Nickel chromium steels, with low percentages of nickel and of chromium, employed for numerous parts requiring resistance to shock, and a certain degree of hardness. (7) A new steel known as N Y, the composition of which has not been published.

The subsequent simultaneous discussion of the two papers was mainly confined to manufacturing testimony as to the value of small percentages of vanadium.

The Second Day's Proceedings.

Owing to the restriction of space I can only give the titles of the papers read, and next month will review their scope and the discussion they occasioned. Their titles were as follows: "Segregation in Steel Ingots," by Mr. B. Talbot; "A Manipulator for Steel Bars," by Mr. D. Upton; "The Reversible and Irriversible Transformations of Nickel Steel," by Mons. L. Dumas, and "The Presence of Greenish-Colored Markings in the Fractured Surfaces of Test Pieces," by Capt. H. G. Howorth. The following papers were not verbally discussed in formal session of the Institute, but being taken as read will form the subject of communications in the proceedings: "Wear of Steel Rails on Bridges," by Mr. T. Andrews; "The Nature of Troosite," by Dr. Carl Benedicks; "The Influence of Nickel and Carbon on Iron," by Mr. G. B. Waterhouse; "The Occurrence of Copper, Nickel and Cobalt in American Pig Iron."

The whole meeting had an air of cosmopolitanism, M. Guillet and M. Dumas having contributed papers from Paris, while Dr. Benedicks writes from Sweden, and Mr. Waterhouse and Prof. Campbell come from the United States. The cosmopolitan spirit will be developed to a greater extent next year, when delegates from the American Institute of Mining Engineers and other kindred bodies are coming over for a joint London meeting.

Market Quotations During September.

In one or two instances chemical products have risen during the month. Sulphate of ammonia has risen from £12.11.3 to £12.17.6 per ton. Bleaching powder and the soda products are unchanged. Arsenic (best white powedred) has risen from £13 to £14.10 per ton. Copper sulphate is slightly lower at £21.15. The coal tar products are in several cases a fraction higher. Potassium bicarbonate and caustic potash are lower, fetching £17.5 and £19.5 per ton respectively. Shellac is steady at £9 per cwt.

Copper fell very smartly at the beginning of September to £68 per ton, but recovered slowly to £71.5 by the 29th. Tin has been irregular, closing at £146.18.9. Lead fetches £14.10 per ton. The upward movement in the iron trade is very marked. Hematite rose from 58s. 9d. per ton to 62s. 3d. Cleveland pig iron rose to 50s. per ton, and has since gone higher.

London, Oct. 7.

CORRESPONDENCE.

The Artificial Production of Diamonds.

To the Editor of Electrochemical and Metallurgical Industry:

Sir:—In an interesting lecture delivered at the last meeting of the British Association, Sir William Crookes described a new formation of the diamond by the explosion of cordite in closed vessels, where a temperature of over 5000° and a pressure of 50 tons per square inch—conditions necessary for the liquefaction[1] of carbon—were realized.

Although we have made no serious attempt to produce diamonds, a description of some experiments on the solution of carbon in thermit iron may be of interest.

These experiments were performed in clay crucibles which were plunged into water as soon as the reaction was over. The iron was dissolved and the insoluble residue treated in the ordinary way.

We obtained, finally, a small amount of a black powder not attacked by hydrofluoric acid, hot sulphuric and potassium nitrate, boiling nitric and potassium chlorate, etc. The particles were not more than a few tenths of a millimeter in diameter, had a density above three, and disappeared when heated to a high temperature. We could not be certain that they scratched corundum, as they were extremely brittle. Iron, not suddenly cooled, gave no particles which could be mistaken for diamonds. Perhaps thermit under pressure might be used to dissolve carbon; but it would be difficult to attain the calculated critical pressure of carbon.

We have also utilized the heat given out in the thermit reaction for the fusion of silica. Graphite tubes filled with powdered quartz were heated in thermit slag, or were specially protected and heated in the iron. The silica fused nicely, and gave rods with few air bubbles. The expense of manufacturing quartz tubes by this method would be heavy; but some combination of electrical and thermit heating might be profitable.

F. M. G. Johnson.
D. McIntosh.

McGill University, Montreal, Oct. 17, 1905.

Metallurgical Calculations.

To the Editor of Electrochemical and Metallurgical Industry:

Sir:—I want to express my appreciation of your series of articles by Prof. J. W. Richards on "Metallurgical Calculations."

Prof. Richards' experience with the practical side of this problem, combined with the theoretical side, due to his contact professionally with many large concerns, is well known. This combination of "theory and practice wedded" is the reason for the warm and hearty welcome these articles are receiving.

Almost every metallurgical engineer I have seen recently has spoken to me about them, and we all have agreed that each of us would purchase the volume, when it is published in book form.

It will certainly fill a known want, for we will have in it the data and methods of calculation in handy form, and will be no longer compelled to hunt through a half dozen reference books (sometimes unsuccessfully).

It is a pleasure to give your journal this tribute.

Woolsey McA. Johnson.

New York, Oct. 24, 1905.

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[1] Crookes says, "Carbon and arsenic are the only two elements that have a melting point above the boiling point, and, among compounds, carbonic acid and fluoride of silicium are the only bodies with similar properties." Acetylene should be added to the latter group. Since acetylene can be exploded by heat, by detonators or by a "sympathetic" explosion, it might be substituted for cordite, but the pressure produced would not be so great.

A Co-operative Analysis of a Copper Slag.

By Thorn Smith.

In the *Engineering and Mining Journal* of Feb. 21, 1903, the writer submitted a report on a "Coöperative Analysis of a Copper Slag." The present article is a continuation of the same work, using another sample of slag. In the former article an endeavor was made to make the object of the work plain, yet many critics have persisted in ascribing wholly different intentions. Let it again be stated that the writer is *not* endeavoring to standardize methods for the analysis of slags or any other material, that he is *not* advocating strictly accurate methods for the analysis of the grosser smelter products, and, finally, that he is *not* endeavoring to displace methods now in use. He further wishes it distinctly understood that he is not combating the work of the committee, appointed by the American Chemical Society, on "Uniformity in Technical Analysis." The sole motive in undertaking the work is to make better chemists and to show the need of better chemical work by practical demonstration. It makes no difference whether a chemist is employed in an experiment station or a copper works laboratory, he is at the some time called upon for an accurate analysis, and that the vast majority of chemists are unable to make a strictly accurate analysis is a subject too one-sided for discussion. Steel works, acid works, cement works and the like, can use standard and practically uniform methods to advantage, but many others, including copper works, cannot, except in a few instances. The methods in use at steel works are pretty well standardized, but there is undoubtedly room for further improvement. The cement chemist, working on a comparatively simple material, is unable to agree with his brother chemist, and the need of standard methods, strictly followed, is very urgent. A coöperative scheme, embracing cement work, published two reports, but has now been abandoned for several reasons, chiefly from the fact that the chemists, those who should have been most interested, were not sufficiently alive to the importance of the work.

The writer, after sending out two series of slag, is not discouraged and will continue his work so long as a generous employer and time will permit. If the need of better chemists, better chemicals, better methods, and last, but not least, better work, can be demonstrated, he will feel that his efforts have not been without avail.

The second series of samples was sent out in May, 1903, and the long delay in making this report is due, to a great extent, to the writer's not stating any definite time in which to make returns. Coöperators were requested to do the work at their leisure, consequently the work was put off by a majority until forgotten or interest had died out. Some fifty odd requests for samples were received, which were sent out at no small expense. Of these but fourteen responded with more or less complete analyses. As in the first series the requests came from all over the world. It was hoped that more instructors would respond, but it appears that they were not sufficiently interested, though why is difficult to understand. Many of the letters requesting samples contained useful hints, and every writer seemed to appreciate the need of coöperation. With the sample was sent a tentative scheme of analysis, which was prepared more as a basis of criticism than as a guide to accurate work. The writer was greatly disappointed at the lack of criticism; in fact, practically none was received, although in several instances inaccuracies in the methods proposed were very evident.

To the following chemists the writer is indebted for more or less complete analysis:

O. W. Knight, Bangor, Me.

Percy Morgan, School of Mines, Waihai, Auckland, N. Z.

W. H. Kauffman, American Smelting & Refining Co., Durango, Col.

J. Watson Bain, School of Practical Science, Toronto, Can.

F. X. Mooney, Tennessee Copper Co., Copper Hill, Tenn.

Otto Huschka, Monterey, Mexico.

Nelson T. Hasenpflug, Grasselli, Chemical Co., Cleveland, Ohio.

Chas. E. Rueger, Butte, Mont.

S. H. Worrell, University of Texas Mineral Survey, Austin, Tex.

Thos. B. Swift, Mountain Copper Co., Elizabeth, N. J.

Miss Florence Jackson, Wellesley College, Wellesley, Mass.

H. M. Backus, Orford Copper Co., Bayonne, N. J.

C. A. Bohn, San Luis Potosi, Mexico.

Gino Bagnoli, Orbetello, Italy.

In the following tabulated statement the numbers are attached for convenience of discussion. They have no relation to the order in which the above names are given, beyond the three exceptions to be noted:

Coöperation Analyses.

Number	SiO_2	Fe	Al_2O_3	CaO	MgO	Mn	Zn	S	Cu
1..........	25.50	40.86	3.17	8.73	2.13	.37	2.94	2.01	.81
2..........	25.88	41.05	1.14	9.24	2.13	.65	2.39	1.64	.50
3..........	25.56	....	....	8.45	1.78	.19	2.91	2.18	1.08
4..........	25.58	41.58	3.33	3.96	Trace	.43	4.91	1.07	.64
5..........	25.16	40.62	....	8.61	....	....	....	....	.82
6..........	26.92	42.05	4.00	8.48	1.23	.66	....	2.01	.77
7..........	28.30	40.50	....	8.20	....	....	1.80	2.10	...
8..........	25.46	40.67	3.14	9.71	2.58	.28	3.99	1.97	.79
9..........	24.69	40.70	....	8.21	....	....	3.47	2.15	.76
10..........	26.20	40.95	3.95	9.20	....	.50	3.10	2.13	.76
11..........	25.68	....	....	....	....	....	....	3.89	.87
12..........	25.53	40.58	3.23	8.84	2.03	.37	2.42	2.17	.81
13..........	30.00	38.40	....	....	....	....	2.90	2.25	....
14..........	25.58	40.91	3.51	8.79	2.16	.37	3.31	2.14	.81
15..........	25.74	....	....	....	2.14	....	....	....	.79

Analysis No. 1 is the work of the writer, and is the results of many hours work. Each constituent was determined several times, and in most instances by different methods. It is not claimed that all of the results are strictly accurate, but it is safe to say that none are in greater error than one-tenth of a per cent. Individuals may disagree with the above statement, but owing to the great discrepancy in the several reports made no *one* individual can dispute the correctness of the writer's analysis. For these reasons it will be assumed as the standard and all comparisons made as such. Nos. 12 and 14 check so closely with that of the writer that he takes the liberty of stating that No. 12 is the work of Chas. Rueger, of Butte, Mont., while No. 14 was reported by Nelson T. Hasenpflug, of Cleveland, Ohio. Both of these chemists have taken the greatest of interest in the work, and the writer is greatly indebted to them not only for the chemical work done but for many useful criticisms.

A comparison of results between the first and second series of slags shows an improvement, but not a marked one. It is possible that the slightly better work shown in this series has come about through a perusal and study of the first, by the chemists coöperating. Attention is called to the fact that but a small number of the chemists reporting on the first series appear in the second. The writer does not believe the chemists reporting on the second series are superior to those of the first. It is not intended to call attention to poor work as a whole, but rather to criticise, in an effort to remedy the cause of divergent results. Inexperience cannot be considered as carelessness, and the writer prefers to ascribe a majority of the poor results, as a whole, to this factor. However, poor work has been done in this series by chemists holding responsible positions with concerns that can afford, and should require, better work.

The most important constituent of a straight copper slag is undoubtedly the copper. Hence the discussion of this element is given first and somewhat at length. Of the twelve results reported nine are satisfactory, and of these five are as near correct as can be made.

No. 2 reports his results obtained by the electrolytic method, but gives no particulars, hence the low figures cannot be accounted for. No. 3 precipitated the copper with hydrogen sulphide, ignited in a stream of hydrogen and weighed as sulphide. He further checked his result by ignition with ammonium carbonate. He overlooked the fact that lead by this method would be reported with the copper. Attention was called to the small amount of lead present when the samples were sent out. In the case of No. 4 a reason for the discrepancy is difficult to find. A glance at this chemist's work as a whole would lead one to believe it due to a lack of experience. His condemnation of standard and almost necessary methods of analysis shows a lack of something, to say the least. Of the above three chemists not one seems to have had any experience in copper work.

Before continuing the discussion the writer wishes to state briefly a method devised by himself, and which he believes superior to all methods yet proposed, not only as regards accuracy but time, which is of great importance. It permits the handling of a large quantity of slag as easily as a small quantity, and in either case without the accompanying mass of gelatinous silica. Either 5 or 10 grams of slag is weighed into a No. 3 beaker, 100 c. c. of hot water added and brought to a boil. HCl is added, a few c. c. at a time, that the boiling may not be interrupted, the beaker being given a rotary motion until danger of sticking is past. When largely in solution about 100 c. c. additional of hot water is added, and the copper thrown out by a rapid stream of hydrogen sulphide. This will require about five minutes, as the greater part of the iron remains in the ferrous condition. The copper sulphide is caught on a Gooch crucible, using a few drops of hydrofluoric acid to hasten the filtration. The sulphide of copper is then dissolved in nitric acid and bromine, and finally electrolyzed in the usual manner.

No other constituent has so many methods of estimation, and every chemist reporting has a favorite scheme requiring anywhere from one to four hours before it is ready for the battery or titration. No one has ever questioned the writer's scheme as given above, and he asserts that were it more generally used a great saving of time would ensue, and in most cases greater accuracy. Twenty grams have repeatedly been prepared for the battery in less than one-half hour. The precipitation on aluminium is tedious, and a trace or more of copper is almost invariably found in the filtrate, as is admitted by No. 5, who adds .2 of a milligram to his results.

Nos. 8, 9 and 14 used the method of the writer. No. 8 removed from the battery too soon, and determined the copper remaining in the solution by a colorometric method. No. 9 secured a dark deposit, due probably to too strong a current, which never gives perfectly satisfactory results. On dissolving in nitric acid and replating the deposit was bright. The writer intimated to this chemist the probable cause of blackening, but the said chemist refused to admit that the current was stronger than necessary. His statement, however, does not alter the fact that too strong a current will cause blackening of the cathode. No. 14 followed the writer's method exactly, but at the time the analysis was made was inclined to believe a roundabout method, including the complete removal of the iron, would give more accurate results. This chemist was also annoyed by the blackening of the cathodes, but discovered that he was using too strong a current. This blackening is more liable to occur with materials low in copper. It may not be out of place at this point to state that even though the cathode stem does not show a copper stain after raising, the solution should always be tested with hydrogen sulphide water after removing from the battery. Further, that because no copper is deposited on a fresh surface after the lapse of a half hour is no indication that all of the copper has been deposited. Two hours is often necessary. The remaining six chemists determined the copper in various ways. No. 5 took the nitric acid solution down to white fumes with sulphuric acid, and precipitated the copper on aluminium, followed by titration with cyanide in the usual manner. No. 12 took 5 grams down to fumes with sulphuric acid, filtered, dissolved in nitric acid and potassium chlorate, and again took down to fumes with sulphuric acid, then electrolized for twenty hours. No. 15 evaporated 5 grams to dryness without the use of sulphuric acid, took up with hydrochloric and precipitated the copper with hydrogen sulphide, then filtered and ignited the precipitate. Then dissolved in nitric acid and electrolized. No. 11 fused 10 grams with potassium bisulphate and ammonium fluoride, took up with dilute HCl, followed by precipitation with hydrogen sulphide. He then ignited and dissolved in nitric and hydrochloric acids and carried down to white fumes with sulphuric, followed by electrolysis in the usual manner. No. 10 weighed 5 grams into a casserole, added hydrochloric and nitric acids, followed by hydrofluoric, and ran down to white fumes wit hsulphuric acid. Then cooled and added water and strips of aluminium and boiled (!) two hours, and finally added hydrogen sulphide water to complete the precipitation. He then filtered and determined the copper by cyanide. No. 6 reported his result as obtained by the electrolytic method but gave no particulars.

It can readily be seen that all of the above methods give satisfactory results on this particular sample of slag. If time is a factor, and it is so generally considered, then the majority of the chemists reporting used a method requiring too much of a valuable factor. The simpler the method the less the number of details and manipulations, then the more accurate the result. The writer will concede that his method is open to slight improvement, but will not admit that any of the above methods will give more satisfactory or accurate results whatever the time required for the preliminary operation. He further claims that no other method has been published which admits the handling of a large quantity of slag without attending difficulties. Copper is an element that can be estimated accurately, and there is absolutely no excuse for poor results by one who assumes to be familiar with its estimation.

The next constituent to be considered is the silica. Dr. Hillebrand, of the Geological Survey, has done much to put the determination on a more accurate basis, and either many chemists are not reading current chemical literature or are *persisting* in the use of obsolete or inaccurate methods. It has been demonstrated beyond a doubt that a double evaporation and volatilization with hydrofluoric acid is absolutely necessary for the correct estimation of silica. It is true that statements have appeared to the contrary, but the fact is not changed. The results on this slag show what can be done when a definite method is followed. It must also be added that blasting at a high temperature for at least twenty minutes is another essential.

Of the fourteen reports on this element, Nos. 3, 4, 8, 9, 11, 12 and 14 are good, while 5, 10 and 15 are not far wrong. Nos. 6, 7 and 13 are poor, the latter most emphatically so. No. 11 evaporated twice but found no residue on volatilizing with hydrofluoric acid. This surprising statement calls for no comment. No. 7, 10 and 13 evaporated but once, and did not correct by volatilization with hydrofluoric acid. Thus, here are three results, all obtained by the same method, two of them radically wrong and all different. This is the usual method employed in smelter laboratories. Its use is necessary, but it does seem as though greater care might have been employed. Nos. 3 and 5 evaporated once and corrected with hydrofluoric acid. No. 2 and 9 did not specify the method used. No. 6 used a method to which attention was called in the writer's first report, but his result is not in conformity with what should be expected from this method. This chemist took the slag to dryness, dissolved in acid, filtered, ignited, and then fused with sodium carbonate and again ran to dryness.

By this method of procedure we would expect a double loss, yet the result reported is nearly 1½ per cent higher than the standard.

On the whole, the results on silica may be considered fairly satisfactory. The poor results are due, in most cases, to rapid methods, something entirely out of place in an accurate determination. There should be a balancing of errors in any rapid method of estimating silica, if ordinary care is used, but this is not usually found true.

A discussion of the results on iron naturally follows. Probably no element is more difficult of correct determination. This statement will undoubtedly be challenged, yet, until we have a method that can be depended upon, chemists will continue to differ. Iron wire is never pure, and one chemist uses a factor of purity of .996, while another thinks .998 is better. When zinc is used as the reducing agent iron is introduced in varying amounts and a blank is difficult to run. It is true that uniform results can be secured by the individual; but on comparing with another's work there may be a marked difference. The writer uses as a standard an iron ore on which several iron works chemists obtained almost identically the same results. He considers it superior to iron wire or the double salt of iron and ammonia. The method of reduction employed by Dr. Hillebrand is undoubtedly the best. It obviates the possibility of introducing iron or any other contaminating substance. In brief, the sulphuric acid solution of the slag is reduced with hydrogen sulphide, the copper sulphide filtered off, and the excess of hydrogen sulphide removed by boiling in an atmosphere of carbon dioxide. The solution is then cooled and titrated with permanganate in the usual manner. The writer followed this method, and after a little practice was able to produce perfectly satisfactory results. The results given in No. 1 is the average of a large number of determinations varying less than one-tenth between the highest and lowest. He believes his results as accurate as can be made. The method is not adapted to rapid work but is a sure check on the more rapid methods. The fact must be emphasized that by this method the titanium, which is always present in this slag, is not estimated, hence in the following discussion, results which are higher than the standard by one-tenth of a per cent may be considered as correct.

Of the eleven results reported Nos. 2 ,8, 9, 10 and 14 are good, of which No. 10 and 14 are exceptionally so. Nos. 5, 7 and 12 are fair. No. 4 shows results in conformity with the rest of the analysis. Nos. 6 and 13 are also very poor, showing a difference of 3.65 per cent.

No. 2 used the permanganate method but gave no particulars. No. 8 reduced with iron-free zinc (?), and titrated with bichromate. He believed his result a little high, owing to a weaker titrating solution than at first supposed. No. 9 did not state the method used. No. 10 reduced with stannous chloride and titrated with bichromate. No account was taken of the iron held up the silica. No. 14 secured all the iron by fusing the insoluble residue, then precipitated with ammonia, washed well, dissolved in HCl, reduced with stannous chloride and titrated with bichromate. He overlooked the small amount of iron which nearly always passes the filter paper on prolonged washing. This chemist expressed himself as dissatisfied with his standardizing material. Of the fair results No. 5 neglected to take into account the iron retained by the silica, which may vary from a trace to a marked amount, depending on the preliminary treatment. He probably suffered a further loss in throwing out the copper on aluminium. He reduced with zinc and titrated with permanganate. No. 7 used an ordinary rapid method but gave no particulars. No. 12 decomposed with acqua regia and without running to dryness, filtered out the silica and made an ammonia separation Then dissolved in HCl, reduced with stannous chloride and titrated with bichromate. Here should have appeared a loss of iron by that held up by the silica and the great liability of passing through the filter on washing the ammonia precipitate.

Of the poor results, No. 4 used the Zimmerman-Reinhardt method, which, in the opinion of most authorities, does not always give correct results. This chemist also precipitated with ammonia first and washed well. No. 6 precipitated the iron and alumina with ammonia and washed well, dissolved in sulphuric acid and titrated with permanganate, preceded probably by a reduction with zinc. Like several others he overlooked the liability of iron to pass through the filter on washing, but this might have been balanced by the iron in the zinc. No. 13 was obtained by what the chemist called the Colorado practice. On the whole the results on iron may be considered as fair and show improvement over those of the previous series. When all of the possible factors of error are considered it is very probable that a majority of the co-operators could secure first-class results if, in addition, due care were taken. Until we have a definite standard and a definite method of procedure there must follow a lack of uniformity.

Alumina has to bear a heavy load. Practically all of the errors are thrown upon it, and that even passable results can be secured is surprising. The "difference method," in the hands of the average chemist, is worse than useless. There are not a half dozen chemists in the United States to-day who can, by the use of this method, present accurate figures on any material. This statement will, of course, be questioned, but will not alter a fact. Of the whole fourteen chemists reporting on the slag 50 per cent failed to make returns on alumina. The writer naturally concludes that these failures are due to a lack of confidence in the difference method. Yet this lack of confidence was not at all common before the writer published the results on the first series. The method has been in use for years and its correctness seldom questioned. Undoubtedly thousands of analyses have been made in which the alumina has been reported from 25 to 100 per cent more than actually was present. This statement applies to all materials of which alumina is a constituent.

Of the seven who sent in results Nos. 4, 8 and 12 are very close to the standard. No. 14 is not very far out, while Nos. 2, 6 and 10 are not satisfactory.

The standard result was obtained by the phosphate method, which, notwithstanding numerous statements that results obtained by this method are low, the writer is prepared to prove accurate. It may give results low by one-tenth of a per cent, but this can be overlooked in almost any analysis. However, in the hands of a chemist working in the average laboratory the method gives results far superior to the difference method and without the disagreeable odor and taste of the phenylhydrazine method. The only objection in its present state is the great length of time required to wash the precipitate, but this is sure to be overcome shortly.

Of the results close to the Standard, No. 4 was obtained by the phosphate method and checked exactly by the difference method. In view of the fact that this chemist's iron oxide was higher than the standard by 1 per cent it would follow that his good result was an accident. In addition, his method of freeing the iron and alumina from other bases is open to question. The writer can only say that the coincidence was a remarkable balancing of errors. No. 8 obtained his figure by the phosphate method, and the result reported is the average of two widely varying results, hence as given in the tabulated statement is misleading and will not be considered. No. 12 is the work of Mr. Rueger, and is from every point of view the most satisfactory reported, not excepting the standard. Mr. Rueger used a method devised by himself, particulars of which can be found in the *Engineering and Mining Journal* of March 3, 1904. By the indirect or difference method he secured practicall the same results, but as his iron is a little high it would follow that his alumina, by the difference method, would be higher than by his own method. There are, however, so many factors that may cause error in the difference method that the last statement is made with due regard. The rather unsatisfactory result secured by No. 14 was obtained by the phosphate method. He made but one separation, whereas two are necessary in a material so high in iron. Taking the last statement into consideration the result is satisfactory. Of the unsatisfactory results No. 2 is most decidedly so. The determination was made by the phosphate method, but evidently this chemist's experience with the method was limited. No details were given, hence a criticism must be limited to the above statement. The writer's early experience with the method gave results entirely too low, but with a very little practice satisfactory results became the rule. No. 6 separated the oxides or iron and alumina with sodium hydroxide; he made but one evaporation for silica and precipitated the iron and alumina twice with ammonia. Hence a serious error may have been introduced, at least from failure to evaporate twice for silica. The statement is made in most text books that sodium hydroxide, free from alumina and silica, is difficult to obtain. In the writer's experience he has never found a sample free from these elements. If due preliminary precautions were taken, the sodium hydroxide made in the laboratory by the individual and from the pure metal, using platinum dishes throughout, except for the fusion, the method would give as good results as the best. No. 10 used the phosphate method, but instead of the theoretical factor .4185 he used .45, and this together with but one precipitation is probably the cause of the high result. The high factor was undoubtedly used to offset the low results this method is supposed to give.

As a whole, the results on alumina are far superior to those of the first series. That this is due, in part at least, to the abandonment of the difference method is very evident. This latter method is absolutely worthless as ordinarily practiced. Mr. Rueger's method has not been adversely criticised, but does not give better results, in the writer's hands, than the phosphate method. There is much to be desired in the way of shortening this method, and that this will come in time cannot be doubted.

(*To be concluded.*)

The Ruthenburg and Acheson Furnaces.

By F. A. J. FitzGerald.

An early example of a furnace, in which the material under treatment was heated by passing a current through the charge, was that of Cowles, in which aluminium alloys were made. The objection to a furnace of this type is that an excess of carbon must be used in order to make the charge a conductor of electricity. In making alloys such as ferrosilicon, etc., at the present day, the charge is heated, in part, by means of the current passing through the mixture and in part by an arc, so these furnaces are not purely of the resistance type.

The Ruthenburg Furnace.

There is so much evidence of misconception of the working and object of the Ruthenburg furnace that it seems to be advisable to consider it in some detail. The primary object of this furnace is to agglomerate magnetic iron ore by causing the individual particles of the finely ground material to fuse or frit together, thus forming lumps or masses of relatively large size. This may be done by passing an electric current through a mass of ore; but since the granular ore is a very poor conductor of electricity it is necessary to use a furnace in which the cross-section is very large compared to the length. Moreover, it is not desirable to fuse the ore into large, solid lumps, but merely to frit it together so that a quantity of the treated material will form a porous mass.

The Ruthenburg furnace consists of a powerful electromagnet in horse-shoe form, the poles of the magnet being electrically insulated from one another, since they form the terminals of the furnace. The magnetite, in a finely ground condition, is fed between the poles of the magnet and held there, forming a bridge which connects the poles electrically. The poles are connected to a source of current which thus passes through the bridge of ore. Now it is evident that in a circuit of this kind the places of highest electrical resistance will be at the points of contact of the ore grains, consequently, it is at these points that the greatest generation of heat will occur, thus causing the grains of ore to fuse together. The heat generated at the points of contact is conducted into the bodies of the grains themselves, thus raising the whole mass to a bright, red heat. At this temperature the magnetism of the ore disappears, and the fritted mass falls from the poles of the magnet.

The rest of the process has nothing to do with electric furnaces; but it may be mentioned that the scheme is to cause the heated ore to fall from the poles of the magnet into a suitable receptacle, carefully insulated, so that the heat given to the ore is not allowed to escape, and then to pass reducing agents, such as carbon monoxide, through the porous mass, with the object of reducing it to metallic iron.

Mr. Ruthenburg has asserted that in working his furnace he uses only 250 kw-hours per ton of ore, and this has caused much adverse criticism, due to a misunderstanding of the nature of the process. It must be remembered that the mass of ore is not fused, but merely agglomerated and raised to a temperature at which reduction with carbon monoxide gas will occur. The fusing temperature is only reached at the points of contact of the ore grains. Keeping this in mind it may easily be shown that no extraordinary efficiency of working is demanded by Ruthenburg's assertion.

Let it be assumed that the ore mass is to be raised to a temperature of 1000° C., and that the specific heat of magnetite is 0.154. Then if W is the watt-hours required to raise 1 kilo. of ore from 0° to 1000°, and there is no loss of heat, we have:

$$W = \frac{4.2 \times 0.154 \times 1000 \times 1000}{3600} = 180$$

Where 4.2 is the conversion constant.

For a ton of 2000 pounds of ore the energy required under the assumed conditions would be 164 kw-hours. Thus, Ruthenburg's assertion only demands an efficiency of 65.6 per cent, which does not seem to be impossible. It should be noted that the specific heat of magnetite used in this calculation is that determined by Kopp, between 18° and 45°. It may be that the average value between 0° and 1000° is greater than this, and if so, the efficiency called for is greater.

The Acheson Graphite Furnace.

The Ruthenburg furnace shows clearly the great advantage of having a material in granular form when it is desired to heat it electrically for, as is well known, magnetite in a fused form is a comparatively good conductor of electricity. This is further illustrated in the Acheson furnace for converting carbon into graphite where the heating is done by passing the current through the charge. Here again the main generation of heat occurs at the points of contact of the individual grains of carbon.

In the earliest patent connected with the Acheson process of manufacturing graphite[1] the object is the purification of impure carbonaceous materials, such as ordinary coke, by heating these in a granular form to such a high temperature that the impurities, silicon, iron, aluminium, etc., are volatilized. In the subsequent patents, relating more especially to the manufacture of graphite from anthracite coal[2], and the process of converting petroleum coke into graphite[3] the volatilization of impurities also enters in. The theory of the Acheson process is that graphitization is brought about by the formation and subsequent decomposition of carbides, hence a carbonaceous material, such as anthracite coal, is chosen, because it contains a certain quantity of carbide-forming materials distributed throughout it. On the other hand, when a very pure carbon, such as petroleum coke, is employed, the carbide-forming substance is mixed with the charge. The various reactions which occur in the furnace undoubtedly take place at widely different temperatures, the absorption and release of heat energy that accompany the chemical changes are unknown quantities; finally, the temperature of the furnace is unknown, so it is obviously a hopeless task to make any estimate of efficiency.

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[1] U. S. Patent No. 542,982, July 23, 1895.

[2] U. S. Patent No. 645,285, March 13, 1900.

[3] U. S. Patent No. 711,163, October 14, 1903

E. G. ACHESON.

As to the furnace used for converting carbon into graphite the only published information is that supplied in the patent[4] on making graphite from petroleum coke. The furnace is described as having a length of 30 feet, and a sectional area of 18 by 14 inches. The loss of heat by radiation from a furnace of these proportions must be considerable. In the case of a furnace of rhomboidal form, as near an approach as possible to a cube should be made in its construction, since that rhomboid presents the smallest surface for a given volume. The furnace described in the patent contains a charge of 52.5 cubic feet. In the form of a cube this volume would have a surface area of 84 square feet; but with the dimensions given the surface is 164 square feet. Therefore, other conditions being the same, the radiation losses would be nearly double in the case of the furnace described as compared with a furnace of cubical form. The difficulty encountered in approaching the cubical form is found in the low electrical resistivity ultimately produced in the charge. At first the resistivity of the charge is so high that a resistor, in the form of carbon rods embedded in the charge, is used to connect the terminals of the furnace, and the current is confined nearly altogether to this resistor. But as the surrounding charge becomes hot, and the carbon is converted into graphite, its conductivity is greatly increased, so that finally the whole mass becomes a conductor. By regulating the voltage at the terminals of the furnace the power is kept constant; but in order to do this with a shorter furnace than that described in the patent, it would be necessary to use a lower voltage and larger current than is at present the practice.

In the present state of our knowledge we are ignorant of what the losses by radiation in furnaces of this type are, hence it is impossible to say what gain in efficiency would be obtained by approaching more closely the ideal form. Study of this point would be very useful. Let it be assumed that the furnace described above uses 750 kw. and that when the highest temperature is reached a current of 7500 amps. at 100 volts is used. From these data it may be calculated that if a current of 15,000 amps. at 50 volts could be obtained, the dimensions of the furnace would be: length, 15 feet; sectional area, 24 by 21 inches. This would give a surface to the charge of 119 square feet. Similarly, with 30,000 amps. at 25 volts the dimensions would be: length, 7.5 feet; sectional area, 36 by 28 inches, giving a surface of 96 square feet. These furnaces, more nearly approaching the cubical form, would be more expensive to install than the furnace described in the patent, because it would be necessary to use much more copper, in order to carry the currents of large ampereage. Moreover, it would be more difficult to obtain a good power factor in the furnace circuit. Still another point to be kept in mind is the increased size of the furnace terminals, which, being good conductors of heat as well as electricity, would increase heat losses in that way. The important thing to determine, however, is where the greatest efficiency is obtained.

Electrode Furnace.

It was originally proposed to graphitize carbon articles by the Acheson process in a furnace where the articles were embedded in powdered carbon[5], but in a later patent[6] an improved method of heating electrodes to the graphitizing temperature is described. In Fig. 1 is shown a vertical longitudinal section of the furnace, where *c c* are the terminals, *e* the carbon electrodes arranged in piles, *g* granular carbon, *h* refractory lining, *i* covering material of ground coke and sand. It will be observed that when the current is thrown on, it passes alternately through sections of the furnace filled with granular carbon or electrodes, and as the resistivity of a mass of the granular carbon is very much greater than that of the electrode the chief generation of heat takes place in the former. Herein lies the chief feature of the invention, for by this ingnious arrangement the use of currents of very low voltage and enormous ampereage is avoided.

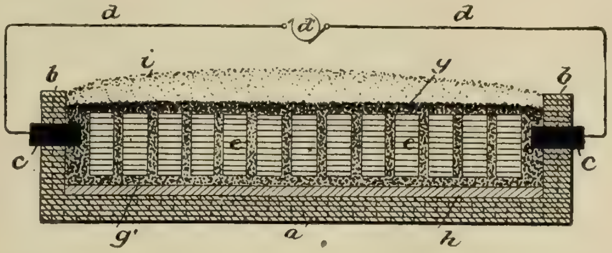

FIG. 1.—LONGITUDINAL SECTION OF ELECTRODE FURNACE.

Details of the construction of an electrode furnace are given in the patent specifications, and from these the following table has been drawn up:

Distance between terminals of furnace	360 inches
Length of space filled by electrodes	302 "
Length of space filled by granular carbon	58 "
Length of electrodes under treatment	24 "
Width of electrodes under treatment	5 "
Height of pile of electrodes	17 "
Initial voltage	210 volts
Initial ampereage	1,400 amps.
Volts at end of five hours	210 volts
Amperes at end of five hours	3,600 amps.
Final voltage	80 volts
Final ampereage	9,000 amps.

From this table we may calculate the resistance of the furnace at various stages of the run, assuming that if the volts and amperes were obtained from a voltmeter and ammeter, respectively, the power factor is 1:

	Ohm.
Resistance of furnace, initial	0.150
Resistance of furnace at end of five hours	0.058
Resistance of furnace, final	0.0089

For the resistivity, *i. e.*, the resistance of 1-inch cube of the electrodes we have the following data, assuming that at the end of five hours the electrodes have been heated to a temperature where their resistivity is reduced one-half:

	Ohm.
Resistivity of electrodes, initial, cold	0.00124
Resistivity of electrodes at end of five hours, hot	0.00062
Resistivity of electrodes, final, graphite	0.00032

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[4] Ibid and "Electrochemical Industry," Vol. 1, No. 4, p. 130.

[5] U. S. Patent No. 617,979, January 17, 1899.

[6] U. S. Patent No. 702,758, June 17, 1902.

To calculate the resistance of the electrodes in the furnace we have the equation:

$$R = P\frac{l}{A}$$

where R is resistance, P resistivity, l length and A sectional area. From the first table we find that the total length of the space filled with electrodes is 302 inches and the sectional area 17 × 24 = 408 inches. Thence the following table is obtained:

Resistance of electrodes in furnace,	Ohm.
Initial	0.00092
End of five hours	0.00046
Final	0.00024

Since the energy developed in any part of the furnace is given by the product of the resistance and the square of the current, we have the following table for the distribution of energy:

	Granular Carbon.	Electrodes.
	Kws.	Kws.
Initial	293.82	0.18
At end of five hours.	750.05	5.95
Final	700.50	19.44

It should be noted that the resistivity of the graphitized electrodes assumed in the calculations is that found at ordinary temperatures. It is, of course, different when the electrode is hot; but this would not seriously affect the results given in the last table, which shows clearly the important part played by the granular carbon in the working of the furnace. When the carbon electrodes to be graphitized have a circular cross-section the furnace is built as shown in Fig. 2, since with this form sufficient resistance is obtained at the points of contact of the electrodes.

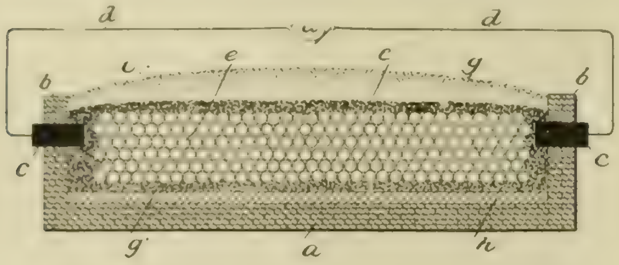

FIG. 2.—FURNACE FOR CIRCULAR GRAPHITE ELECTRODE.

New Assay Furnace Tools for Works Laboratories.

In our September issue we published the first part of a paper recently presented by Dr. Edward Keller before the American Institute of Mining Engineers on labor-saving appliances, devised by the author for the Baltimore laboratory of the Anaconda Copper Mining Co. On account of limitations of space we had to hold over the description of Dr. Keller's new assay furnace tools, which is herewith given.

The implements used in performing the furnace work in assaying are shown in Fig. 1. A and B are the traditional tongs, universally used for handling singly the scorifiers and the cupels. The former has been entirely replaced with a fork C, with which a set of twenty scorifiers can be handled at one time. In silver assaying each set of scorifiers is placed in the muffle twice and taken out twice; first put in the muffle for the incineration of the filters, then taken out for the addition of the test-lead, then returned for scorification, and finally taken out for pouring the slag and molten lead into the molds. By the use of the fork, which works perfectly if the muffle be properly supported so that it will not sag to any marked extent, sixty handlings are reduced to three.

Four scorifiers, constituting a longitudinal row in the muffle, are poured at one time by means of a pair of tongs D, and with a little practice the pouring is made just as easy as with a single scorifier. It is necessary only that the mold correspond to the arrangement of the scorifiers and that the pockets are shallow.

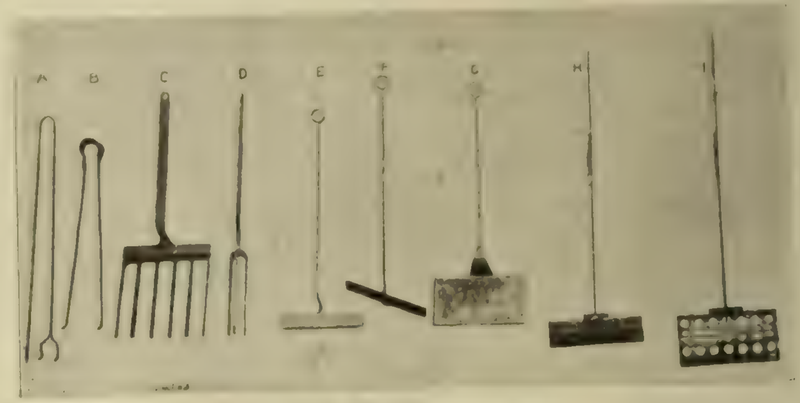

FIG. 1.—ASSAY FURNACE TOOLS.

The cupels are placed into, or taken from, the muffle in sets of one or more rows, by means of the tools E, F and G, an idea which was first put into practice by the author's brother, Richard Keller, of Durango, Col. E and G are sharp-edged shovels, the latter having upturned sides. F is a rabel, with which the cupels are raked onto the shovel, removed therefrom to the place where they are to be deposited by placing it behind them and withdrawing the shovel.

The tool H and I is entirely new, and by its use one or more rows of cupels in the muffle may be charged with the lead-buttons from the scorifiers. Fig. 2 shows the idea of the device more clearly. It comprises a top sliding-plate with openings corresponding exactly to the position of the cupels. The openings in the lower plate correspond exactly with those of the upper one; the plate, however, rests on two adjacent sides extended downward at right-angles to the plate and to each other, thus forming two closed sides of the instrument, one at the front, and the other at the right-hand side. The height of these sides is such that, when resting on the bottom of the muffle, the bottom plate will be some distance above the cupels and, by a slight pull forward and a push to the left with the handle of the instrument, the set of cupels will be perfectly aligned in both directions, and the apertures in the lower plate will exactly cover the tops of the cupels.

The lead-buttons are placed in the apertures of the upper plates and rest on the lower plate before introducing the instrument into the furnace, and when it is placed over the cupels, which have been properly aligned in the muffle, the upper plate is pushed forward to a stop-point, bringing the

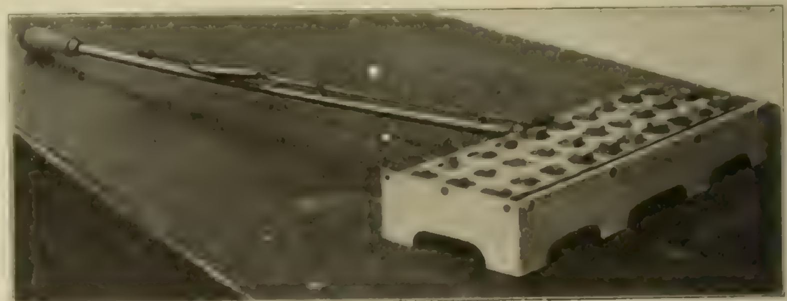

FIG. 2.—DEVICE FOR CHARGING SCORIFIER BUTTONS INTO CUPELS.

apertures of the two plates to register, thus causing the lead-buttons to drop down into the cupels. The handle of the upper plate runs through guides fixed to the handle of the lower plate; both handles are connected by a spring, which acts as a brake when the upper plate is pushed forward to drop the buttons, and also serves to bring it back into its original position, in which the buttons cannot drop through the apertures in the lower plate.

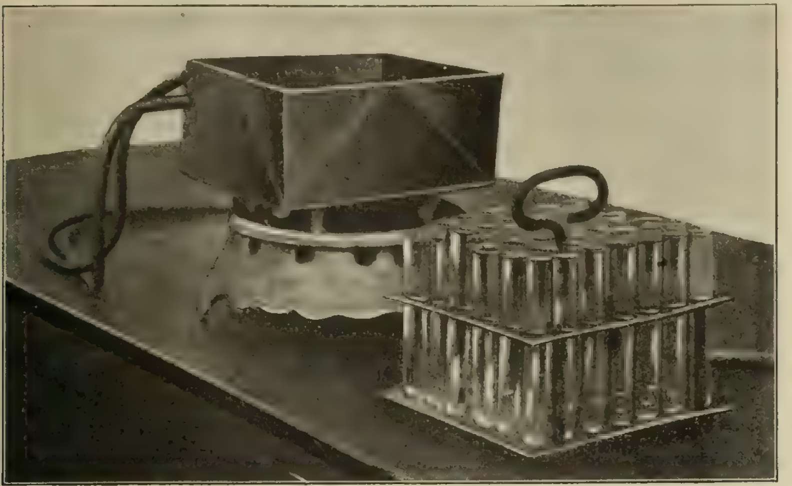

FIG. 3.—CONVENIENT PARTING BATH.

Charles Tookey[1] first recommended the use of hydrochloric acid (1HCLaq,2H_2O) instead of a brush for cleaning the buttons, and for this purpose a small silver dish and tray having perforated pockets give excellent satisfaction. By the use of this device fifty or more beads at a time can be treated, washed and dried without transfer.

Fig. 3 shows a very handy parting-bath, which though old in principle has not been in general use. The vessel is a constant-level water bath, and the tray an ordinary test-tube holder. The silver beads to be parted are dropped into the test-tubes, and the latter filled with dilute nitric acid of a strength of one of acid (sp. gr. 1.42) to nine of water. The water in the bath is first brought to the boiling point before the tray with its contents is set into it. Treated in this way, the gold almost invariably remains in the form of a small coherent bead, even from an alloy as low as one part of gold to 500 of silver.

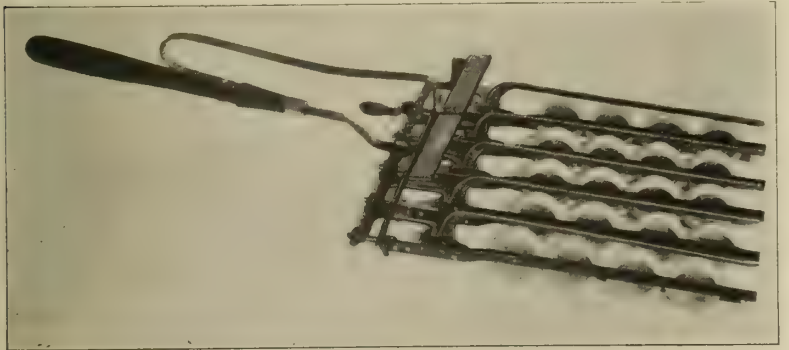

FIG. 5.—MULTIPLE TONGS; SCORIFIERS TIPPED FOR POURING CONTENTS.

Dr. Keller recognized that the system of handling everything in sets was incomplete as long as he was unable to take a whole set (twenty) scorifiers from the muffle and pour their contents simultaneously into the molds. Recently a tool for that purpose, shown in Figs. 4 and 5, has been perfected. It is composed of multiple tongs, corresponding to the five longitudinal rows of scorifiers in the muffle. The lower part of each pair of tongs consists of a fork, on which the scorifiers rest, and one of whose prongs is rectilinearly extended through two bearings in a frame, and held in position by collars. This extension is free to revolve in the bearings, and it is the axis of rotation of the tongs. To each of them is attached, at right angle, a lever, extending upward at an angle of 45°, and all the levers are connected by slotted joints to a cross-rod. Therefore, if by means of a crank, fastened to the end of one of the extended prongs, one of the forks is turned and the scorifiers tilted to the desired angle, the others perform the same rotation. The center of gravity of the scorifiers lies to one side of the rotation point, and they would, therefore, on being lifted, tilt in that direction; this, however, being prevented by the cross-bar resting against a post at that end of the frame toward which the inclination tends.

The scorifiers are clutched by the upper prong of the tongs, which is fastened to a spring on a post of the fork below, and which is free to move in a vertical plane, the pivotal point lying over the spring and post. By bringing pressure on the extended ends of these clutch bars behind the pivot, their other end will rise from above the scorifiers, and thus release these, or permit the placing of them onto the tongs. The pressure exerted on the rear ends of the clutches is accomplished by means of a cross-bar fastened to a spring-bar, which is itself riveted to the handle of the instrument; all of which may be plainly discerned in the illustrations. In pouring the contents of the scorifiers, the frame of the tool rests on the edge of the mold, leaving the tongs free to turn.

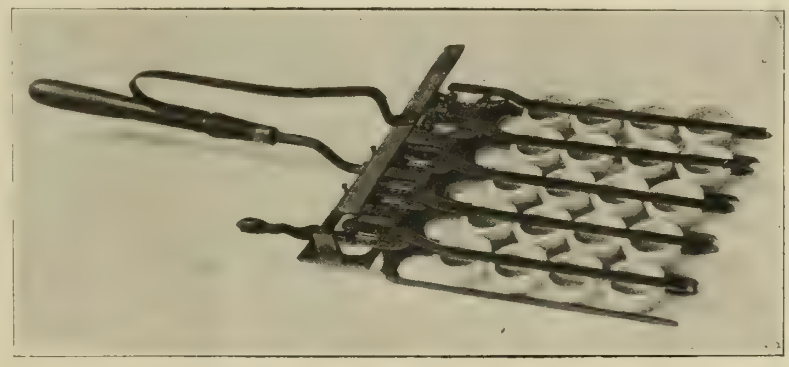

FIG. 4.—MULTIPLE TONGS; SCORIFIERS IN POSITION FOR REMOVAL FROM FURNACE.

The introduction of these new labor-saving appliances in Dr. Keller's laboratory has resulted in economy in several ways. Much labor has been saved; breakage of expensive glassware has been very largely eliminated, and the time of the furnace work, and, consequently, the consumption of gas has been much reduced. Furthermore, the gain has been a moral one, and work formerly regarded as tedious has become more of a pleasure; especially has the sojourn in the furnace room during the hot summer months been rendered cooler by being greatly shortened.

The appliances described in Dr. Keller's paper may eliminate the laboratory boy; but if not, they will make him more reliable. They also increase enormously the quantity of chemical work that one man can do in a day. In the domain of Dr Keller's own laboratory the change from the old system to the new is considered to bear about the same relation as the change from ancient horse-car to the modern rapid transit

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[1] Journal of the Chemical Society (London), vol. 23, page 366 (1870).

The Electrochemistry of the Metallic Arc.

By Isador Ladoff

(*Concluded from page 342.*)

Observing the metallic arc, produced by a high tension, alternating current in a mirror rotating with a suitable angular velocity, many periods of the alternating current may be distinguished. In the middle of each period a maximum of light is noticeable, and then a lowered luminosity to the end of the period is to be observed. The color of the copper arc under these conditions appeared yellowish-red and violet. In the middle or interval between two periods there was a dark space followed by a bright white scarlet. The last was the discharge spark relighting the arc, extinguished at the end of each half period. Blondel was the first to point out, that a high tension is necessary to relight the metallic arc.

Guye and Monasch (Ecl. El. 35, p. 18, 1904) showed that in metallic arcs produced by high tension, alternating currents, the tensions after a certain distance between the electrodes is reached, increase with diminishing length of the arc instead of dropping when the current is kept constant, and called this zone in which the arc shows this unexpected behavior "the critical zone." In the silver and copper arc the increase of tension is so high in the "critical zone," that no measurements can be taken. However, the "critical zone" appears only in short arcs. It was determined by Guye and Monasch for silver, copper, gold, platinum, nickel, aluminium, magnesium, cadmium and chemically pure iron.

The appearance of such short arcs is different from those of longer dimensions. It produces a slight humming noise, due to the change of periods in the "normal" zone, but causes a sharp hissing sound in the "critical" zone. The only metal not showing any "critical" zone was iron containing carbon. In the last the tension dropped with the diminishing length of the arc if the current was kept constant and everything was normal. However the "critical" zone appeared in chemically pure iron.

The "critical" zone appeared only in pure metallic electrodes of a length of 0 to 3 mm. The presence of carbon in the electrodes does away with the "critical" zone entirely, and the "normal" zone starts with O length of the arc. The reflection of a metallic arc in the "critical" zone in a rotating mirror represented an irregular sequence of bright and dark spaces. Under identical conditions a carbon arc did not show any discontinuity. Monasch (Dissertation Darmstadt 13, 7, 1903) demonstrates the discontinuity of the arc between pure metals in the "critical" zone with the aid of Tomasina's electroradiophone. The telephone of the electroradiophone produced sounds when the arc produced between pure metals was shorter than 3 mm., and there were discontinuous discharges in the arc. In a carbon arc of the same length the telephone remained mute.

In this brief review we will have to pass in silence some interesting phenomena in the arc between a metallic and a carbon electrode.

However our review would be incomplete if we would not say a few words of the arc in metallic vapors in an exhausted space.

We will follow the investigation of E. Weintraub (Philosophical Magazine VI., Series H38 Feb., 1904, Vol. 7).

The different methods by means of which a mercury arc can be started in an exhausted tube may be summarized as follows:

1. Bringing the cathode into contact with the anode and separating the two electrodes.
2. Application of high voltage to the two electrodes.
3. Bringing a cathode in contact with an already active electrode (E. Weintraub's method).
4. Mere mechanical agitation.
5. Contact of the cathode with ionized vapor (Weintraub's method) (p. 101).

In consequence of the high vacuum and volatility of mercury, the arc can be made of any desired length. In common with all the arcs, the voltage across the mercury arc varies only little with the current. If, for instance, the current increases four times, the voltage will vary by only 5 to 6 per cent. In the first approximation, therefore, the ***resistance of the arc can be considered as inversely proportional to the current.*** In contradiction to the ordinary carbon arc, however, the voltage varies in the same sense as the current (p. 105). The conductivity and the luminosity of the mercury arc do not go parallel, the ***maximum conductivity nearly coincides with the minimum light emission*** (p. 107). To be perfectly correct we ought to distinguish between three kinds of mercury vapors in the arc stream; one ionized and conductive, the other non-conductive, but light-emitting, and third, non-conductive and non-luminous ordinary mercurey vapor (p. 108).

There is strong evidence in favor of the assumption that the cathode is the electrode at which the primary generation of ions takes place (p. 111). The anode must have a relatively high melting point, must not enter into combination with mercury, must have in its normal condition as little gas occluded as possible, etc. The material of it does not seem, however, to affect the nature of the arc in the least. The action of the magnetic field on the arc is very peculiar and not easily accounted for. The observed phenomenon can be briefly described as follows: The field and the arc being both horizontal and perpendicular to each other, the arc is deflected up or downward, according to the common rule of the action of the magnetic field on a current. The deflection downward is accompanied by a motion of the bright cathode spot along the surface of the cathode in the direction of the current; the arc is thus lengthened, and when it reaches the wall of the tube it digs into the mercury where the latter is in contact with the walls of the tube. If the direction of the current in the arc is reversed, the arc is pushed upward and thus tends to become as short as it possibly can, so that the spot on the cathode moves in the same direction as before. Changing the direction of the field changes, of course, the direction in which the arc is deflected, as well as that in which the cathode spot moves (p. 113).

The characteristic of arcs in vapors of alkali metals except the spectrum, are essentially the same as those of the mercury arc. The electrodes possess the same properties. The potential drop across the arc is of the same magnitude as in the mercury arc. The amount of light given off is very slight.

The alternating current phenomena in the mercury arc, although very interesting from both scientific and technical point of view, do not concern us much here.

Metallic oxides giving electrolytic arcs deserve our attention next.

E. Rasch (Elect. t. Zeitschrift, Berlin, 22, p. 155, 1901) produced arcs between electrodes of magnesia, calcium oxide, oxide of thorium and oxide of Zirconium and arrived at the conclusion that the arc produced by these oxides shows a great luminous efficiency. The oxides used by Rasch belong to the conductors of the second class, which, as is known, do not conduct while cold. With increasing temperature their resistance diminishes. An arc between them may be produced after the electrodes were raised to a certain temperature. Rasch, therefore, proposed to accomplish this preliminary raise of the temperature of the electrodes by means of an ordinary carbon arc. Some metallic oxides, metallic silicides and metallic borides possess a high melting point and a high temperature of evaporation, and their arcs are determined by a high temperature, especially in the tips of the electrodes.

The spectrum of the electrolytic arc contains few ultra but mostly light-giving yellow-green rays. Especially magnesium and zirconium oxide are claimed to approach sunlight in their character.

Rasch makes the distinction between electrolytic pencils possessing very high resistance while cold, "hard electrodes" and electrolytic pencils possessing moderate resistance. When cold, "soft electrodes." The arc of "soft electrodes" is as unsteadily as metallic arcs. Rasch did not succeed to ascertain in soft electrodes any increase of tension parallel to the increase of the length of the arc. This may be due to the difficulty of taking exact measurements of tension on account of the softening of the tips of the electrodes in the arc. Rasch determined the relation between current, tension and length of arc according to Thomson's formula $E = a + \frac{CL}{J}$ giving the value for "a" $= 31.35$ volts.

Nernst (Electroch. Zeitschrift, Berlin 22, p. 256, 1901) found in his experiments with "soft electrodes" a great loss through the "burning of material" (Abbrand) in the arc, and what is especially remarkable, that *the consumption of the negative electrode is stronger than of the positive, just the reverse of the carbon pencil.*

There were many attempts to utilize the electrolytic arc as a source of light, attempts that crystalized in a number of patents, but did not lead to any technical success. The fundamental idea of all these attempts to utilize the electrolytic arc for lighting purposes consists in the combination of the oxides possessing lumeniferous properties with non-luminiferous but more conductive oxides. Some inventors went even so far as to propose to make the surface of the pencils conductive by surrounding them with a metallic tube or wire net, or even reducing the oxide *on the surface* to a metallic state.

My own experiments seemed to me to prove conclusively that only titanium (zirconium appears to be excluded on account of commercial considerations) alloyed with some other well conducting metal, for instance, copper or iron, promises to be an ideal material for arc-light pencils. (*Electrical World and Engineer*, Vol. 45, p. 757, 1905.)

The light produced by titanium pencils is agreeable to the eye and pure white in color, and produces the entire spectrum, with especial luminosity in the yellow zone.

The candle-power efficiency of titanium pencils is very high. At corresponding current values and voltages the energy consumption per mean spherical candle-power is from one-third to one-fourth of that required by carbon electrodes.

The light from the titanium arc, as shown by the distribution curves, is produced in about equal amounts above and below the horizontal. It will be necessary, therefore, to use a lamp with a reflector to obtain the best results of illumination.

When operated under proper conditions, the life of the titanium pencils is much greater per inch of material than carbons. The rutile pencil gives longer life than the ferro-titanium, and with the former there is no doubt that a pencil can be designed to give as long a life as the present inclosed arc lamp.

The light efficiency of the ferro-titanium pencil is higher than the rutile, but the difference is so slight that the superior life of the rutile makes it more desirable.

Very superior results are obtained by using titanium pencils in both electrodes. While objections to this might be revealed by further investigation, or an electrode of some other material might be found to give equally good results, this method of operation offers a promising field for investigation. For the same degree of illumination the use of these pencils will reduce the consumption of energy from one-third to one-half that used in the present carbon arc lamps.

The Application of Peat-Fuel.

By L. A. Stillings, Ph. D.

The advisability of seeking cheaper fuel for use in many industrial plants is more than ever apparent in those sections of the country where water-power is not abundant. Competition is growing, and unless the poorly located manufacturer can obtain a cheaper fuel as a substitute for water-power, only the manufacturer who is able to locate or transfer his plant to a water-powered locality will be able to stand the general decrease in price of production.

The great value of peat-fuel is unknown to the average person in the United States. We do not yet appreciate the vast resources offered all over the country for a cheap, clean fuel. When we find that the United States has more than 100 times the acreage of peat to that of all Europe and that Europe has been to a great extent dependent on peat as a fuel for many years, and, at that, only used up a small portion of its peat bogs, we can see the immense future of such a fuel in this country—either in connection with steam or gas power plants.

Since the present use of peat as a fuel will touch only those factories already equipped for steam power, I shall dwell at greater length on those processes. The smaller factories will do well to try the raw peat, while I should advise the larger ones to use the coked product or to refit their works with gas apparatus.

The cost of fuel is governed by its heat and waste. The use of both coal and wood is very wasteful, only a comparatively small part of the heat units being utilized. With coal the clinkers, with wood the live embers which drop through the grate are an additional source of loss. When peat is first placed on the gate it burns with a short, blue flame, which gradually becomes a yellowish glow. It emits an intense heat which is easily controlled by draughts. A peat fire will burn until utterly exhausted, and is nearly smokeless and ashless. The smoke of peat, unlike other fuels, is good for the lungs, and serves as an excellent deodorizer.

The use of peat as a fuel dates back many centuries. The ancient Roman naturalist, Pliny, tells us of Teutonic tribes on the borders of the North Sea, who dried and burned "mud" (peat). In the last century, when fuel for steam engines was required on a larger scale, and, in very late years, high prices of coal have attracted a renewed and most active attention to the use of peat.

The bogs in which peat is contained cover vast areas in the north temperate latitudes, both in Europe and the United States. In Germany they cover nearly 30,000 square kilometers (11,580 square miles), and in Ireland almost one-tenth the entire country. The depth is very variable, but is on an average 5 to 7 meters (about 16 to 23 feet) and more. There are bogs existing in Ireland 15 meters deep (nearly 50 feet). It may be estimated that 1 square kilometer (0.386 square mile) of 5 meters (16.4 feet) depth will give about 700,000 metric tons of dried peat, hence it will be seen that the amount of fuel in these bogs is enormous.

Peat is decaying vegetable matter which in centuries of time would become a form of coal. This organic decomposition is best expressed by the equation:

$$6C_6H_{10}O_5 = 7CO_2 + 3CH_4 + 14H_2O + C_{26}H_{20}O_2$$

Carbon dioxide and methane (marsh gas) being given off, and the resulting mass in water being peat. Eventually, through the pressure and heat generated, this would become coal. The approximate percentage composition of peat is: 16.4 per cent water, 41.0 per cent carbon, 4.32 per cent hydrogen, 23.8 per cent oxygen, 2.6 per cent nitrogen, 11.9 per cent ash constituent, with a specific gravity of 1.05. Sulphur is almost never found. The ash constituent will vary from a small fraction of 1 per cent up to 15 per cent, the average of the peats of the United States being 3.07 per cent, while that of the German peats is 7.9 per cent.

Peat as a fuel ranks next in order after lignite, in fact, it is of similar construction, and only differs in being of a more recent geologic formation. It contains more water and is but slightly carbonized, and, therefore, has a correspondingly lower thermal value.

Peat is known in general as the rich, dark brown mud found in marsh and swamp lands. However, one should not forget that all rich earth in which plant life grows readily is a form of peat, and will be easily burned if it is carefully dried.

The European peats are mostly composed of decayed mosses and grasses. In the United States we have several kinds of peat. The main one, however, is like the European peats, and is composed of decayed sphagnaceæ. On the Atlantic sea coast we have a variety of so-called "salt march" which produces a peat which would not be of any use for our purposes of fuel. In Europe we have the general term "turf" (German torf) applied to peat.

In the chemistry of the earlier nineteenth century there were many queer analyses of peat. The term "humus" was applied to the deep-brown earth of the bogs, and according to analysis was composed of humic acid. Later, humic acid was found to be a comopund of rather complex nature, which they called crenic and apocrenic acids. The great bogs of this country are: the Dismal Swamps of Virginia, the Everglades of Florida, the Savannahs and rice swamps of Georgia, the great lagoons of the South and the vast sycamore and cedar swamps of the West. Almost all of New England is one vast peat bog These bogs are coal measures in the primary process of formation. All coal has been mud, and hence, many kinds of mud can now be converted into a substitute for coal. The great value of one of the component parts of peat has only recently been investigated. There is a substance called "pentosane," which comes naturally in all peat and which serves as a natural binder under compression. Its exact nature has not been worked out, as we cannot isolate it or make any compounds with it.

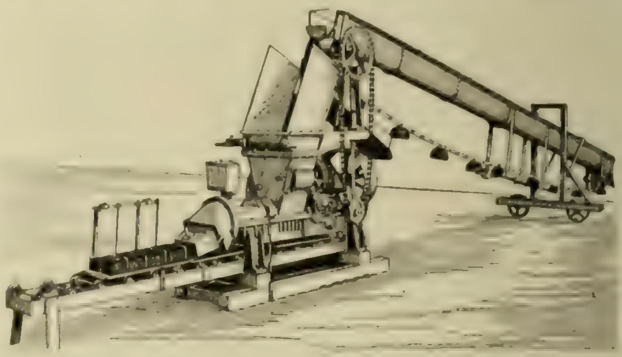
Peat is all organic matter, formed from mosses and other minor plant life which are submerged in water, and thus preserved in the bogs. As it stands, the greater part is water, only about 10 per cent being dry matter. In its use, therefore, much water must be removed, and this is the main difficulty in manufacturing. The general rule is to "air-dry" and then finish by pressure and a slight raising of the temperature. In almost all districts the peat is spread out on the surface of the bog; only in certain very rainy countries, as in Bavaria, Styria and Western Norway, shelters are sometimes used. Peat thus dried in the air will contain 20 per cent to 40 per cent of water.

The ancient method of digging peat was by hand, a special form of spade, called a "slane," being used, which cut a block of a special size. Hand-cut peat is still much harvested in farming regions. When cut by hand, peat is of a very loose consistency and of large bulk, a hectoliter weighing 20 kilograme (12 pounds per cubic foot). This would make it necessary to cut 9 cubic feet to equal 1 cubic foot of coal, and transportation, storage and fire room would have to be larger in proportion.

In order to reduce the volume, peat-milling or crushing machines are now commonly used. In these machines the fibers are torn apart, the mass kneaded, mixed and pressed together, water removed and the volume reduced to about one-eighth of the original. Thus the peat gains a very compact form, and bears a close resemblance to the form of coal known as lignite, both in appearance and density. Peat thus dried is not hygroscopic, and only contains about 20 per cent water. The peat is sometimes used in this form. However, the general procedure is to vaporize the amount of water remaining by a slow heat and leave the briquettes in the hard, bone-dry state in which the greatest efficiency is shown.

Most of the machines knead the peat as it is taken from the bog, taking it up in an elevator or chain of buckets to the mixer, and then it is forced through a mouth piece, which gives it the shape of the briquette and chops them off at a given length. This block is called "compressed peat" (German presstorf). It is wet and soft, and is passed on in this state to a drying table or plain. The first of these machines was constructed in 1861, by C. Schlickeysen, in Berlin. The modern machines are all modifications of his design.

The Schlickeysen process may be described as follows: Raw turf comes up on an endless belt running in the long, sloping trough which leads to the peat bog (for these machines are generally portable, and set up on the edge of the bog and moved from one part to the other as the work of taking the peat out progresses). From the summit of the elevator the raw material drops into the machine, where it is kneaded, cut, torn and compressed into about two-thirds of its original bulk, and delivered at the mouth of the machine in blocks or bricks of any desired size or shape. These are first dried until they lose about two-thirds of their water, and then the drying is continued by artificial means until they are bone-dry. The principle upon which this machine is operated depends on the fact that peat is almost 80 per cent water in its original state, of which four-fifths is held in mechanical suspension between the hairy fibers, and the remaining fifth is contained in the capillary tube running through the hairy vegetable fibers of the peat. Nine cubic feet of raw peat is thus converted by this machine into 6 cubic feet of prepared peat which still contains 20 per cent water. This is further dried and compressed into 1 cubic foot of black, fossil-vegetable stone of about 1.5 specific gravity, which can be sawed, planed and even polished like cannel coal. Such briquettes contain about 66 per cent of inflammable matter, and approximately about 96 per cent the heat value of lignite.



SCHLICKEYSEN BRIQUETTE MACHINE.

The Stauber process was worked out at Charlottenburg in 1901. The briquette made by this process contains 45.14 per cent fixed carbon, 4.54 per cent hydrogen, 29.34 per cent oxygen, 9.09 per cent ash constituent. It has a thermal value of 3,806 calories. The process includes a method for the rapid drying of the moist peat by means of heated and compressed air within the closed chamber or channel communicating with conduits in such a manner, heater air can be forced out of the drying chamber and condenses it in the conduits, this greatly stimulating the process of evaporation by which the peat is dried. Peat is quickly and properly made into briquettes by this process. The machine consists of a cylindrical boiler with conduit pipes, and has a capacity of turning out 5 tons of completed fuel in a day. The briquette device is the same as the other general types of briquette machinery.

The Schoening-Fritz process consists in compressing dried peat between hot rollers, by which it is simultaneously carbonized and briquetted.

The present style of machine may be simple or very elaborate, some moving from place to place under their own power. Samples and pictures of all types of peat digging and briquetting machinery may be seen at the warerooms of Julius Bordollo, at Kingsbridge, N .Y.

One modification of the processes generally in use is to pump the peat in an almost liquid state from the bog through pipe lines to the place where the pressing and drying machinery is located. This method is greatly used in Sweden and Norway.

Any one of the above processes may be used for small factories, but the following process, known as the Ziegler process, will be found the most suitable and economical for the large works. The Ziegler process was invented by Martin Ziegler in 1892, and was the outcome of many years of hard study and research:

Concisely stated, the Ziegler process consists in carbonizing the peat in closed ovens, heated by burning the gases generated in the coking process itself. Such a plant is, therefore, self-supporting, the only fuel required being coal or wood sufficient to start the ovens for the first charge, when the gases generated by the coking process becomes available and enable the operator to continue the production indefinitely. Not only this, but the off-heat from the retort is passed on to the drying chambers, and serves to dry the wet peat as it is taken from the bogs. This feature presents the most economical way of preparing the wet peat for carbonization.

By the Ziegler process, 1 ton of peat will produce 350 kilograms of peat-coke, 40 kilograms of tar, 6 kilograms of methyl alcohol, 4 kilograms of ammonium sulphates, 6 kilograms of acetate of lime. Hence, from the present market reports we can see that the by-products will pay for the process of getting out the peat-coke and leave a fair margin of profit. The Ziegler process is in great use in Germany and Russia. In Germany, at Oldenburg, there has been a five-oven plant working since 1893, and showing a large profit. At Redkino, in Russia, the government has a station, which they are increasing all the time, with the idea in view of supplying all the locomotives with coke instead of wood and coal to burn. This has been very successful, and the work is very interesting from the steam engineer's end.

In Western Norway, near Bergen, a peat-coking plant has been erected built on different lines. This plant is furnished with electricity, and the electricity takes the place of the fire. This method gives a coke which is even superior to that made by the Ziegler process.

The final and best way for the utilization of peat as power is that of transforming it into gas and using this generator gas in the gas motors or engines of to-day. This process consists in placing air-dried, machine-treated peat in gas generators, preferably such as are described in the *Zeitschrift für Electrotechnik* (Vienna) of Sept., 1905, by Arnold Zlamal, or any of the retorts in use here for making producer-gas. The gas thus generated is stored in tanks or gasometers and piped to the engines, sometimes for long distances. This is by far the most economical method of utilizing a low-cost fuel yet placed before the public, and will become of more general use as the years go by.

The cost of producing peat-fuel is very low, in all the cases mentioned the cost not exceeding $2 per ton delivered. In such processes as the Ziegler, the initial cost is greater, but the sale of by-products reduces the cost of the fuel to a minimum, and in time the profit side appears and pays for the initial expense. This has been proven in the case of the Oldenburg plant. One man can dig with a peat digger about 5 tons of raw peat in an 8-hour day. Therefore, any ten men could dig a sufficient amount to run a plant for some few days.. The saving in any plant should amount in one year's time to the cost of the machinery required to make the peat fuel.

It should be remembered that all analyses go to show that any portion of our country contains bogs of higher thermal value than any of those existing in Europe.

The Microstructure of Silicon and Alloys Containing Silicon.[1]

By A. B. Albro.

Introduction.

This paper is written, not to propose or to defend any theories, but to show photographically recorded facts.

Enough discordant theories regarding the effects of silicon upon metals have already been advanced. For example:

(*a*) Von Jonstorff[2] claims that increasing the silicon in steel increases its brittleness, while Hadfield[3] states that 2 per cent of silicon with about 1 per cent of carbon increases the toughness of the steel considerably.

(*b*) Carnot and Goutal[4] state that silicon usually exists free in cast iron, and that any silicide formed in the furnace disappears on cooling the iron, and in proof of their claims quote the results of Le Chatelier's[5] experiments on the electrical resistance of steel and iron containing silicon, while Stead[6] states that silicide of iron dissolved in iron is one of the constituents of the phosphoric eutectic in gray silicious pig irons.

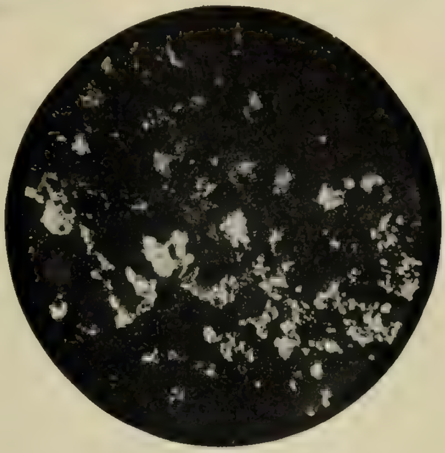


FIG. 1.—GRAPHITOIDAL SILICON × 23.

Others might be quoted regarding the effect of silicon on the properties of the phosphoric eutectic, the melting point and fluidity of cast iron and steel, and various other properties of iron and its alloys, but the above serve to show the considerable disagreement among authorities.

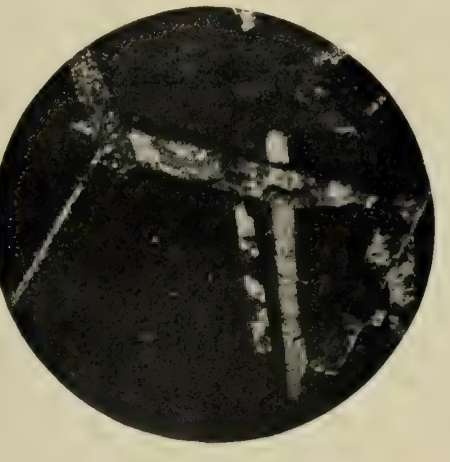


FIG. 2.—SILICON CRYSTALS × 34.

In contrasts with this, the belief that silicon reduces the percentage of iron carbide in pig iron is universally held. Moissan's[7] concise statement of this phenomenon is as follows: "* * * * silicon displaces carbon in iron and iron carbide. These bodies when maintained at sufficient temperatures behave exactly as the aqueous solution of certain compounds in which

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[1] A paper presented before the American Electrochemical Society, Boston meeting .

[2] Report presented at Third International Congress of Chemists at Vienna. Stahl und Eisen, March 7 and 15, 1899.

[3] Reprint of paper read at Institute of Civil Engineers. The Metallographist, vol. 1, pages 135, 136.

[4] Annales des Mines, October, 1900.

[5] Translation in American Manufacturers, Feb. 12, 1903, from Comptes Rendus.

[6] Journal Iron and Steel Institute, 1900, No. 11.

[7] Moissan, Electric Furnace, Eng. edition, page 69.

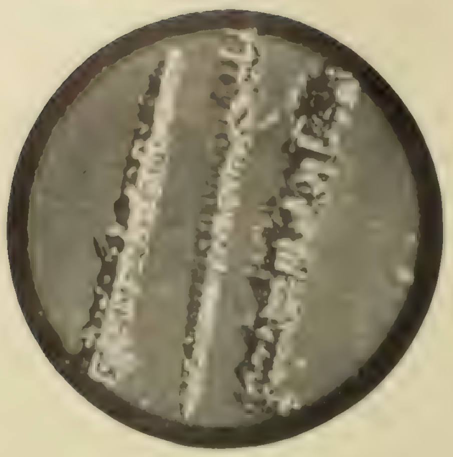

FIG. 3.—SILICON CRYSTALS × 23.

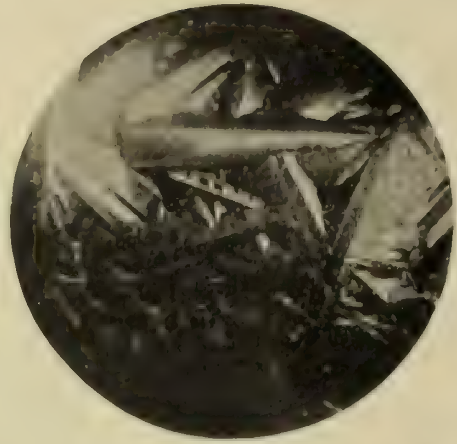

FIG. 4.—SILICON SURFACE CRYSTALS × 8.

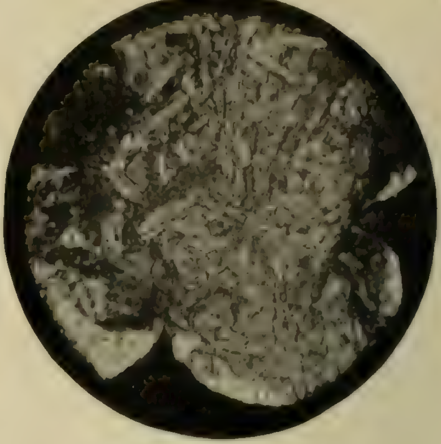

FIG. 5.—SILICON NODULE × 5.

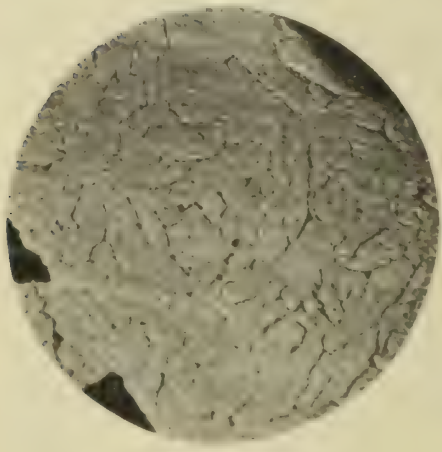

FIG. 6.—SILICON NODULE × 8.

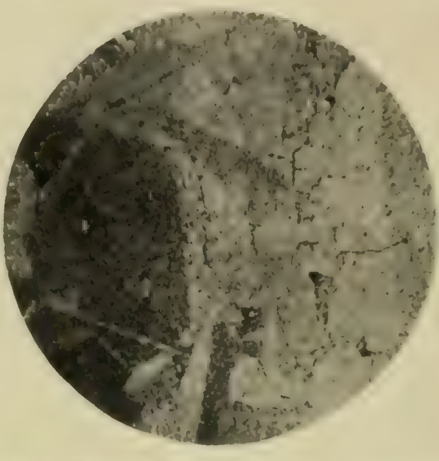

FIG. 7.—SILICON NODULE, VERTICAL SECTION × 8.

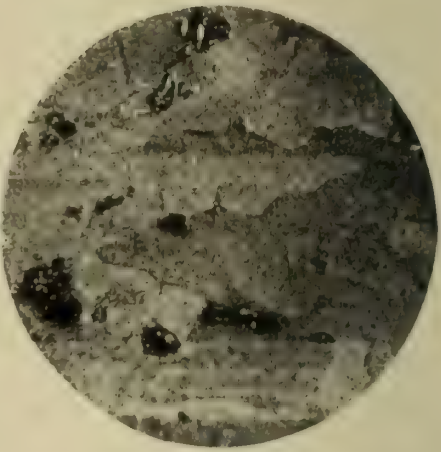

FIG. 8.—SILICON NODULE, HORIZONTAL SECTION × 8.

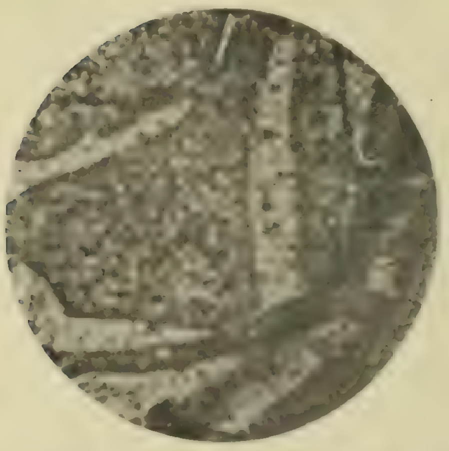

FIG. 9.—[illegible] × 8.

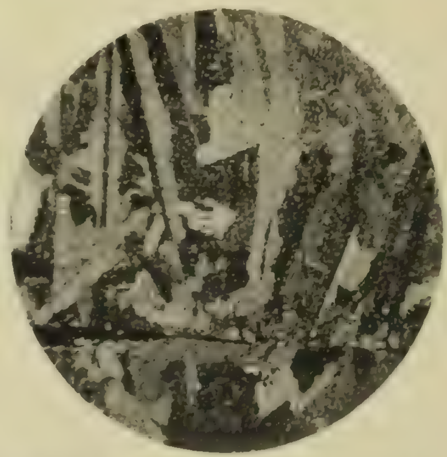

FIG. 10.—"A" SILICON, VERTICAL SECTION × 8.

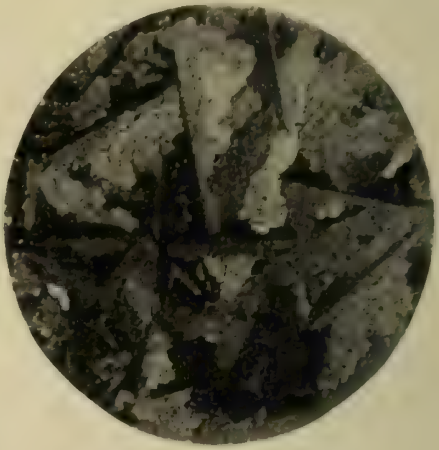

FIG. 11.—"A" SILICON, HORIZONTAL SECTION × 8.

we precipitate or displace substances in solution or combination. If the displacement of the carbon is not entirely complete a state of equilibrium is established between the silicide and carbide of iron, an equilibrium in which the conditions vary with the temperature and with the impurities which are contained in the bath. This is generally the case with white or mottled iron."

Most investigators agree that the presence of silicon lessens the liability of blow-holes, and thus promotes soundness in cast iron and steel, but they differ greatly as to how this result is attained.

Undoubtedly some of the diversity of opinion concerning the effects of silicon is due to certain observers attributing to it the bad qualities of its oxy-compounds.

Iron containing up to about 14 per cent silicon can be produced in blast furnaces.

The electric furnace is capable, under proper conditions and expert manipulation, of producing ferro-silicon of any percentage composition, and recently silicon, practically free from iron, has been produced in commercial quantities by the reduction of silica by calcium carbide or by carbon.

In Europe the process of Scheid is in operation, while in this country the work of Mr. Tone has recently attracted considerable attention.

A novel silicon process has been devised for Mr. George Westinghouse by Mr. Henry Noel Potter, under whose direction these investigations were made.

Silicon.

Fig. 1 shows graphitoidal silicon magnified 23 diameters. This sample was furnished by Mr. Frederic S. Hyde, and was from the original material made by him and described in his article on graphitoidal silicon.[8] To the unaided eye it has the appearance of coarse graphite flakes, and in Fig. 1 it is seen to consist of irregular shaped masses, some of which have plane surfaces.

The silicon needle-like crystals shown in Fig. 2 were also furnished by Mr. Hyde, and are here magnified 34 diameters. These needles were made by solution and crystallization in molten zinc. The actual length of these crystals is about 1 millimeter.

Fig. 3 shows three complex crystals of pure silicon, magnification 23 diameters, made by dissolving pure silicon in molten zinc. The pure silicon was obtained from crystalline silicon which had been very finely powdered and then purified and washed. Only the particles which remained in suspension for 48 hours were used. The resulting powder analyzed 99.8 per cent silicon, gave absolutely no gritty sensation when placed between the teeth, and showed very few crystals under a magnification of 205 diameters.

These complex crystals resemble very much the dendrites of alpha iron shown and described by Osmond and Cartaud.[9] Their actual length was from 1.5 to 2.5 millimeters. The same zinc solution also contained single octahedra of silicon.

When fused in the electric arc furnace, silicon collects in the fluid phase, and in its subsequent solidification extrudes some silicon in the form of worm-like nodules. The surface of these nodules exhibits fern-like crystals, and Fig. 4 shows them magnified 8 diameters. Single crystals have been observed which measured 15 by 50 millimeters.

Before passing to the internal structure of silicon, a word regarding the method of preparing sections for examination is in order. Samples are ground on a carborundum wheel until the desired section is reached, and then ground to a plane surface on a glass plate with "FFF" carborundum powder and water. The last stages of preparation are like those of ordinary steel sections. Etching is done with dilute mixed hydrofluoric and nitric acids in the case of all silicon sections, but the etching acid differs with alloy sections, and will be stated for each one.

In the case of silicon there is a difference between the appearance of a section cut vertically and one cut horizontally through a mass of nodule, and it is, therefore, necessary to know how the mass lay while cooling and to state which sections are vertical and which horizontal.

All photomicrographs having a magnification less than 167 diameters were taken with oblique light, the higher enlargements with vertical illumination. The source of light was a single-glower, 50-candle-power Nernst lamp, with globe removed. This gives a very even illumination of such intensity that with a 1-6 inch objective and 2-inch eyepiece, the exposure necessary with Seed "L" orthochromatic plates ranges from 10 to 60 seconds.

Fig. 5 shows a vertical section of a nodule strongly etched and under a magnification of 5 diameters.

Fig. 6 shows a horizontal section of a nodule strongly etched, and then repolished to bring out only the division lines between individual grains. This shows clearly where impurities at certain places between the grains have been etched away. Magnification 8 diameters.

Fig. 7 shows a slightly etched vertical section of a similar nodule, and Fig. 8 a horizontal section under the same treatment. The dark areas in No. 8 are actual blow-holes, and no nodule was found whose central horizontal plane was free from them. Magnification in both cases 8 diameters.

A peculiar formation noticed in one case under a magnification of 89 diameters is shown in Fig. 9. It is a slightly etched vertical section of a nodule which had been twisted once around its horizontal axis during the formative period.

Owing to the fact that the nodules are under stress during freezing, the crystalline structure is probably distorted, and we should expect to find an undistorted structure only in sections taken from an unstrained mass in the interior of a large block of material.

The next few figures show such interior sections under different magnifications. They are from three samples of silicon called A, B and C.

Sample A was from a 500-pound lot purchased in France in March, 1904, and contains 92.6 per cent silicon, the balance being mainly calcium, iron and aluminium.

Sample B was very kindly furnished by Mr. Frank J. Tone, of the Carborundum Co., of Niagara Falls.

Sample C was made by Mr. Potter, and contains 98.8 per cent silicon, the balance being iron and aluminium.

The next two photomicrographs show sample A magnified 8 diameters. Fig. 10 is a vertical section showing the junction of two masses partially fused together. Long, irregularly overlapping and interlocking grains are seen, many of which seem to have two faces shown in this section.

In Fig. 11, which is a horizontal section, these apparently two-faced grains are shown more clearly, and several triangles are formed by them. In this appearance the section resembles, in a striking degree, the fern-leaf surface crystals shown in Fig. 6. In addition to these, several of the interlocking grains can be seen, but very much shorter than in the vertical plane.

This difference in length of grains viewed vertically and horizontally tends to explain why large masses of silicon fracture much more easily in the plane vertical to the surface uppermost during solidification. It is practically impossible to produce two parallel fractures in the horizontal plane, while this can be readily done in the vertical plane.

Fig. 12 shows a horizontal section of sample B, magnified 8 diameters. This is seen to consist of two large, irregular-shaped grains, together with several smaller ones. In the neighborhood of the smaller grains, silicon carbide crystals are seen.

Sample C, next shown (Fig. 13, 8 diameters), is characterized by its large grains, it being impossible to include a complete grain in the field, 7.6 mm. in diameter, covered by this photograph. The interlocking of the grains is very clearly shown, as well as their homogeneity and the absence of any impurities between them.

The next three slides show horizontal sections magnified 34 diameters.

No. 14 is sample A, and shows the interlocking grains and impurity pits. No. 15 is sample B, and shows a smooth, homogeneous field with the exception of the three areas containing silicon carbide. No area could be found which was free from this compound. In No. 16, which is sample C, a remarkable interlocking of the two dark grains with the H-shaped light green is shown. The field shows such a homogeneity that it can safely be called a photomicrograph of absolutely pure silicon.

The ultimate structure is approached in the next three figures, in which the three samples are shown magnified 167 diameters.

Fig. 17, sample A, shows a very close-grained surface with some octahedra in various portions, but the structure has not the clearness that purer silicon possesses.

In Fig. 18 we have sample B, and this shows a field of pure silicon, with one area containing silicon carbide. Individual crystals can be distinguished here, together with numerous striated areas.

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[8] Journal Am. Chem. Soc., vol. 21, No. 8, August, 1899.

[9] Annales des Mines, August, 1900.

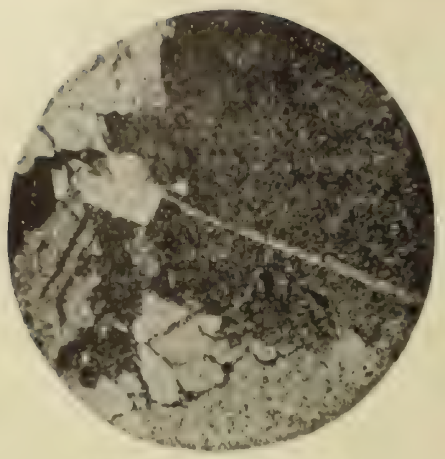

FIG. 12—"B" SILICON × 8

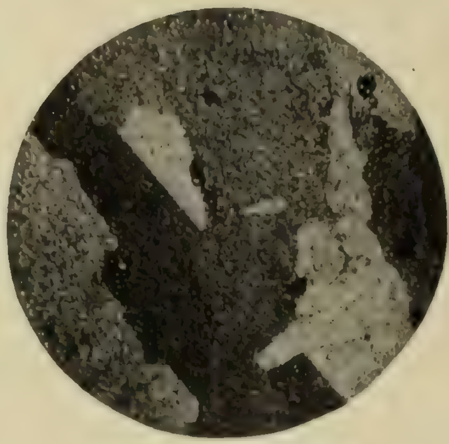

FIG. 13—"C" SILICON × 8.

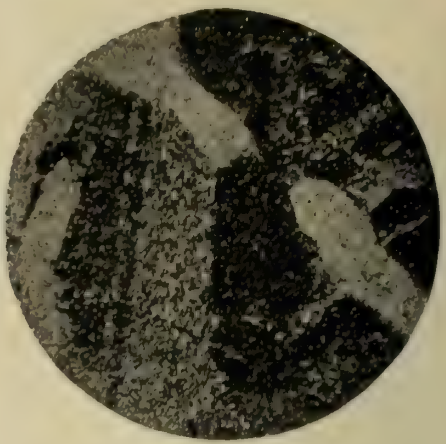

FIG. 14—"A" SILICON × 34.

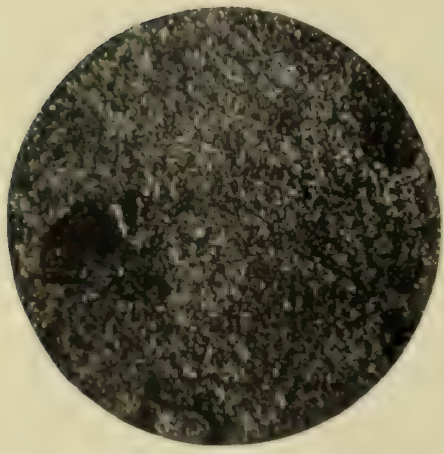

FIG. 15.—"B" SILICON × 34.

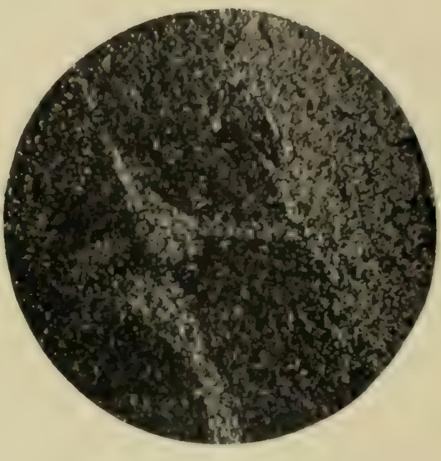

FIG. 16.—"C" SILICON × 34.

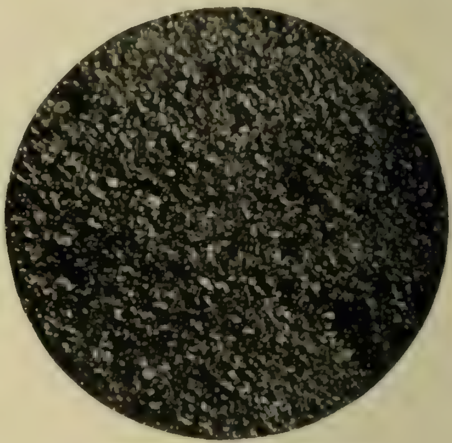

FIG. 17.—"A" SILICON V × 167.

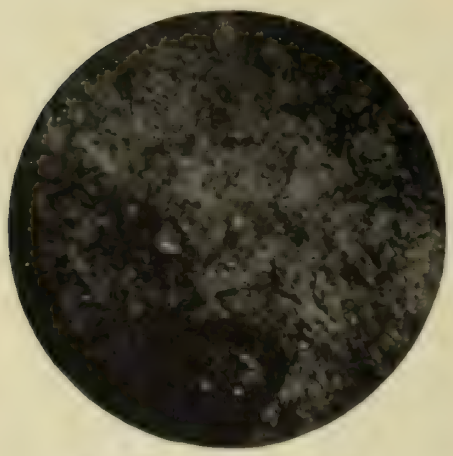

FIG. 18—"B" SILICON V × 167

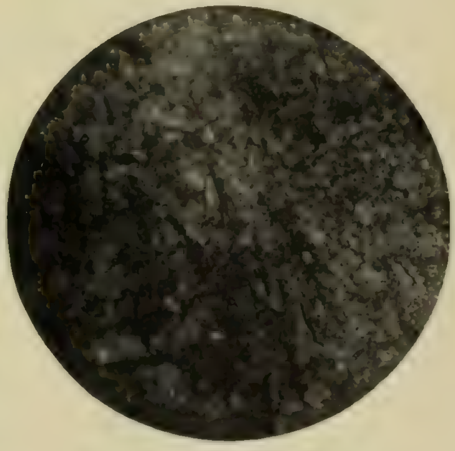

FIG. 19—"C" SILICON V × 167

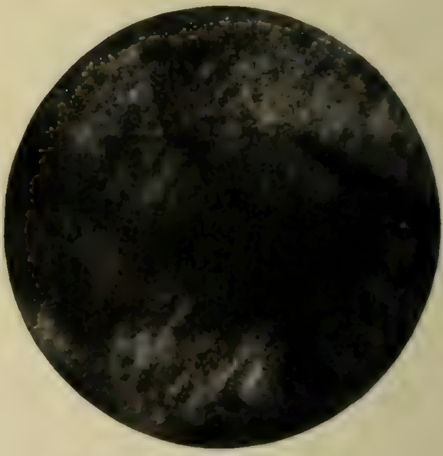

FIG. 20—"C" SILICON V × 1025.

Sample C, as shown in Fig. 19, shows the structure more clearly than No. 18, and at the same time is free from any impurities. Numerous octahedra and clearly striated areas are visible.

With this latter section under a magnification of 1,025 diameters, as shown in Fig. 20, we have the characteristic [illegible]

Despite the slight lack of clearness, octahedra are clearly shown which are striated on the surface, the striations overlapping each other like shingles on a house, the same structure, as Stead observed in his sample of steel, containing 4 per cent of silicon.[10] The structure of silicon does not appear to be affected by any temperature below its freezing point, as the grains and crystals have the same appearance when quenched at a bright red heat as when the cooling occupies 24 hours.

This may explain why the structure of steels containing 4 per cent of silicon is unaffected by heating to 1,100° C., as stated by Stead[11] and others.

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[10] Reprint of paper read before Iron and Steel Institute, May, 1898. The Metallographist, vol. 1, pages 327, 328.

[11] Iron and Steel Institute, May, 1898.

(To be concluded.)

Metallurgical Calculations.—IX.

By J. W. Richards, Ph. D.
Professor of Metallurgy in Lehigh University.

ARTIFICIAL FURNACE GAS.

There are many different forms of producers for making artificial furnace gas. For the purposes of making calculations upon them they may be conveniently divided into four classes, as follows:

1. Simple producers, those which use ordinary fuels, such as wood, peat, lignite, bituminous coal or anthracite, and in which no water or water vapor is introduced other than the water in the fuel itself and the normal moisture of the air used.

2. Mixed gas producers, in which water vapor or steam is introduced with the air for combustion, in such amount as to be entirely decomposed in passing through the fuel.

3. Mond gas producers, in which, for a special purpose, more steam is introduced than can be decomposed in the producer, thus producing very wet gas.

4. Water gas producers, in which air and steam alone are alternately fed to the producer, the former for heating up, the latter for producing water gas.

As far as the calculations are concerned, the essential difference between these classes is the varying amount of water vapor or steam introduced under the fuel bed while producing the gas, from all air in Class 1 to all steam in Class 4.

The calculations which it is of immediate interest to make, and the results of which are of immediate value to the metallurgist, are those concerned with the volume of gas produced per unit of fuel, its calorific power compared to that of the fuel from which it is produced, the items of the heat losses during the operation of transforming the solid fuel into gaseous fuel, the function of steam in the producer, the limits up to which the use of steam is permissible, the increase of efficiency of the gas by subsequently drying it, the advantages as to final efficiency which are gained by gasifying the fuel over burning solid fuel directly.

For information as to the construction and operation of gas producers, reference may be made to treatises such as Sexton's "Fuel," Sexton's "Producer Gas," Groves and Thorp's "Chemical Technology, Vol. V., Fuels." Trade pamphlets and catalogues, such as those of R. D. Wood & Co., of the De la Vergne Machine Co., of the Wellman-Seaver-Morgan Co., the Colorado Iron Works Co., the Weber Gas & Gasoline Engine Co., the Morgan Construction Co., etc., contain a great deal of exact and useful information, and may be usually had for the asking. The monograph of Jüptner and Toldt on "Generatoren und Martinöfen" (Felix, Leipzig, 1900), is concerned wholly with calorimetric calculations concerning the production of gas and its utilization in regenerative gas furnaces.

1.—Simple Producers.

In these a deep bed of fuel is burnt by air or fan blast, introducing no more moisture than happens to be in the atmosphere at the time being. The fuel fed into the producer is first dried by the hot gases, then is heated and distilled or coked, and finally is oxidized by the incoming air. The residue is the ash of the coal, which is ground out at the bottom or drops through the grate, containing more or less unburnt fixed carbon. Great loss of efficiency sometimes occurs from the ashes being rich in carbon. The escaping gases issue at temperatures of 300° up to 1000° C., carrying much sensible heat out of the producer.

Calculations as to the amount of gas produced per unit of fuel consumed are to be based entirely on the carbon. The gas must be carefully analyzed, so that it can be calculated from this analysis how much carbon, in weight, is contained in a given volume of gas. (An alternative method is to take a carefully measured volume of the gas, mix it with excess of oxygen, and explode in a gas burette, determining the amount of carbon dioxide formed, and from that calculate the weight of carbon in the volume of gas taken.) Knowing this, the rest is simple: the carbon in unit weight of fuel minus the carbon lost in the ashes by poor combustion, gives the weight of carbon gasified; this divided by the weight of carbon in unit volume of gas produced, gives the volume of the latter per unit weight of fuel.

Illustration: A fuel used in a gas producer contains 12 per cent of *ash* and 72 per cent of carbon. The *ashes* made contain 20 per cent of unburnt carbon; the gas produced contains by analysis and calculation 0.162 ounce of carbon per cubic foot of gas, measured at 60° F. and 29.8 inches barometric pressure. What volume of gas, measured at above conditions, is being produced per ton of 2240 pounds of coal used?

Solution: The distinction between *ash* and *ashes* must be noted; the former is the analytical expression for the amount of inorganic material left after complete combustion during the chemical analysis; the latter term means the waste matter produced industrially, and consists, if weighed dry, of the true ash, plus any unburnt carbon. The calculations are therefore as follows:

		Lbs.
Ash in 2240 pounds of coal = 2240 × 0.12	=	268.8
Ashes corresponding = 268.8 ÷ 0.80	=	336.0
(the ashes are 80 per cent ash)		
Carbon in the ashes	=	67.2
Carbon in the coal = 2240 × 0.72	=	1612.8
Carbon going into the gas (gasified)	=	1545.6
Carbon in 1 cubic foot of gas = 0.162 ÷ 16	=	0.010125
Volume of gas produced per 2240 pounds of coal = 1545.6 ÷ 0.010125	=	152,652 cu. ft.

It will be noted that these calculations absolutely require the percentage of *total* carbon in the fuel, as determined by chemical analysis. This is not a difficult analysis, as it consists in burning the carbon in a heated tube in a stream of oxygen or air free from carbon dioxide; the products of combustion are dried and then passed through caustic potash solution to absorb CO^2 gas, the weight of which is obtained by the increased weight of the potash bulb, and the total carbon thus obtained. The fixed carbon and volatile matter of the coal, as determined by the ordinary proximate analysis, cannot be used in this calculation, since while all the fixed carbon is carbon, the volatile matter is of variable composition, containing such varying proportions of carbon that no fixed percentage of the latter in it can be assumed without considerable possible error.

The calorific power of the gas, per cubic meter or cubic foot, can be calculated from its analysis, using the calorific powers of the combustible constituents as already given in our tables. This, multiplied by the volume of gas produced per unit of coal, gives the calorific power of the gas as compared with that of the coal from which it is made. The difference is the heat loss in the operation of producing the gas, including loss by unburnt carbon in the ashes. In fact, we may state that the heat balance is based on the following equations:

Heating power of the coal, per unit
— Heating power of the gas per unit of coal
= Calorific losses in conversion.

The latter item is composed of:
Loss by unburnt carbon in the ashes.
Sensible heat of the hot gases issuing.
Heat conducted to the ground.
Heat radiated to the air.

These items may be modified as follows: If the air used is hotter than the normal outside temperature, its sensible heat above this datum should be added to the heating power of the coal, because it increases the total available heat. If the ashes are removed hot, and not allowed to be completely cooled by the incoming air, their sensible heat should be included in the of the losses during conversion. If the air used is moist, its moisture will be decomposed to hydrogen and oxygen, but the heat absorbed in doing this is exactly represented by the calorific power of this increased amount of hydrogen in the gases, and the heat absorbed is not lost but really represents so much saved as available calorific power of the gases. This item must, therefore, *not* be counted as one of the heat losses during the operation, as those losses have been defined by us. If the fuel is wet, considerable heat is required to evaporate the moisture in it, but this heat is *not* to be reckoned as one of the losses in conversion, if we have taken as the heating power of the coal the practical metallurgical value; that is, its value assuming all the water in its products of combustion to remain as vapor and none to condense. If this value has been so taken, the heat required to vaporize the moisture in the coal will have already been allowed for. Similarly, it may take a little heat energy to break up a bituminous coal so as to expel its volatile matter, but this should not be reckoned in as a heat loss in the producer, because a little reflection will show that, whatever this amount may be, it has been properly allowed for in the determination or calculation of the total calorific power of the fuel. Jüptner and Toldt call this the "gasifying heat," and use it in all their calculations, but it is doubtful whether it really amounts to an appreciable quantity, for one thing, and even if it does it should not be reckoned as a heat loss in the producer.

Problem 14.

Jüptner and Toldt ran a gas producer with lignite of the following composition: (*Generatoren*, p. 49.)

Carbon	69.83 per cent
Hydrogen	4.33 "
Nitrogen	0.50 "
Oxygen	12.38 "
Moisture	7.25 "
Ash	5.71 "

Of this coal, 3214 kilograms was used in 8 hours, 50 minutes, producing gas which contained, analyzed dry, by volume:

Carbon dioxide, CO^2	5.21 per cent
Carbon monoxide, CO	23.99 "
Oxygen, O^2	0.63 "
Methane, CH^4	0.25 "
Hydrogen, H^2	10.64 "
Nitrogen, N^2	59.28 "

The ashes produced weighed 22.23 kilograms per 100 kilograms of coal used, and contained 68.76 per cent of unburned carbon. The calorific power of the coal, determined in the calorimetric bomb (in compressed oxygen, moisture resulting condensed) was 6949 Calories per gramme. Temperature of hot gases, 282° C.; temperature of air used, 9° C.; humidity, 76 per cent; barometer, 712 millimeters of mercury.

Required:

1. The volume of gas, measured at 0° and 760 mm. pressure (and assumed dry), produced per metric ton (1000 kilos. = 2204 pounds) of fuel used.
2. The calorific power of the coal, per kilogram, with moisture formed by its combustion assumed uncondensed.
3. The proportion of the calorific power of the coal developable by burning the gas produced from it.
4. The loss of heat in conversion.
5. The loss of heat by unburnt carbon in the ashes.
6. The loss of heat as sensible heat in the gases.
7. The loss of heat by radiation and conduction, expressed:
   (*a*) Per unit of coal burnt.
   (*b*) Per minute.
8. The volume of air required by the producer, at the conditions of the atmosphere, per kilogram of coal burnt.

Solution:

(1) The gas contains, per cubic meter at standard conditions:

$$\begin{array}{ll} \text{Carbon in } CO^2 & 0.0521 \times 0.54 \text{ kilos.} \\ \text{Carbon in } CO & 0.2399 \times 0.54 \quad " \\ \text{Carbon in } CH^4 & 0.0025 \times 0.54 \quad " \\ \hline \text{Total} & 0.2945 \times 0.54 = 0.1590 \text{ kilos.} \end{array}$$

The carbon gasified from 1000 kilograms of coal is:

$$\begin{array}{lr} \text{Carbon in coal} & = 698.3 \text{ kilos.} \\ \text{Carbon in ashes } 222.3 \times 0.6876 & = 152.8 \quad " \\ \hline \text{Carbon gasified} & = 545.5 \quad " \end{array}$$

Therefore,

$$\text{Gas (dry) produced} = \frac{545.5}{0.1590} = 3430 \text{ cubic meters.} \qquad (1)$$

(2) The calorific power of the coal as given must be diminished by the heat required to vaporize all the moisture formed by its combustion, leaving such moisture as theoretical moisture at 0° C. There will be formed per kilogram of coal:

$$\begin{array}{lr} \text{From moisture of coal} & = 0.0725 \text{ kilos.} \\ \text{From hydrogen } 0.0433 \times 9 & = 0.3897 \quad " \\ \hline \text{Total} & = 0.4622 \quad " \end{array}$$

To evaporate this to theoretical moisture at zero (thus putting the water vapor on the same footing as the other products of combustion, CO^2 and N^2) requires:

$$0.4622 \times 606.5 \text{ (Regnault)} = 280 \text{ Calories.}$$

leaving as the metallurgical or practical calorific power

$$6949 - 280 = 6669 \text{ Calories.} \qquad (2)$$

(3) The calorific power of each cubic meter of gas (measured dry at standard conditions) is

$$\begin{array}{lr} CO = 0.2399 \text{ m}^3 \times 3{,}062 = & 734.6 \text{ Calories.} \\ CH^4 = 0.0025 \text{ m}^3 \times 8{,}598 = & 21.5 \quad " \\ H^2 = 0.1064 \text{ m}^3 \times 2{,}613 = & 278.0 \quad " \\ \hline \text{Total} = & 1034.1 \quad " \end{array}$$

Calorific power of gas from 1 kilogram of coal:

$$1034.1 \times 3.43 = 3547.0 \text{ Calories.}$$

$$\text{which equals } \frac{3547.0}{6669.0} = 53.2 \text{ per cent.} \qquad (3)$$

(4) The loss of calorific power in conversion is 100 — 53.2 = 46.8 per cent of the calorific power of the coal, or per kilogram of coal:

$$6669 - 3547 = 3122 \text{ Calories.} \qquad (4)$$

(5)

$$\text{Carbon in ashes} = 0.1528 \times 8100 = 1237.7 \text{ Cal.}$$
$$= 18.6 \text{ per cent.} \qquad (5)$$

(6) The gases produced carry off, per cubic meter measured dry, the following amounts of heat:

Volume × mean specific heat (0° — 282°) = heat capacity per 1

$$\begin{array}{ll} CO^2 & 0.0521 \times 0.432 = 0.0225 \\ CH^4 & 0.0025 \times 0.428 = 0.0011 \\ \left.\begin{array}{l} CO \\ O^2 \\ H^2 \\ N^2 \end{array}\right\} & 0.9454 \times 0.311 = 0.2940 \\ \hline & \text{Sum} = 0.3176 \end{array}$$

Heat carried out = 0.3176 × 282 = 89.56 Calories.

Per kilogram of coal = 89.56 × 3.43 = 307 Calories.

$$\text{Proportion of calorific power} = \frac{307}{6669} = 4.6 \text{ per cent.}$$

The above result is, however, subject to a small correction, because some of the moisture in the coal goes undecomposed into the gases, and is not represented in the analysis of the dried gas. The amount of this moisture can be obtained with sufficient accuracy by finding how much moisture would be obtained by burning the dried gas from 1 kilogram of coal, and comparing this with the moisture which would be obtained from 1 kilogram of coal itself; the difference must represent the moisture accompanying the gas as water vapor, and which has not been included in the above computation.

Burning 1 cubic meter of gas, the H^2O vapor is:

From CH^4 0.0025 × 2 = 0.0050 cubic meters.
From H^2 0.1064 × 1 = 0.1064 " "

0.1114 " "
Per kilo. of coal = 0.1114 × 3.43 = 0.3821 " "

But the weight of water vapor from burning 1 kilogram of coal has already been found to be (2) 0.4622 kilograms, the volume of which is

$$0.4622 \div 0.81 = 0.5706 \text{ cubic meters.}$$

Leaving, therefore, 0.5706 — 0.3821 = 0.1885 cubic meters of water vapor as such accompanying the 3.43 cubic meters of (dried) gas from 1 kilogram of coal. This would take out

$$0.1885 \times 0.382 \times 282 = 20.3 \text{ Calories.}$$

Thus increasing the sensible heat in the gases to

$$307 + 20.3 = 327.3 \text{ Calories} = 4.9 \text{ per cent.} \qquad (6)$$

[In reality, a still further correction should be made; viz.: to add in the moisture in the air used, because it would also reappear as moisture on final combustion of the gases. Its amount is found from the amount of air used, which, if 76 per cent saturated at 9° would carry moisture having 0.76 × 7 mm. (if saturated) = 5.3 millimeters tension, which represents, barometer being 712 mm., 0.7 per cent of the volume of the air used, or practically 0.9 per cent of the volume of nitrogen in the air. Since the nitrogen in the gas represents almost entirely the nitrogen in the air used, the moisture to be accounted for from the air amounts to 0.5928 × 0.009 = 0.0053 cubic meters per cubic meter of gas, or = 0.0182 cubic meters per kilogram of coal burnt. This correction is altogether too small to affect the results in this case, but should be taken into account whenever the air used is warm and moist.]

(7) The calorific loss in conversion was 3122 Calories. Of this we have accounted for:

Lost by unburnt carbon in ashes	1237.7	Calories.
Sensible heat of gases (including moisture)	327.3	"
Total	1565.0	"
Loss by radiation and conduction	= 1557.0	" (a)

Per 8 hours 50 minutes there is burnt 3214 kilograms of fuel, making the loss of heat by radiation and conduction per minute =

$$\frac{1557 \times 3214}{530} = 9{,}442 \text{ Calories.} \qquad (b)$$

(8) At the conditions given, each cubic meter of moist air used contained 0.7 per cent of its volume of moisture, making its percentage composition by volume:

Water vapor	0.70	per cent.
Air: Oxygen	20.65	"
Air: Nitrogen	79.65	"

The volume of gas produced per kilogram of coal is 3.43 cubic meters, of which 59.28 per cent is nitrogen, equal to 2.0333 cubic meters, and weighing 2.0333 × 1.26 = 2.562 kilograms. Of this 0.0050 kilograms came from the coal itself, leaving 2.512 kilograms to come from the air, or 2.512 ÷ 1.26 = 1.9921 cubic meters. This would correspond to 1.9921 ÷ 0.7965 = 2.5011 cubic meters of moist air if measured at standard conditions. At 9° and 712 mm. pressure the real volume of moist air used at prevailing conditions, per kilogram of coal burnt, is

$$2.5011 \times \frac{273 + 9}{273} \times \frac{760}{712} = 2.76 \text{ cubic meters.} \qquad (8)$$

It must not be thought that the conditions of working in the above producer represent good practice; they are very poor practice as far as concerns the utilization of the fuel. Many producers make gas having 75 to 90 per cent of the calorific power of the coal from which it is made, so that the losses by unburnt carbon and radiation and conduction in this case must be regarded as highly abnormal and very poor practice. The writer chose this example for calculating, because of the carefulness with which Jüptner and Toldt had collected the necessary data, and because it illustrated so well the principles to be employed in similar calculations.

Problem 15.

A gas producer run in Sweden uses saw-dust of the following composition:

Water	27.0	per cent.
Ash	0.5	"
Carbon	37.0	"
Hydrogen	4.4	"
Oxygen	30.6	"
Nitrogen	0.5	"

Assume that it is run by dry air and that 0.5 per cent of ashes are made. The gas formed, dried before analysis, contains, by volume:

Carbon dioxide, CO^2	6.0	per cent.
Carbon monoxide, CO	29.8	"
Ethylene, C^2H^4	0.3	"
Methane, CH^4	6.9	"
Hydrogen, H^2	6.5	"
Nitrogen, N^2	50.5	"

The gas actually produced is partly dried before use by having its temperature reduced by cold water, in a surface condenser, to 29° C., in order to increase its calorific intensity of combustion.

Required:

(1) The proportion of the moisture in the moist gas which is condensed out.

(2) The calorific intensity of the moist gas, if burned preheated to 800° C. by the theoretical quantity of air preheated also to 800° C.

(3) The calorific intensity of the dried gas, burnt under exactly similar conditions.

Solution:

(1) It is first necessary to find the weight or volume of water vapor accompanying the gas before condensation, next that accompanying it after passing the condenser. The first can be calculated best on the basis of the hydrogen present in the fuel and in the (dried) gas made from it; the difference is the hydrogen of the moisture removed before analysis, *i. e.*, the hydrogen of the moisture in the wet gas.

The first step is to find the volume of gas (dry) produced per unit of fuel, as follows:

Carbon in 1 kilo. of fuel = 0.370 kilos.
Carbon in 1 m^3 of gas = $CO^2 + CO + CH^4 + 2C^2H^4$ (0.060 + 0.298 + 0.069 + 0.006) × 0.54 = 0.2338 "

$$\text{Dry gas per kilo. of fuel} = \frac{0.370}{0.2338} = 1.5825 \text{ m}^3$$

The next step is to calculate the water which would be formed by the combustion of 1 kilogram of fuel:

Water present in fuel	0.270	kilos.
Water produced by hydrogen	0.396	"
Total =	0.666	"

$$\text{Volume at standard conditions} \quad \frac{0.666}{0.810} \quad 0.8222\ m^3$$

From this we subtract the moisture which would be produced by the combustion of the 1.5825 cubic meters of dry gas, obtained as follows:

Water from ethylene $= 0.003 \times 2 \times 1.5825\ m^3$
Water from methane $= 0.069 \times 2 \times 1.5825$
Water from hydrogen $= 0.065 \times 1 \times 1.5825$

$= 0.209 \quad \times 1.5825 = 0.3307\ m^3$

Difference = water vapor to 1.5825 m^3 of dried gas
$= 0.8222 - 0.3307 = 0.4915\ m^3$
$= 0.4915 \div 1.5825 = 0.3106\ m^3$ per 1 m^3
of dried gas, as analyzed.

This is to be compared with the amount of moisture accompanying the same quantity of (dried) gas as it escapes from the condenser. This is obtained directly from the fact that the gas escaping will be saturated with moisture at 29° C., that the latter will, therefore, have a tension of 30 millimeters (tables), and that, assuming the barometer normal (760 mm.), the partial tensions of moisture and gas proper are as 30 to 760 — 30, or as 30 to 730. Since their respective volumes (if both were measured separately at normal pressures) are in the same proportion, it follows that each cubic meter of (dry) gas is accompanied by $30 \div 730 = 0.0411$ cubic meters of uncondensed moisture.

The respective quantities of moisture accompanying 1 cubic meter of dry, uncondensable gas, are, therefore, 0.3106 before cooling and 0.0411 after cooling, showing that 13.5 per cent of all the moisture escapes condensation, and that, therefore,

86.5 per cent of moisture is condensed. (1)

(2) The wet gas has a calorific power, calculating on the basis of 1 cubic meter of dried gas analyzed:

$CO \quad 0.298 \times 3{,}062 = 912.5$ Calories.
$C^2H^4 \quad 0.003 \times 14{,}480 = 43.4$ "
$CH^4 \quad 0.069 \times 8{,}598 = 593.3$ "
$H^2 \quad 0.065 \times 2{,}613 = 169.8$ "

Total = 1719.0 "

There is added to this available heat, when burned, the sensible heat in the gas itself, at 800° C., and also that of the necessary air, also at 800°.

Heat in gas:

$CO, H^2, N^2 = 0.630\ m^3 \times 0.3246 = 0.2045$ Cals. per 1°
$CO^2 = 0.060\ m^3 \times 0.5460 = 0.0328$ "
$CH^4 = 0.069\ m^3 \times 0.4485 = 0.0309$ "
$C^2H^4 = 0.003\ m^3 \times 0.50 = 0.0015$ "
$H^2O = 0.3106\ m^3 \times 0.460 = 0.1429$ "

Calorific Capacity = 0.4126 "

Total sensible heat $= 0.4126 \times 800 = 300.1$ Calories.

The air required theoretically is:

For $CO \quad 0.298\ m^3 = 0.1490\ m^3$ oxygen
For $C^2H^4 \quad 0.003\ m^3 = 0.0090\ m^3$ "
For $CH^4 \quad 0.069\ m^3 = 0.1380\ m^3$ "
For $H^2 \quad 0.065\ m^3 = 0.0325\ m^3$ "

Sum $= 0.3285\ m^3$ "
$= 1.58\ m^3$ air

Heat in this at 800° $= 1.58 \times 0.3246 \times 800 = 410.3$ Cals.

Sum total of heat going into the products:

Developed by combustion...........	1719.0	Cals.
Sensible heat in gas................	330.4	"
Sensible heat in air................	410.3	"
Total	2459.4	"

The products of the combustion are CO^2, H^2O and N^2, as follows: $CO^2 = 0.060$ (in gas) + 0.298 (from CO) + 0.006 (from C^2H^4) + 0.069 (from CH^4) = 0.433 m^3.

$H^2O = 0.311$ (with gas) + 0.006 (from C^2H^4) + 0.138 (from CH^4) + 0.065 (from H^2) = 0.520 m^3.

$N^2 = 0.505$ (in gas) + [1.58 — 0.33 = 1.25] (from air) = 1.755 m^3.

Since the 2459.4 Calories remains as sensible heat in the above products, at some temperature t, we have:

Heat capacity of the $CO^2 \quad = 0.433\ (0.37 + 0.00022t)$
Heat capacity of the $H^2O \quad = 0.520\ (0.34 + 0.00015t)$
Heat capacity of the $N^2 \quad = 1.755\ (0.303 + 0.000027t)$

Heat capacity of the products $= \quad 0.8688 + 0.00022065t$

and the calorific intensity t must be

$$t = \frac{2459.4}{0.8688 + 0.00022065t}$$

whence

$$t = 1907^\circ \qquad (2)$$

(3) When the dried gas is burned under similar conditions, the only difference is that 0.2695 m^3 of water vapor are absent from the gas and from the products, having been condensed. This reduces the available heat by the sensible heat in this much water vapor at 800°, viz.:

$$0.2695 \times 0.460 \times 800 = 99.2 \text{ Calories,}$$

and decreases the calorific capacity of the products by

$$0.2695\ (0.34 + 0.00015t).$$

Our equation, therefore, becomes

$$t = \frac{2360.2}{0.7772 + 0.00018023t}$$

whence

$$t = 2056^\circ \qquad (3)$$

The increased efficiency of the dried gas for obtaining high temperatures is too evident to need further comment, the difference being in round numbers 150° C. (= 270° F.) in favor of the dried gas.

Welding With Thermit.

In the meeting of Oct. 18 of the Franklin Institute of Philadelphia, Mr. Ernest Stütz, general manager of the Goldschmidt Thermit Co., presented a paper on recent developments of the thermit process in American practice. Since our readers are well acquainted with the process[1] our report will be restricted to new information, of which much is found in Mr. Stütz's paper.

Rail Welding.—The thermit welding process of obtaining a continuous rail for electric street railways has been applied during a little more than the last year in about thirty cities in the United States, and 13,000 joints have been welded; among these, 3000 in Cleveland, Ohio, and 1000 in Holyoke, Mass. In both cases the street railway companies trained their own men in the application of the process. They trained their "gangs" of four men, who, in the course of a day, made about thirty joints. They made and dried their own molds. In New York City, some 300 to 400 joints have just been completed on parts of Grand Street.

Repairs by Thermit.—The advantages of this application of thermit are seen in the problem of repairing broken locomotive frames. In this case the operation is performed with the aid of thermit without dismantling the engine, and at a cost of about $50, and the time required is two days, while previously such frames could only be repaired by dismantling the engine, and at a cost of $250 to $300, the time required being two weeks.

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[1] See the papers of Dr. Hans Goldschmidt in our vol. 1, page 527; vol. 2, pages 145, 406; vol. 3, pages 168 and 226.

The principle guiding the construction of the mold is that the thermit steel must not impinge directly on the casting to be repaired, but must run into the gate below the lowest point and be made to rise from there, through and around the fracture, enclosing the latter with a strong collar, which fuses with the material of the frame. The first steel running out of the crucible into the mold, however, becomes chilled when coming in contact with the casting, which, even when preheated, has a considerably lower temperature than the thermit steel. This chilling effect can only be overcome by a sufficiently large supply of thermit steel, so that the cooler part of the liquid mass is driven up into the riser and is replaced in the collar surrounding the frame, by metal which has practically the full temperature it received during the reaction.

The important point in making welds by thermit is to obtain a good circulation of the thermit steel, around and through the piece to be repaired, and the rule is given, that to ensure a really good weld, the quantity of thermit should be twice that which would be sufficient to fill the actual collar which is to be welded on. The main requirements for successful welds are the use of a sufficient quantity of thermit; design of the mold so as to provide sufficient vent for the air; the mold must be sufficiently porous and sufficiently dry.

If sharp sand and ordinary brickmakers' clay are used for molding material, they must be mixed in equal parts and must be burnt dry. To only skin-dry such molds means metal full of blow-holes, as the moisture at the back of the molds is sure to turn into steam and force its way out into the liquid steel.

For small operations, satisfactory results have been obtained by mixing ten parts of coarse sand with one part (by volume) so-called, of "no-grade" flour. These molds cannot be burnt dry, as they would go to pieces, but through the action of the steel the flour gets burnt together with the sand and carbonizes to such an extent as to protect the back parts of the mold. With molds made in such a manner, perfectly sound pours have been made in street railway work.

Wherever available, firebrick, cut down to size and carefully luted, offers a very convenient material for molds. They do not require hard drying, although even such a mold ought to be warm before being used.

A very important application of the process is in marine engineering. The author described at some length the process of a repair made on the Clyde line steamer "Apache."

An improved method of making such welds has lately been developed at the Essen works of the Goldschmidt Co. The mold is placed in position after the fracture has been opened out and cleaned, but before it is heated. A suitable coke stove is then placed over the mold and connected with it is a blower, which drives the fire gases through the coke, around the piece to be welded, and brings not only the latter but also the mold to a high state of temperature, thus facilitating the weld and eliminating all possibility of moisture in the mold. This process has been found particularly valuable for the repair of pieces of great diameter.

Thermit in Foundries.—Most defects in castings are naturally detected before they leave the foundries, and for this reason steel foundries were among the first to appreciate the great value of the thermit process, since it enabled them to save a large number of castings.

The author described some noteworthy repairs of gray-iron castings, especially the method applied in the repair of the spoke of a 14-foot fly-wheel at the works of C. M. Robertson & Co., in Montville, Conn. Work with gray-iron castings requires more experience in regard to preheating and cooling down gradually. More thermit is necessary to effect the weld in this case, on account of the hard, glassy scale on such castings, which resists fusion, and an addition of ferrosilicon (about 2 per cent) is advisable, to prevent hard spots at the line of junction between the thermit steel and the cast iron.

Some of the smaller defects have even been repaired without the use of a crucible, by placing a "pouring cup" over the defect and igniting the thermit directly in this cup. The cup, of course, must be absolutely dry and must be carefully luted from the outside.

Many foundries have adopted the use of thermit for reviving dull iron in the ladle and keeping open the risers (see our Vol. I., p. 533, and Vol. II., p. 405). Where there is a question of obtaining particularly dense castings, such as are used for cylinders and valves, a special thermit is recommended, which contains a slight addition of titanium. (Concerning the use of nickel thermit and titanium thermit see p. 251 of our Vol. III.)

Pipe Welding.—The first application of the thermit process for pipe welding has recently been made in New York City for the Manhattan Refrigerating Co. The pipes were laid in the ground on Fourteenth Street, and are used as service return pipes in the delivery of liquid ammonia—180 pounds pressure—to various cold storage houses along the street. Twenty-nine 1¼-inch and twenty-seven 2-inch joints were welded. The pipe was welded on the ground, to lengths varying from 40 to 100 feet, then placed into the ditch, cuts made for connections, and the final welding done with the pipe in place. On the 2-inch line an expansion bend was placed, while on the 1¼ inch this was not thought necessary. Both lines were tested out before being thrown into service, and no failures showed themselves.

ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS.

ELECTROLYSIS.

Purification of Metals.—H. M. Chance, 800,984, Oct. 3. Application filed June 2, 1905.

The metal is freed from impurities by means of reagents, which have a higher affinity for the impurities than for the metal. So far the method is exactly the same as the usual purification of metals by means of a suitable slag. But in the present case the reagent is applied in the "nascent state," being set free at the fused metal (which forms one electrode) from an electrolyte containing the purifying reagent as an ion. For instance, the fused metal may be made cathode, and as electrolyte may be used the salt of an alkali or alkaline-earth metal, not volatile at the temperature used. In this way iron and steel may be purified by contact with fused calcium chloride as electrolyte. Here calcium is set free at the surface of the metal and then serves as the purifying reagent. If a metal is to be freed from easily-oxidizable impurities, it is made anode, and an electrolyte is used which yields oxygen at the anode.

Electrolytic Production of Metallic Sodium.—E. A. Ashcroft, 801,199, Oct. 10. Application filed Oct. 3, 1903.

The inventor uses a cell, which is in principle like the Castner-Kellner; but on account of the employment of [illegible] he cannot use mercury, and employs, therefore, fused lead like Acker. The Ashcroft cell is shown in Fig. 1. At the bottom of the whole cell is the fused lead F. The central compartment A contains fused sodium chloride as electrolyte above the fused lead. D is the carbon anode; it is grooved to provide a large surface. C is the outlet for the chlorine gas set free at the anode, B the salt feed. The sodium which is set free at the surface of the lead (which here acts as cathode) alloys with the lead. By means of the electromagnet G, set in the iron pot G^1 to close the magnetic circuit, the contents of the cell are forced to whirl more or less violently, so that the sodium-lead alloy is brought from the central inner compartment into the annular outer department. Here the sodium-lead alloy forms the anode; the electrolyte J is fused caustic soda. The cathodes L are of nickel or iron. The electrolytic action in this outer compartment is that sodium passes from the sodium-lead alloy into the electrolyte, and metallic sodium is obtained at the cathode. The electrolyte in this outer compartment is, therefore, not consumed. The only material that is consumed in the whole cell is the sodium chloride fed through B into the central compartment. The end products are chlorine, set free at the anode in the central compartment, and metallic sodium set free at the cathode in the outer compartment. The temperature of the NaCl in the central compartment is maintained at 750° to 850° C., the temperature of NaOH in the outer compartment is 300° to 350° C.

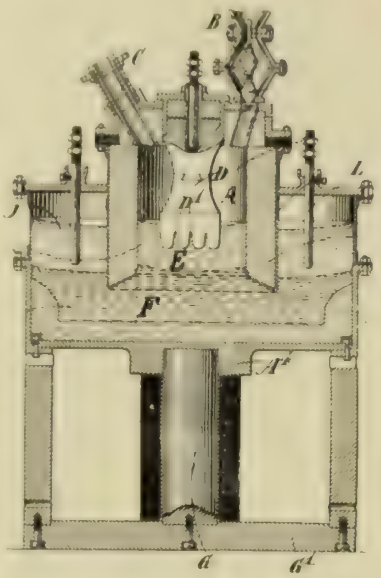


FIG. 1.—ELECTROLYTIC PRODUCTION OF SODIUM.

Producing Chlorates and Bichromates.—A. E. Gibbs, 802,-205, Oct. 17. Application filed March 1, 1904.

Chlorate and bichromate are simultaneously produced by means of the reaction, $6Na_2CrO_4 + 6Cl = 3Na_2Cr_2O_7 + 5NaCl + NaClO_3$. A stream of chlorine gas is passed through a chromate solution at 70° to 80° C., and the products of the reaction are bichromate, chlorate and chloride. They may be separated by crystallizing, etc. In this form the process is applicable in plants which produce chlorine by electrolysis of common salt, and do not wish to work up the whole output of chlorine into bleaching powder. It is, however, possible to obtain the above reaction by direct electrolysis in a cell, the chlorine being in this case obtained by electrolytic decomposition of chloride in the cell which also contains chromate. The inventors mention two possible arrangements, the cell being in both cases divided into two compartments by a porous diaphragm of asbestos, etc., and the anode and cathode being of platinum or iron respectively. Either both compartments are filled with a mixture of chromate and chloride or the chloride solution is put into the cathode compartment and the chromate solution into the anode compartment. If it is desired to increase the proportion of chlorate relatively to the bichromate in the resulting solution, the inventor adds from time to time to the anode compartment some alkali, to produce the reaction,

$$Na_2Cr_2O_7 + Na_2O = 2Na_2CrO_4$$

Reducing Metals and Making Alloys.—H. S. Blackmore, 802,153, Oct. 17. Application filed Aug. 22, 1904; Jan. 6, 1905.

To produce an alloy of two metals from their oxides, a mixture of the oxides is dissolved and electrolyzed in a solvent of molten oxides of such metals as have greater affinity for oxygen than the metals which shall form the alloy. For instance, for the production of aluminium alloys the solvent is a mixture of lithium oxide and calcium oxide in proportion of four to one. This mixture is fused by an alternating current. If the object is to produce an alloy of aluminium with copper, copper aluminate or copper oxide with aluminium oxide is added to the molten bath of calcium and lithium oxides and readily dissolves. On electrolysis with direct current a copper aluminium alloy is obtained. The alternating current for keeping the mass in a molten state is passed through the cell in a path which is at an angle to the path of the direct current for the metal deposition.

ELECTRIC FURNACES.

Induction Furnace.—F. A. Kjellin, 800,857, Oct. 3. Application filed July 6, 1905.

The object is to protect the primary winding of an electric induction furnace from the heat of the furnace chamber. The construction is shown in Fig. 2; a is the brickwork, b the annular furnace chamber, c a central opening in the brickwork, d the laminated iron core forming the closed magnetic circuit, and e the primary winding to which the electrical energy is supplied. In the central opening c between furnace chamber b and induction coil e there are placed one or more concentric double-walled casings, or jackets f of sheet iron; they do not form a closed ring, but are divided at one or more places by electric insulating material g, to prevent the production of inductive currents in the jacket. On both sides of the interruption g the inlets h and outlets h' for the cooling medium (air or water) are provided, so that it may be circulated through the jacket.

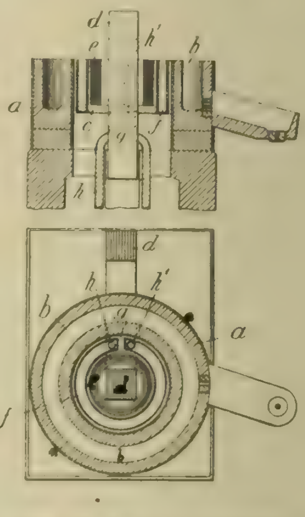


FIG. 2.—ELECTRIC INDUCTION FURNACE.

Production of Molded Blocks, Tubes, etc., of Carborundum.—F. Bölling, 801,296, Oct. 10. Application filed March 14.

A body of the desired form is first made of pure carbon and then embedded in finely powdered silicon carbide (or boron carbide). By means of a "firing" process the body (or its surface) is converted into silicon carbide by "absorption." To prepare tubes of silicon carbide, a solid carbon cylinder is treated in the manner described, whereby an outer layer of silicon carbide is formed, and an inner core of carbon remains present, and is afterwards burnt out by keeping the tube at a red heat; the outer layer of carbide remains, since it is almost incombustible.

Containers for Acids and Chemicals from Fused Quartz.—Edward Hart (assigned to General Chemical Co.), 801,378, Oct. 10. Application filed Dec. 4, 1902.

A mold A (of carbon or "metal-lined with graphite") is formed of the desired shape and connected by the conductor B with one pole of the source of electric current. A small quantity of crushed quartz or highly-silicious sand D is placed in the center of the mold, and an arc is started between the mold A and the electrode C, whereby the quartz particles become pasty or plastic, indicating an incipient fusion. When the central part of the article has thus been formed, more crushed quartz is added around it and treated in the heat of the arc, and the article is thus gradually built up. If the mold is made of non-conducting material, two electrodes must be used above it, and both moved together relatively to the mold. "When using highly-silicious material, the resulting articles present a glazed surface, and are not very strong, but are little liable to attack by acids and are wholly refractory. They cannot only stand high temperatures, but their coefficient of expansion is so small that an article of this character may be heated to a very high temperature and immediately plunged into cold water without suffering any injury.

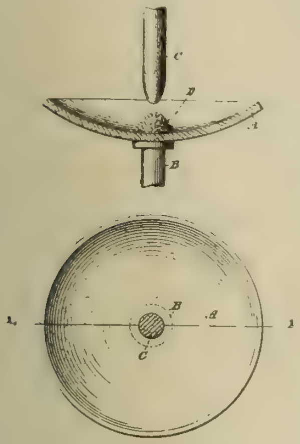


FIG. 3.—MANUFACTURE OF VESSELS OF FUSED QUARTZ.

Batteries.

Grid for Lead Accumulators.—W. Gardiner, 800,638, Oct. 3. Application filed June 11, 1904.

Claim 1 reads: "A storage-battery plate, comprising a lead grid having cross-pieces, each formed with longitudinally-extending top and bottom offsets facing in opposite directions, and suitable material to become active held in recesses formed between the said cross-pieces, the top and bottom surfaces of said cross-pieces extending horizontally from one face of the plate to the other, the said offset surfaces being vertical and at right angles to said horizontal surfaces, said horizontal portions of the grid having comparatively thin and straight unbroken outer edges, whereby the opposite surfaces of the plate, when in use, present a maximum of active surface and a minimum of inactive surface."

Zinc Accumulator.—C. B. Askew, 800,619, Oct. 3. Application filed Aug. 24, 1903.

In a "zincate" cell (with zinc as one electrode and an oxide, like copper oxide, as other electrode) there is a difficulty during charging, due to the deposition of the zinc in non-coherent form. Zinc, therefore, drops to the bottom of the cell, and when it has accumulated sufficiently there will be a short circuit. To prevent such an accumulation, the inventor divides the space between two vertical oxide electrodes into a series of transverse compartments, one above the other; curved troughs of insulating material extending transversely from one oxide plate to the next one. If any zinc drops off from the zinc plate (midway between the two oxide plates), it cannot drop down to the bottom of the cell, but only to the next trough below. This is so shaped that no contact between the zinc with the oxide plates is possible.

Miscellaneous.

Purifying Water.—J. F. Lester, 799,605, Sept. 12. Application filed Nov. 28, 1904.

Details of an apparatus combining electrolytic treatment of water with sterilization by ozone and with filtering. The water enters at the bottom of the apparatus into the receiving chamber, where the larger foreign particles settle down. This receiving chamber is closed at the top by a porous partition, through which the water passes upwards into the electrolytic chamber, containing electrodes which also act as baffle-plates, forcing the water to assume a zig-zag course. Next above the electrolytic chamber comes the ozonizing chamber. Above this is another chamber in which the ozone continues to act on the water. Now, the water passes downwards again through a filtering medium to remove the "milky scum" produced by the action of the electrical energy. The apparatus is cleaned from time to time by forcing water into the various chambers through a system of pipes.

SYNOPSIS OF PERIODICAL LITERATURE.

A Summary of Articles Appearing in American and Foreign Periodicals.

Industrial Electrochemistry in Europe.

Exposition at Liege.—The electrochemical and electrometallurgical exhibits at the International Exposition in Liege are the subject of an interesting report by Maurice Laneau, a member of the Jury, in the "Bulletin Mensuel" of the Société Belge d'Electriciens. This paper is to some extent representative of the present condition of the electrochemical and electrometallurgical industries in Europe, or, at least, in France and Belgium. The following abstract will be mainly restricted to those portions of Mr. Laneau's report which contain new information.

Electrometallurgy of Iron and Steel.—The exhibits of chief interest were those of the French and Swedish companies making ferroalloys and special steels in the electric furnace. Keller, Leleux & Co. have two plants, one of 600 hp., at Kerrousse (Morbihan), the other of 15,000 hp., at Livet (Isère); they make 250 tons of ferrosilicon per month, the percentage of Si ranging between 25 and 75 per cent. They also make 150 tons of silicospiegel per month. "The manufacture of alloys rich in Mn 38 to 40 per cent, and Si 22 to 24 per cent, consists in treating a fused charge of ferrosilicon, silica, carbon and manganese ore." The company also produces 80 tons of ferrochrome monthly and occasionally ferrotungsten. A model of Keller's furnace for the reduction of pig iron is exhibited; it is the same which was described and illustrated in our Vol. II., p. 483, Figs. 1 to 3. Keller's steel furnace is similar to that of Héroult. "An important steel works, the firm of J. Holzer, in Unieux, is at present installing an electric furnace of this type for refining open-hearth steel. It will permit the production of 8-ton ingots for the manufacture of guns of large calibre."

The Société Electrométallurgique Française, which operates under the Héroult patents, has a 13,000-hp. plant at La Praz (Savoie), a 6,000-hp. plant at St. Michel-de-Maurienne (Savoie), and a plant at Gardanne (Bouches du Rhône). They produce aluminium, ferrochrome, ferrosilicon and other ferroalloys, and 3,000 tons of tool steels per year. Concerning the Héroult steel process see our

Vol. I., pp. 63, 287, 440, 467, Vol. II., pp. 408, 481; Vol. III., pp. 45, 155, 337.

The Societe Anonyme Electrometallurgique, Procedes P. Girod, has two plants, one of 4,000 hp., at Courtepin (Switzerland), and the other of 8,100 hp. at Ugine (Savoie). The company is said to be the largest producer of ferrotungsten in the world. "This company also studies the production of steel, and has made for several years continued industrial tests, and the reason why it has not yet entered this field on a commercial scale is because it does not wish to compete with its customers for ferroalloys." Fig. 1 shows the Girod furnace, the upper diagram being a vertical section along the line AB in the lower diagram. It comprises a containing vessel of magnesia brick and has a cover of silica brick, through which pass several electrodes, all of which are connected to one pole of the supply circuit. From there the alternating current passes through the bath to the lower electrodes, which are embedded in the brick work, and consist of water-cooled pieces of cast iron. There are fourteen such lower electrodes for a furnace of 2 meters diameter, and each of them is in contact with a canal reaching up to the crucible and filled with extra soft iron before the furnace run begins. The furnace is placed on a cradle to facilitate operation. The charge consists of scrap mixed with a small amount of good cast iron. The upper electrodes are suspended in the slag, which is made up from the very pure ores of the Pyrenees and from Batère hematite. The decarburization is pushed as far as possible, and the steel is afterwards recarburized. In the 250-kw. furnace, which has been in actual operation, 1.5-ton charges are treated, 1 ton taking about 4½ hours. Concerning the cost per ton of steel, the following figures are given:

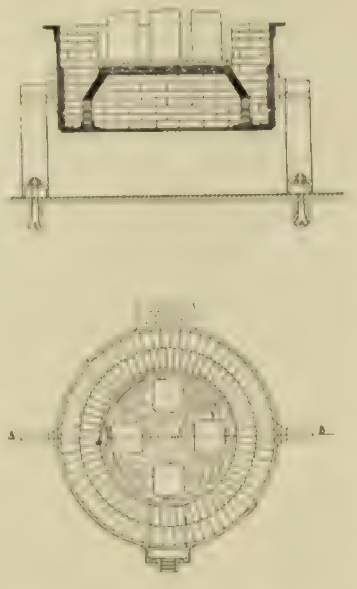


FIG. 1.—OSCILLATING STEEL FURNACE OF GIROD.

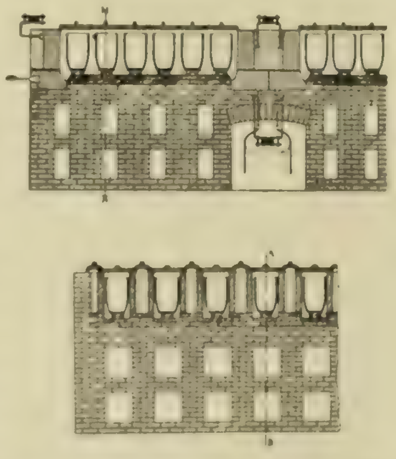


FIG. 2.—RESISTANCE FURNACE OF GIROD.

Energy (1,060 kw-hours at 0.5 cent)....	$5.30
Electrodes (10 kg.).....................	0.20
Maintenance of furnace................	1.60

Girod has also made experiments with electric resistance furnaces in which crucibles are heated by means of resistors from the outside. (See also our Vol. II., p. 309, Figs. 1 and 2.) The crucibles are placed in retorts between which the resistor material is heaped up (evidently in crushed form), consisting of carbon and silica. The nature of the material which forms the walls of the retorts permits a temperature up to 1400° or 1500° C. (while with alumina brick temperatures of 1600° to 1700° C. can be obtained). The construction of the furnace is shown in Fig. 2, the upper diagram being a section through line AB, the lower diagram through line MN.

To obtain higher temperatures Girod has devised the arrangement shown in Fig. 3. He places resistances on the bottom of the retort. On account of the restricted section of these resistances, obtained by the insertion of "fromages" 3, which support the crucibles, the temperature may be brought up to 2000° C. In order to protect the bottom and the sides of the retort, two zones of different mixtures of resisting material 1 and 2 are provided in such a way that the central portion conducts the greatest part of the current. A higher potential difference is applied at the terminals of these resistors than at those of the others. It is stated that the temperature can easily be adjusted by 50° or 60° up to 2000° or 2500° C., by varying the impressed e. m. f., the connections between the resistors and the pressure on the resistors. From tests made in a 25-kw. furnace, the cost of steel per ton is given as $4.60 (being the sum of cost of power, 1440 kw-hours, $2.28; crucibles, $1; maintenance, labor and amortization, $1.30).

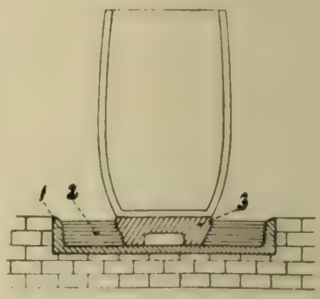


FIG. 3.—DETAILS OF GIROD RESISTANCE FURNACE.

G. Gin exhibits models of two types of steel furnaces. His first type was described in our Vol. II., p. 20, and Vol. III., p. 297; it is characterized by a long, narrow channel filled with the charge (somewhat like an incandescent lamp filament on a very large scale). A furnace of this type is stated to be in operation in Plattenberg, Germany, since Jan., 1905, but nothing is yet known concerning the economical results. The second type of furnace, exhibited by Gin in form of a model, was described on page 372 of our October issue.

The Metallurgiska Patentaktiebolaget, of Stockholm, Sweden, produces annually in the Kjellin induction furnace about 1500 tons of high-grade steels (this is about 1½ per cent of the total production of steel in Sweden). Concerning the Kjellin furnace, see our Vol. I., pp. 141, 283, 376, 562, 526, 546; Vol. II., p. 479; Vol. III., pp. 134, 299, 341, and a patent of Kjellin in the Analysis of Current Electrochemical Patents in our present issue.

Electrolysis of Water.—L'Oxhydrique Société Anonyme of Brussels, Belgium, makes regular demonstrations at the Exhibition on welding with the oxyhydrogen flame. This company uses the electrolyzer of Garuti for the production of oxygen and hydrogen gas from water. One of the latest applications—which, however, seems to be still in the experimental stage—is the cutting of iron sheets by a stream of oxygen. The Garuti electrolyzer was noticed in our Vol. I., p. 463. Both caustic potash and caustic soda are used as electrolytes; caustic potash has the advantage of smaller resistivity, and consequently of smaller power consumption; caustic soda has the advantage of lower cost.

Electrolysis of Sodium Chloride; Mercury Cathode.—A special pavillion in the Exhibition contains the exhibits of the Société Solvay et Cie, of Brussels, Belgium. This company has studied since 1895 the electrolysis of sodium chloride, the starting point being the patents of Castner and Kellner. The Solvay company has developed its own mercury cathode cell, the advantages of which are said to be large units and small attendance required. The Solvay cell is shown in Fig. 4. The arrangement of the carbon anodes is shown in the lower diagram. The cathode is formed of mercury covering the bottom of the cell. The mercury enters through pipe B; the amalgam, which is lighter than the mercury, flows over at C, and is taken out of the cell through tube D. The amalgam is then treated outside of the cell to obtain sodium hydroxide, and the regenerated mercury is returned into the cell through M and B. (It will thus be seen that the mercury acts in this cell as cathode pure and simple, and does not act as a bipolar electrode.) The sodium chloride solution enters the cell through S^1, and leaves through S, the direction of its flow being, therefore, opposite to that of the mercury. The 1000-hp. plant at Jemeppe-sur-Sambre, the cell room of which is shown in Fig. 5, produces in the whole 7500 tons of various products per year; caustic soda, caustic potash and bleaching powder. The caustic soda is of high purity, containing 98.95 per cent of NaOH and only 0.09 per cent of NaCl. (The caustic soda of the Castner-Kellner Electrolytic Alkali Co., of Niagara Falls, contains 99.5 per cent NaOH.)

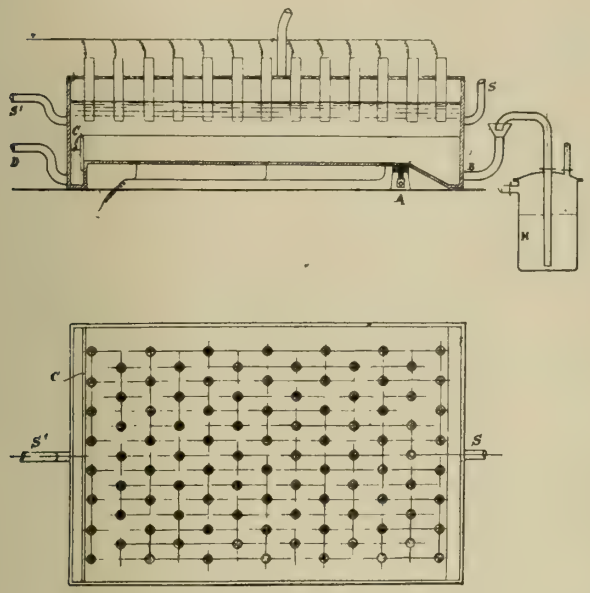


FIG. 4.—SOLVAY CELL FOR ELECTROLYSIS OF SODIUM CHLORIDE.

FIG. 5.—CELL ROOM OF JEMEPPE WORKS OF SOLVAY CO.

Chlorates, Sodium Peroxide, etc.—The Société d'Electrochemie has two plants at St. Michel de Maurienne and at Vallorbe. The principal products are potassium and sodium chlorates, of which 13,000 to 15,000 are made per year by the Gall and Montlaur process. They also make annually over 200 tons of peroxide of sodium; this is made by passing a current of air over metallic sodium enclosed in an electrically-heated platinum tube. "Sodium peroxide is highly praised for bleaching of linen and wool; by dissolving it in acidulated water in the desired proportions, one gets oxygenated water." "By compressing the dioxide with copper salt, a product is obtained called oxylithe, which serves for the evolution of oxygen by simply immersing it in water. Oxygenated water and sodium hydroxide are thereby formed. The latter, by reacting with the copper salt, gives a hydrate which determines the decomposition of the oxygenated water formed, thus resulting in the evolution of oxygen." Since the demand for sodium peroxide has somewhat decreased, the company uses part of it for the manufacture of barium peroxide by a process which is not described.

The Société Anonyme des Forces Motrices et Usines de l'Arve, with a 13,000-hp. plant at Chedde (Haute-Savoie), produces 4000 tons of chlorates per year.

Electroplating.—Prof. Classen, of Aix-la-Chapelle, exhibits electrolytic zinc deposits which are perfectly bright when taken from the bath. "In order to give an idea of the appearance of the deposit, we may compare the articles galvanized by his process with tinned articles." The process is not described. The brightness of the deposit is considered to be an indication of its good quality.

Copper tubes made by the Elmore process are exhibited by the Société Française d'Electrométallurgie, which operates this process since 1891 at Dives-sur-Mer (Calvados). The rôle of the agate burnishers is apparently not yet definitely fixed; at Dives the pebble-stones of agate are fixed in their chapes and operate by friction, while in the German plant at Schladern they are mounted on the axles and operate by simple pressure.

Electrolytic Refining Plant.—The Russian refinery of J. K. Nicolaeff, in Moscow, produces per year 2500 tons of different products (500 tons electrolytic copper, 200 tons bronzes, 800 tons alloys, 320 tons of tubes and other articles of lead, 320 tons copper sulphate, 240 tons iron sulphate, 1 ton silver, 0.04 gold, etc.) of a total value of $1,000,000.

Detinning.—In the detinning section of this plant the current is furnished by a 12-kw. generator, giving 600 amps. The tinned iron scrap is placed in iron caskets forming the anode. The electrolyte is a solution of NaOH. Under the action of the current the tin oxidizes, and metastannate forms; the tin is deposited in spongy form on the leaves of metallic copper, which form the cathodes. After a certain time, the solution absorbs CO_2 from the atmosphere and contains impurities, due to the saponification of the varnish, rust and sodium ferrates. "The solution is regenerated in iron vessels by a stream of Co_2, which precipitates the SnO_2 of the metastannate and transforms the electrolyte into a mixture of alkaline carbonate with the impurities mentioned above. They are eliminated, and the solution is regenerated by means of quick lime and heating."

Copper Refining.—This part of the plant contains 240 refining cells, the current being furnished by four generators, giving 25 by 500, 10 by 480, 20 by 400, and 25 by 300-volt amperes, respectively. The first of these supplies the cells, in which the current density is 0.5 amps. per square decimeter, and in which anodes containing 90 to 95 per cent of copper are refined. For the refining of anodes containing 76 to 85 per cent of copper a current density is used not [illegible] to 1 amp. per square decimeter. The electrolyte contains 10 to 15 per cent of copper sulphate and 6 per cent free sulphuric acid.

Batteries.—A. Wedekind exhibits a cell of the Lalande type, with zinc and copper oxide as electrodes and potassium hydroxide as electrolyte. When discharged, the copper plate is reoxidized by heat, while the zinc and the electrolyte must be replaced. Other primary cells are exhibited by the Maison Leclanche et Cie, of Paris, and the Maison Delafor, of Paris. Among the numerous storage batteries exhibited by Belgian and French companies are the well-known Tudor accumulator and the chloride cell of the Société pour le Travail Electrique des Metaux, of Paris (which is affiliated with the Electric Storage Battery Co. of this country).

INDUSTRIAL ELECTROCHEMISTRY.

Chlorate Analysis in Electrolytic Chlorate Manufacture.—In the "Chemiker-Zeitung," Vol. XXIX., No. 18, M. Couleru remarks that a few years ago the ampere-hour efficiency of electrolytic chlorate manufacture was about 60 per cent and the watt-hour efficiency 30 per cent, while with improved modern methods an ampere-hour efficiency of 90 per cent and a watt-hour efficiency of 65 per cent is obtained. For chlorate analysis ordinary direct chemical methods may, of course, be applied, but indirect physicochemical methods are in some respects more convenient. These methods are based on gas analysis. Electrolysis of water yields hydrogen and oxygen in the volume proportions of 2 to 1. In the present case the oxygen is consumed for oxidation of chloride, that is, for the formation of chlorate. A theoretical ampere-hour efficiency of 100 per cent would correspond to pure hydrogen. Any impurities in the hydrogen (consisting mainly of oxygen and traces of chlorine and chlorine products) indicate a decreased ampere-hour efficiency. The amount of oxygen mixed with the hydrogen will, therefore, be a measure of the actual ampere-hour efficiency. If the latter is 0, 50, 100 per cent, the hydrogen contains 33.3, 20, 0 per cent of oxygen, respectively. Since hydrogen is sixteen times lighter than oxygen, small quantities of oxygen are easily found by weighing. Any balance which indicates 1 or 2 milligrams is suitable for this purpose. On one scale a reversed Erlenmeyer flask is suspended, a thin glass tube passing upwards into the upper part of the same. The balance is adjusted for pure hydrogen passing through. As soon as a gas heavier than pure hydrogen passes through, the balance will tip, and the weight required to readjust the balance will indicate the amount of oxygen in the gas and the corresponding ampere-hour efficiency. A balance may thus be empirically calibrated by passing different mixtures of gas through. This simple and cheap method is said to be sufficient for many purposes. For larger experiments and for continuous records the gas balance of Lux is recommended. It is thus possible to find at any time the current efficiency directly. If an exact recording voltmeter is also available it is possible to follow accurately the electrolytic operation and to find at once when something is wrong. Some tables are given, and it is pointed out that some precautions are necessary in the use of this method, since the composition of the electrolytic gas depends on various factors. If, for instance, the temperature in the electrolyte decreases, the content of hypochlorite increases and the oxygen gas is consumed for production of hypochlorite. The same is the case at the beginning of electrolysis if fresh chloride solutions are used. For skilled experts in chlorate manufacture, however, [illegible] highly recommended and considered to be very practicable.

Hypochlorite for Disinfecting Purposes.—The issue of Sept. 22 of the London "Electrician" contains an abstract of a report by Dr. F. W. Alexander, "medical officer of health" to the Metropolitan Borough of Poplar. It is entitled "Upon the Manufacture of Electrolyzed Salt Water (Chloride of Sodium and Chloride of Magnesium) for Disinfecting Purposes." He recommends the distribution of the hypochlorite solution throughout the borough for deodorizing and disinfecting purposes. The chemical explanation of the electrolytic production of hypochlorite, as given in the report, is slightly amusing.

Dry Cell.—For the electrification of electromotor needles and similar purposes a dry cell with constant e. m. f. is useful. J. Brown describes in "Lond. Elec." of Sept. 15, tests of such a pile constructed on Daniell's principle. Sheets of commercial zinc and copper, 9 inches square, were coated on one side with sheets of twilled cotton fabric (as used for glass cloths), wet with hot 10 per cent solutions respectively of zinc sulphate and copper sulphate. When dry these were built up in the appropriate way with a sheet of plain blotting paper between each pair of coatings, to represent the usual porous diaphragm, and the whole compressed in a screw press between rubber sheets as insulators. When built (Feb., 1903) the e. m. f. was one Daniell per cell or rather more. It then dropped to a steady, though lower, value for two and one-half years, registering 0.95 to 0.9 Daniell till July, 1905, when it was 0.9. Thereupon, dismounting the pile for examination it appeared to be unchanged, and on reforming it the same voltage was given, 0.9. After short-circuiting, the pile recovered its e. m. f. immediately. Of course, a very minute current only passed.

Tin Refining.—In the September issue of "Elektrochemische Zeitschrift," H. Mennicke begins to discuss the possibility of applying Betts' process of lead refining with modifications to the problem of tin refining. The present installment contains only some general introductory remarks.

Ozone.—In the July and August issues "Elektrochemische Zeitschrift," O. Kausch gives a summary of progress made in recent years in the production of ozone from silent discharges. Various new progresses are described from patent specifications.

Electrometallurgy of Iron and Steel.—A concise summary of the present situation with respect to the use of iron reduction and steel refining is given in a paper by R. S. Hutton in the "Jour. Soc'y Chem. Ind.," June 15; it contains no new details, but is an impartial, fair and conservative statement of the situation as it is at present. Mr. Engelhardt's elaborate paper on the Kjellin induction furnace, which was reviewed at great length in our August issue, is the subject of a long and well illustrated article in "Iron Age," Oct. 19.

THEORETICAL AND EXPERIMENTAL.

Specific Heat of Iron at High Temperatures.—A careful determination of this quantity, which is of vital importance for metallurgical calculations, has been made in the (Brit.) National Physical Laboratory, the results being given in the October issue of the "Philosophical Magazine." The iron tested was very pure, the analysis being 0.01 C, 0.02 Si, 0.03 S, 0.04 P, trace Mn. The following table gives the results; total heat means total heat from 0° to T° C., and is given in calories; mean specific heat means the mean specific heat between 0° and T°:

T	200	300	400	500	600
Total heat ...	23.5	37.0	51.3	66.9	83.8
Mean specific heat..	.1175	.1233	.1282	.1338	.1396
T	700	800	900	1000	1100
Total heat ..	104.1	127.8	148.0	155.7	168.8
Mean specific heat..	.1487	.1597	.1644	.1557	.1534

The author states that the diminution at 900° and subsequent rise in the specific heat of iron require confirmation.

Specific Heat of Gases at High Temperatures.—The October issue of the "Phys. Rev." contains an account of an investigation carried out by L. Holborn and L. M. Austin in the German Reichsanstalt, on the specific heat of gases at temperatures between 20° and 800°. The gases were heated in an electrically-heated tube, and many precautions were taken which are described. The following table shows the mean of the observed values for the mean specific heat of the simple gases:

	Nitrogen.	Oxygen with 9.5 Per Cent. N.	Oxygen.	Air.	Air Circulated from N. and O.
Between 20° and 440°.	0.2419	0.2255	0.2240	0.2366	0.2377
" 20° " 630°.	0.2464	0.2314	0.2300	0.2429	0.2426
" 20° " 800°.	0.2497	.....	.....	0.2430	.....

The value for pure oxygen was calculated according to the known percentage of nitrogen mixed with it. In the same way a value was calculated for air, derived from the two values for nitrogen and oxygen. The following table gives the variation of the specific heat of carbon dioxide. In the column Cal the values given are derived from an empirical formula:

CO_2	Obs.	Cal.
Between 20° and 200°..........	0.2168	0.2173
" 20° " 440°..........	0.2306	0.2312
" 20° " 630°..........	0.2423	0.2410
" 20° " 800°..........	0.2486	0.2486

Titration Voltameter.—In the "Physik. Zeit." of Sept. 15, D. A. Kreider reports on a research made in Yale University on the construction of a titration voltameter for which a high accuracy is claimed. It is based on the liberation of iodine from an acidulated solution of potassium iodide and its estimation by means of sodium thiosulphate in a burette. The construction of the cell is shown in Fig. 6. The anode is placed at the bottom, and above the iodide solution, which forms the anolyte, a catholyte of lower specific gravity is used; it consists of a solution of hydrochloric acid. This prevents satisfactorily the iodine from diffusing towards the cathode and diminishing the current efficiency.

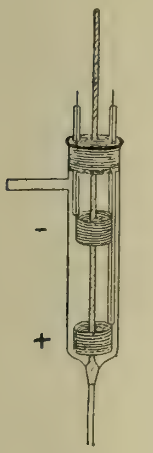


FIG. 6.
TITRATION VOLTAMETER.

Iron and Steel.

Electric Properties of Steel.—At the first general Congress of Swedish Electrical Engineers, held in Stockholm, from Sept. 9 to 11, a paper on iron and steel for electrical purposes was presented by F. A. Enström, who, together with G. Dillner, had studied the influence of chemical composition and of manufacturing methods on the iron and steel used in electrical machines and transformers, at the testing laboratory of the Royal Technical University of Stockholm. The researches were made with sheet iron and with cast steel. The desired properties of the former are a low hysteresis loss and a high electric resistance, in order to keep down the eddy current losses. The main requirement in cast steel, which is used chiefly for pole pieces, is high permeability.

The instrument used for the researches was the Du Bois magnetic precision balance, which was accurate within some tenths of a per cent. With this instrument many different kinds of iron and steel, chiefly Swedish and German, were examined. At the first, samples were taken from the open market and later special samples, made expressly to contain different percentages of silicon and aluminium, were ordered. In casting the silicon steel it was observed that, with a percentage of this element above 1.5, the shrinkage of the steel is so great that cavities were difficult to avoid, while with about 1 per cent of aluminium there was no difficulty in casting the steel. With higher percentages of this element, however, the steel becomes rather thick, and from this cause many of the samples were unsound and unsuitable for testing. A chemical analysis of all the samples was made, the percentage of carbon, silicon, manganese, phosphorus, sulphur and aluminium being determined. The results obtained may be stated as follows:

For sheet iron it was found that the coercive force and the Steinmetz hystersis coefficient is proportional to the carbon percentage within the range of 0.5 per cent. Thus it is obvious that a small amount of carbon is favorable in laminated iron for armature cores and transformers. In cast steel no determinal effect appears to be produced by increasing the carbon percentage. With the highest carbon percentage investigated, about 0.30 the values of the maximum induction and of the permeability at the usual induction in field poles are quite satisfactory. A high percentage of carbon produces an increase in the electrical resistance.

An increase of the silicon causes a quite noticeable increase of the coercive force and of the hysteresis loss and also of the electric resistance, while the properties of permeability, remanence and maximum induction hardly undergo any alteration. Therefore, for sheet material, silicon should be avoided, while in the case of cast steel some silicon in the material may be advantageous owing to its raising the electric resistance. More than 2 per cent, however, should be avoided, because otherwise there will be a tendency to form cavities.

It has been stated that an addition of aluminium quite noticeably reduces the hysteresis loss, whilst it increases the electric resistance, and thus reduces at the same time the eddy-current loss. It has been found, too, that an addition of aluminium will reduce the maximum induction and the permeability for ordinary saturation values. The general conclusion is, therefore, that an addition of aluminium improves sheet material for constructing dynamos. As regards cast steel, the matter is different. With such a high percentage of aluminium as is required in order to increase the electric resistance to an effective degree, the magnetic properties will be too much affected, and the usefulness of aluminium in cast steel is questionable, to say the least. It is of interest, however, to notice the results obtained from the researches on cast-steel rods containing both silicon and aluminium. In this case low hysteresis losses are found, to be ascribed, no doubt, to the influence of aluminium, while the influence of this element on the permeability and maximum induction seems to be neutralized by the addition of silicon. The addition of silicon and aluminium together seems thus to tend towards a general improvement of the magnetic properties.

C. Benedicks has given a formula for calculating the electric resistance of iron from the number of atoms present of any element, which is contained in the iron in solid solution. This formula has been tested, and the agreement considered very close. (A summary of Benedicks' researches will be given in a future issue.)

It has been found that, as a rule, annealing improves the magnetic properties in general, and especially decreases the hysteresis loss. As to sheet material that had been annealed in packs, a decided difference amounting to about 10 per cent in favor of the edges was found between the edges and the center, regarding coercive force and hystere-

[illegible] These researches prove, therefore, that the general assumption that the edges contain a lower grade of material than the center, is false. The time and temperature of the annealing is very important, 147° F. seems to be the best temperature.

The researches have shown that the Swedish-Lancashire forge process and the basic open-hearth process gives the best steel, next comes the acid open-hearth process, while the Bessemer process gives a much lower grade of material, possibly because the Bessemer steel has a greater opportunity of dissolving gases when the air is passed through the bath of molten metal.

During the discussion following the reading of the paper, the question of the ageing of the material was brought up, and it was pointed out that the ageing depends largely on the annealing. If the annealing is carried out at the proper temperature the danger of ageing is very small. Regarding the advisability of annealing the laminated packs after punching, it was stated that Swedish and German manufacturers have abandoned this method which, however, is still in use in America and England.

Cleaning Blast Furnace Gas.—In the October issue of "Cassier's Magazine," A. Sahlin discusses the economies which may be attained by cleaning the blast furnace gas before further use. According to the different uses to which the gas is later to be put, different methods of cleaning with corresponding results, should be adopted. The gas engine would do its best work with absolutely pure gas; it does not work well with gas containing above about 0.044 grain per cubic foot. Stoves and boilers on the other hand do not require the cleaning to be carried beyond 0.130 grain per cubic foot. When the gas is to be used in engines, the cleaning cannot be carried too far, while for use in stoves, furnaces, kilns or under boilers it would be an extravagance to seek to remove the very last fraction of a grain. The author recommends the adoption of a three-stage process. The first stage should consist of the employment of gravity and centrifugal force to deposit such heavy particles as can be separated by these forces. The proper instrument for doing this is a circular dust-catcher, into which the gas is introduced near the bottom in a tangential direction, and whence it is removed through a central pipe connecting with the top of the dust-catcher. The larger such a dust-catcher is the better. At a modern plant in the north of England, a dust-catcher having a diameter of 21 feet, and a height of 80 feet, is employed with good results. This first stage of the cleaning is performed without any tangible cost or expenditure of power. When issuing from the dust-catcher the gas may contain from 0.66 to 2.84 grains per cubic foot. This dust is so fine that it will float in the gas for an indefinite period. It must be removed by cooling the gas by contact with water and with surfaces sprayed with water. For this purpose different machines have been invented, all of which may be divided into the following classes: Stationary cleaners, slowly-revolving fresh-contact cleaners, rapidly-revolving or atomizing cleaners. Several kinds of apparatus of these types are described and illustrated. Rapidly-revolving or atomizing cleaners or fans are effective cleaners, but require considerable power in proportion to the volume of gas cleaned. At the de Wendel Works, at Hayange, the entire quantity of gas from seven blast furnaces is cleaned by fans only, and with gratifying results. The cleaning is carried on in three stages, namely, by means of large fans, which leave about 0.2 gram of dust per cubic meter of gas, which is then sent to the stoves and boilers. The second cleaning is carried on in smaller and more rapidly-revolving ventilators, and leaves less than 0.1 gram of dust per cubic meter of gas. This small quantity is still further reduced by passing the gas through a scrubber before it is carried to the large central electric power plant, which has a capacity of 7,000 hp. In 1900, Thiessen designed a modification of the ventilator. The Thiessen apparatus consists of a conical shell revolving on a central shaft placed inside of a conical containing shell. To the outer surface of the inner cone, fan blades are riveted at different angles. Gas enters the apparatus at the small end of the containing cone, and is drawn off at the large end. Numerous pipes, arranged along the sides of the outer shell, admit water. The gas is thoroughly cooled, that is, dried, and is freed from dust.

RECENT METALLURGICAL PATENTS.

Tin.

In the smelting of tin ores, slags are obtained which, even after repeated smelting in reverberatory or blast furnaces, still contain a certain percentage of tin in the form of ferrous stannate or silicate of tin. C. A. L. W. Witter (801,290 and 801,820, Oct. 10) patents a process by which, he claims, the tin contents of the slags can, with profit, be extracted even down to 0.5 per cent or less, the slags being smelted and reduced in a blast furnace adapted as a hearth-furnace together with lead, lead ores or other plumbiferous materials. In this way an alloy of lead and tin is obtained, the lead serving as a solvent for the tin reduced from the slags. In order to remove the iron, there is added to the slags on smelting, together with the lead, a quantity of sulphur, or material yielding sulphur, corresponding to the iron to be separated, so that the iron as it separates is instantly bound with the sulphur, and on drawing off, the alloy can be removed as a matte from the furnace.

The alloy of tin and lead thus obtained contains a very large proportion of lead. The alloy is melted in a reverberatory furnace, and when the temperature is such that the molten mass shows a red glow, an air-blast is blown on the molten mass. The tin in the alloy is oxidized, but at the same time a certain portion of the lead is oxidized as well. An oxide mixture containing a high percentage of tin is thus obtained, which can be drawn off the surface of the metal bath and then again reduced in a reverbatory or blast furnace to form an alloy containing a large proportion of tin.

Preventing Corrosion of Metallic Screens in Ore Dressing.

Two patents of H. S. Anderson and J. W. Bennie (796,172 and 796,390, Aug. 1) are interesting, since an undoubtedly sound principle of electrochemistry is applied in the same to a new purpose. The metallic screens used in concentrating jigs and similar apparatus are subject to great wear, due to the corrossive action of acids, alkalies or salts in the water used for concentration. To prevent this corrosion, the inventors make the screen the cathode or electronegative element of an electrolytic cell. This may be done in two ways. A second electrode (acting as anode) is required in either case; it is placed on the slanting bottom in the tank below the screen. This second electrode may be made of graphite (not attacked by the solution), and the screen and this graphite plate connected to a dynamo outside of the cell; or the second electrode may be made of zinc and electrically connected with the screen. In the latter case we have a short-circuited cell. If the screen is made of copper and the concentrating liquid contains sulphuric acid or copper sulphate, as is usual in the treatment of ores containing copper sulphide, the electrochemical action will prevent any oxidation or corrosion of the screen; on the contrary, either hydrogen will be set free on the screen or copper deposited on it, thus building up the screen in opposition to the abrasive action of the ore. By thus sacrificing "an inexpensive material in an inexpensive form" (electrical energy in the first case and zinc in the second case) the inventors protect the more expensive screens against corrosion and abrasion. They state to be able to use steel wire or punched steel screens, many times less expensive than brass or copper. It would be very interesting to get exact data as to the cost of the method, compared with the saving. Such data can be obtained only by practical operation on a large scale and extending over a sufficiently long period. The principle is, of course, the same as the use of zinc for the protection of boilers and condenser tubes on vessels, which application has proven commercially successful.

Magnetic Concentration.

The commercial importance of magnetic concentration is indicated by the number of inventors working in this field. J. P. Wetherill (801,947, Oct. 17) patents details of a system in which several magnetic concentrators are used in series, the material not attracted by the first magnet being subjected to a second magnetic concentration and so on.

H. H. Campbell (790,342, May 23) provides a system of magnetic concentration similar to water concentration. There is an inclined table with a diagonal row of electromagnets. These magnets are reciprocated in a direction lengthwise of the table. At the end of their forward stroke the magnets are demagnetized, and immediately before they reach the limit of their return stroke they are remagnetized. The magnetic particles are thus forced to travel in a diagonal direction, being subjected both to magnetic attraction and gravity, while the non-magnetic particles descend simply under the action of gravity.

E. Langguth (793,137, June 27) patents details of construction of separators in which a non-magnetic screen is interposed between the active surfaces of the magnets and the material, and the latter is conveyed along the screen, and thereby distributed according to the magnetic properties of its constituents. The invention consists in a specific disposition of the rotating parts of the active electromagnet by which the field is divided into a number of radially-disposed stripes, which retain and convey the magnetic particles of the ore and permit the non-magnetic material to drop out between the radii as the latter in rotating approach a vertical position.

L. T. Weiss (791,305, May 30) endeavors to extend the application of magnetic concentration to the separation of gold particles from sand and quartz with which it is mixed. For this purpose the mixture is passed through an electroplating tank in which the gold particles are coated with a film of iron. This is then followed by magnetic concentration.

BOOK REVIEWS.

Engineering Chemistry. By Prof. Thomas B. Stillman. Third edition. Easton, Pa.: The Chemical Publishing Co. Price, $4.50.

Prof. T. B. Stillman, of the Stevens Institute, has revised his "Engineering Chemistry" to July, 1905. This excellent book is founded on the series of lectures he gives to the students of Stevens, and covers the ground of analysis, chemical engineering and metallurgy needed by an engineer. Although some important subjects are omitted, due, we suppose, to the fact that others are taken up in detail, nevertheless, the practical nature of the chapters are enough to make the volume a valuable one to the chemist and metallurgist as well as to the mechanical engineer.

Among the prominent features are up-to-date analytical methods; water for sanitary or boiler purposes; practical calorimetry; producer gas; coal and coke, Portland cement, building stone and brick, etc., etc. In fact, we can but recommend this book most heartily, for it is written for the needs of the practical man and fulfills the purpose for which it was designed.

Highly Sensitive d'Arsonval Galvanometer.

To the numerous types of d'Arsonval galvanometer, made by the Leeds & Northrup Co., Philadelphia, recently a new one has been added which is considered to be the most refined instrument of the series. It is particularly intended for those who make considerable use of a very sensitive galvanometer for technical measurements or for purposes of research.

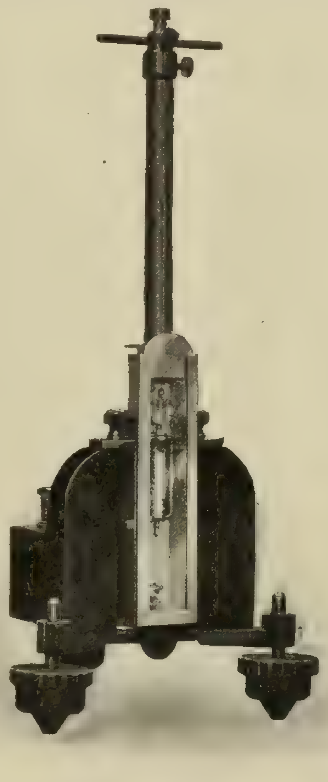
In design it is a radical departure from the old style laminated magnet with suspension tube, which permitted but small clearance for the coil and in which the suspended system was not visible.

The following features of the improved design render this highly sensitive instrument very convenient for practical use.

Through the large open-glass front the entire system is always exposed to view, and when anything is wrong with it the trouble can be readily detected.

In such cases the removal of the tube containing the moving system is readily effected by the loosening of two thumb screws. The tube may then be removed from the magnet and laid horizontally on a table, which is often of great convenience, if any adjustments are to be made, such as the insertion of a new suspension.

The upper suspension is protected against breakage by being attached to a small spiral spring just strong enough to support the coil. When the instrument is jarred, or the coil is knocked, the spring yields and protects the suspension. This arrangement does not change the manner of adjusting the zero or the length of suspension required, nor makes it more difficult to put in a new one.



GALVANOMETER.

The tube has at its back a simple device for clamping the coil, so that the galvanometer and tube may be moved about.

The coil is of the rectangular type, similar to the Leeds & Northrup type H coil, but is made smaller, so that the system will be lighter, and give a greater sensibility with a given period of swing.

A small index point on the coil shows when the galvanometer is exactly level and the coil is swinging centrally in the field.

To provide the highest possible insulation, the entire galvanometer is insulated from earth by means of three hard rubber leveling screws, terminating in steel points. The bottoms of the leveling screws have deep grooves turned in them, thus making a "petticoat" insulation.

Stoneware Apparatus for Chemical Purposes.

In a very interesting paper presented a few years ago before the American Electrochemical Society (our Vol. I., p. 62, "Transactions"; Vol. II., p. 290), Mr. David H. Browne pointed out that the successful conduct of any electrochemical process depends not only upon the care with which the electrical and the chemical parts of the subject are worked out, but to a great extent upon the mechanical questions involved and the means provided for conveying the solutions to the baths and disposing of the products of the same. The same is really true of every chemical process. In the paper mentioned, Mr. Browne recorded in a most interesting and humorous way the troubles which he has had in many years' work with pumps and other accessories in chemical operations. A considerable part of his paper dealt with stoneware apparatus, which have proven extremely suitable for many chemical purposes.

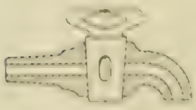


FIG. 1.—STONEWARE COCK.

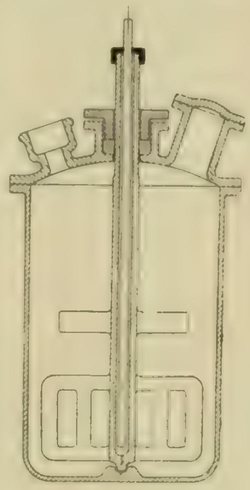


FIG. 2.—STONEWARE STIRRER.

In connection with this subject, a paper presented by Prof. Georg Lindner, in Karlsruhe, before the Association of German Engineers, and published in this year's volume of the *Zeitschrift des Vereins Deutscher Ingenieure*, contains much of interest, giving a review of the various stoneware apparatus which are now being built in Germany for chemical purposes.

The author also gives mathematical formulas for the calculation of centrifugal pumps and exhausters, for which the readers may be referred to the original German paper. In the following we will mainly reproduce drawings of the stoneware apparatus described by Prof. Lindner. These drawings are quite interesting, since they clearly indicate the details in design in which they differ from patterns made of iron. In order to get the necessary mechanical strength the designs had, of course, to be changed in many details.

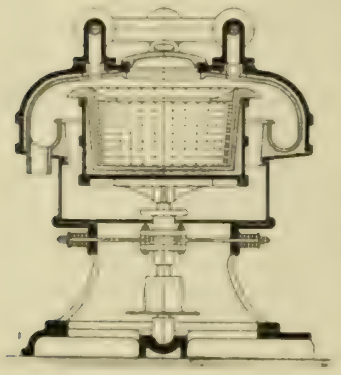


FIG. 3.—CENTRIFUGAL MACHINE.

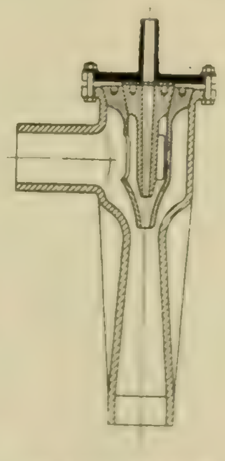


FIG. 4.—EXHAUSTER.

Fig. 1 shows a stoneware cock. Its design presented originally difficulties with respect to tight fitting, but these difficulties are considered to have been completely overcome long ago.

Fig. 2 is a sectional diagram of a stirring apparatus; the vertical axle of the stirring mechanism is continued outside of the apparatus in form of a wrought iron shaft coupled to the driving motor. The main difficulty is here, the junction between the axle and the driving shaft. One method of overcoming this difficulty is to make the axle of the stirring mechanism hollow, and insert in its interior the driving shaft, the space between the driving axle and the interior surface of the hollow shaft being filled with sulphur or concrete.

Fig. 3 shows a centrifugal machine in which the number of revolutions of the drum is 800 per minute, the internal diameter of the stoneware drum being 850 mm. In order to protect the drum against cracking, it is covered with a mantle of sheet steel. An interesting conclusion from exact calculation is that a mantle of cast iron would not do, because when revolving with the same speed the cast-iron mantle would expand more than the stoneware drum, so that the latter would not be protected. The liquid rises along the walls of the drum and is thrown over the rim into the outer casing, the internal surface of which is also made of stoneware. The top of this casing contains an opening for filling and lateral pipes for sucking off acid vapors. The many small holes in the walls of the drum do not pass through to the outside, but each vertical row of these perforations connects with a canal inside of the stoneware wall and reaching up to the rim. The canals, like the walls, are nearly vertical, the slope being 10 to 1 or 12 to 1, so that at speeds above 150 r. p. m. of liquid is thrown out of the canals at their upper end.

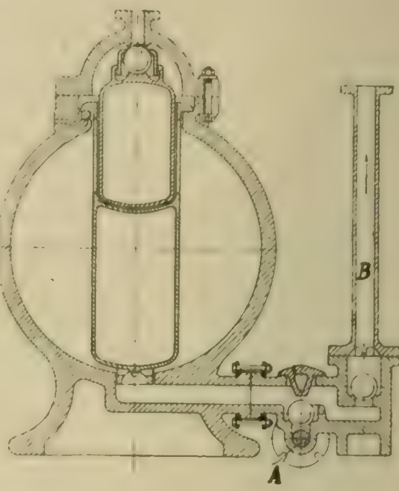


FIG. 5.—MONTEJUS APPARATUS.

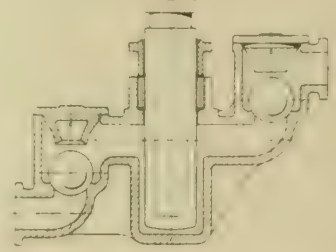


FIG. 6.—PISTON AND PLUNGER PUMP.

Fig. 4 shows the Guttman exhauster, which is specially adapted for sucking off gases with steam or compressed air. These exhausters are made for a tube width of 50 to 450 mm.

Fig. 5 shows the "Montejus" apparatus in which the liquid is driven out of a closed vessel by means of compressed air into the tube B and thus raised. After the closed vessel has been emptied the air valve is turned off and the enclosed air escapes, whereby the vessel is filled again with a liquid from a higher reservoir through a tube with a suitable valve. The liquid enters through A. Fig. 5 shows an improved construc-

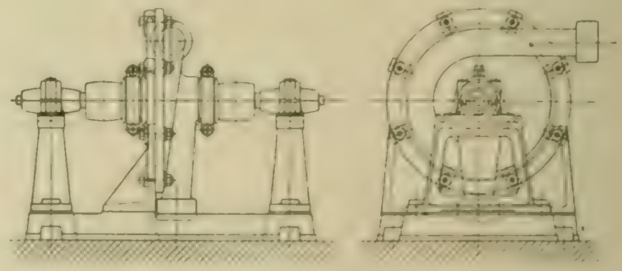
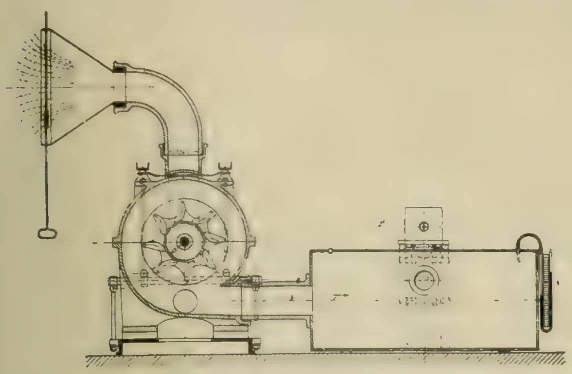


FIG. 7.—CENTRIFUGAL PUMP.

tion operating with the expansion of the compressed air in order to utilize perfectly its capacity of work. The opening and closing of valves is, of course, performed automatically.

Fig. 6 shows a piston and plunger pump with iron frame and gearing. The valves are polished stoneware balls, which are covered in some cases with rubber. Rubber rings, cotton impregnated with paraffin or asbestos are used for packing.



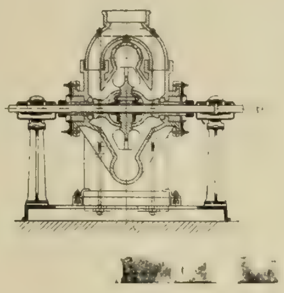


FIG. 8.—STONEWARE EXHAUSTER.

Fig. 7 shows a centrifugal pump; the width of the pipe is 50 mm., the diameter of the fan 300 mm. At 1000 r. p. m. it raises per hour 12 cubic meters to a height of 10 meters. Some tests made with this pump are discussed by Prof. Lindner at some length on the basis of mathematical analysis.

Fig. 8 shows a stoneware exhauster running at a circumferential speed as high as 35 to 45 meters per second. The capacity of these stoneware exhausters are very satisfactory. They are built in sizes of 300, 200 and 150-mm. width of tube. The gases have a normal speed of 10 to 20 meters per second. The maximum effect is therefore for the three sizes 80, 40 and 20 cubic meters per minute, respectively. For more special details reference may be made to *Zeit. f. Angewandte Chemie*, 1903, No. 49, and 1905, No. 6. The box II. at the right-hand of the left-hand diagram of Fig 8 was used in the tests of the author.

The Deutsche Steinzeugwarenfabrik für Kanalisation und Chemische Industrie, in Friedrichsfeld i/B., which make these stoneware apparatus, is represented in this country by Mr. Frederick Bertuch & Co., of New York City.

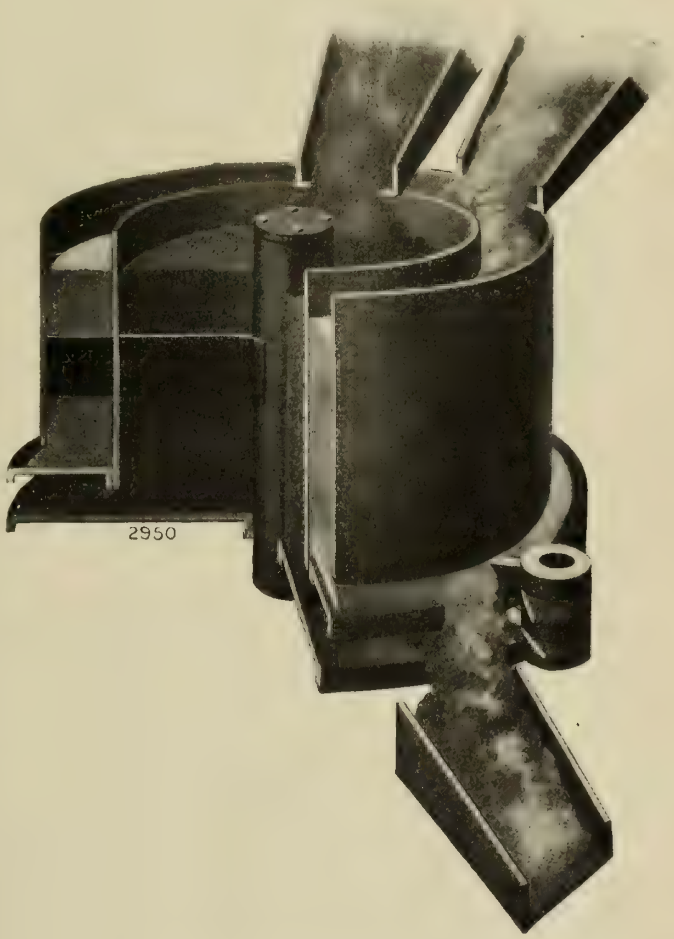
A New Machine for Measuring Materials.

We have repeatedly referred in our columns to the problem of continuous versus intermittent operation in metallurgical and chemical practice, and have pointed out the natural tendency of using continuous processes wherever the object is to push the production to a maximum.

Where materials are to be fed to metallurgical apparatus, to kilns or furnaces, the intermittent introduction from weighing machines, cuts down the capacity of the apparatus to considerable extent. However, weighing can only be done intermittently, so that if the materials fed into the apparatus are measured by weighing them, the feed is necessarily intermittent. Moreover, while scales accurate in themselves can readily be obtained, yet where the scale hoppers are fed from overhead hoppers, the cut-off gates often introduce error through failures to stop the flow in time, particularly with fine materials. If men close the gates by hand they sometimes get careless and inattentive. If the scales operate the gates automatically, they do not begin to operate until the scale beam tips, and all material entering between that time and the complete closing of the gates is in excess of that actually weighed.

The Trump measuring machine, which will be described in this note, is an accurate means of determining the mass of the material by volume rather than by weight.

In measuring granular materials or powders allowances must, of course, be made for the proportion of voids, and it would seem at first sight that this might be a considerable source of error. Materials produced by different companies may vary in the proportion of voids, as they are prepared by different methods and different machinery, so that a cubic foot of clay, for example, may mean anywhere from 80 to 100 pounds, depending on what preparation it has received. Clay dug from the same bank and crushed in the same crushers will weigh the same now, 5 minutes, an hour, day or month hence, the only variation being that due to the slow wear of the machinery which prepares the substance. This is the condition obtained in most factories, so that the proportion of voids can be easily ascertained and allowed for. The variation due to wear is slow and slight, and is corrected by tests at infrequent intervals. Measuring, therefore, is more accurate than weighing where each batch weighed may be different, provided that the measuring apparatus does its work accurately.



MEASURING AND MIXING MACHINE.

Before the invention of the Trump measuring machine, measurement was not considered sufficiently accurate, principally because of inaccuracies introduced by the measuring device itself.

Rolls with pockets often did not clear themselves perfectly, and [illegible] were affected by the depth of material in the overhead hopper, which caused a variable density.

Screw feeds wore to smaller diameters very quickly with gritty substances, and so decreased the amounts fed out. With fine, powdery materials they would sometimes flush through, regardless of the screw, and sometimes hang back. Even at its best the screw delivered pulsating feed.

Years ago a device was invented consisting of a circular revolving table, on which was a conical pile of material to be measured, this pile being constantly replenished from a stationary spout some distance above. The measuring part was a stationary blade extending diagonally into the pile near the table.

It was only partially successful for two reasons:

First, the spout and the material in it being stationary, and the table with the material on it revolving, there was a zone where it was neither quite stationary nor yet moving quite at the speed of the table. The zone was not fixed in its position, but raised and lowered according to the varying consistency of the materials, dampness, etc. Consequently, the spout had to be some distance above the table to insure that the zone never was as low as the cutting blade.

Secondly, the spout being such a distance above, left a long slope of the material unsupported. Accidental jar, dampness, etc., would cause some little variation in the slope. Very slight changes would cause considerable variation in the diameter at the bottom of the pile, because of the length of the slope. This meant a corresponding variation of the distance which the knife extended into the pile, and consequently serious inaccuracies in the result.

The device, however, was automatic, continuous, cheap to operate, and satisfactory when great accuracy was not desired, or where the materials were not too fine.

The Trump measuring machine operates on the same principle of the revolving table and stationary knife, but the cylindrical supply spout is made to revolve with the table. There is, consequently, no zone of irregular motion, and the cylinder can therefore be brought down close to the upper edge of the knife, reducing the unsupported conical slope to a minimum, so that changes in the angle have little effect on the total diameter of the pile. The material being at rest at all times also reduces the liability of the slope to change.

The cylinder is also made larger, nearly the diameter of the table, so that the knife can cut deeper; and of the amount cut off, the slope is a very small part. Repeated tests show that the Trump machine has worked well within a ½ per cent variation—more accurate than scales in the same situation, and cheaper to operate.

The cutting blades, or "knives," are cast with a bell crank lever, which connects with a micrometer screw adjustment, so that the amount pared off per revolution can be varied at will. A dust-proof casing surrounds the measuring cylinders, to prevent loss of the finer materials or pollution of the air.

When more than one material is to be measured, a table and cylinder are provided for each material—the tables are superposed and the cylinders placed one within the other, as shown in the adjoining diagram.

These machines were designed for use at the works of the Solvay Process Co., by Mr. E. N. Trump, chief engineer of that company. They were so successful that Mr. Trump protected his rights by patent, and arranged with the Link-Belt Engineering Co., of Philadelphia, for the manufacture of them.

The machines have been with perfect success, for such varying purposes as the feeding of kilns, the mixture of chemicals, manufacture of coke briquettes, feeding of rocks, making of concrete, etc., and materials have been handled with it of 6-inch cube dimensions to the finest of powders, both wet and dry.

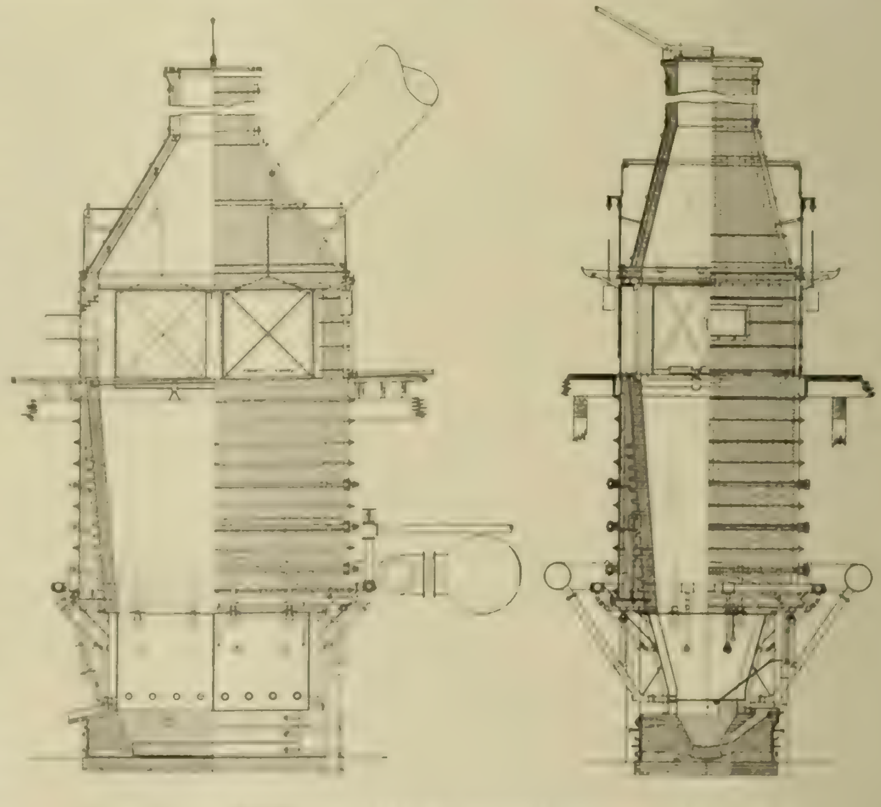
Blast Furnaces for Lead and Copper Smelting.

Smelting is an exact science. Given a certain ore with proper fluxes and proper attendance, it is possible to predict almost exactly the result that will be obtained in a smelting furnace of modern construction. The modern smelting furnace is the result of a process of evolution. It has been developed by the coöperation of the metallurgists at the smelting plant with the manufacturers of smelting furnaces.

The following notes describe the latest types of blast furnaces, both for lead and for copper smelting, made by the Allis-Chalmers Co., as the result of some forty years experience as manufacturers of machinery for the mining and reduction of ores. Small furnaces are made round and will not be considered in the following. Large furnaces are built rectangular in shape. All the dimensions given in the following are measurements made inside the jackets of the tuyere line.

BLAST FURNACE FOR LEAD SMELTING.

The general design of the modern 44-inch x 144-inch steel water-jacketed lead furnace, made by the Allis-Chalmers Co., is shown in Figs. 1 and 2, which give a half section and end elevation. The curbs of these furnaces are of steel or cast-iron plates, resting upon a steel-bottom plate, and very strongly bound on the outside. The jackets are made of the best flange steel, having either welded or riveted joints, and are braced between the front and back sheets by our special steel stays.



FIGS. 1 AND 2.—RECTANGULAR STEEL WATER JACKETED LEAD FURNACE. 44" x 144".

No rivet heads appear anywhere on the front sheets. All jackets are provided at their sides with heavy steel lugs and bolts for binding them to the adjoining jackets, and the entire set of jackets is bound together by strong steel binders encircling the jackets, or by jackscrews, resting against the mantle frames or a binding frame provided for that purpose, encircling the furnace outside of the main columns.

Suitable tuyere openings are provided in the side jackets near the bottom of the jacket. Tuyeres are seldom placed in the end jackets of modern lead furnaces. Tuyeres of the usual lead furnace type, such as those shown in Figs. 1 and 2, are connected with the bustle pipe by flexible canvas tubing or, if desired, rigid tuyere boxes connected to the bustle pipe by light steel tubing are provided. The bustle pipe is built of heavy galvanized steel. It is provided with a main flanged nozzle for connecting it to the main blast pipe, and with connections for the tuyere pipes, and is supported from the mantle frame or furnace shaft.

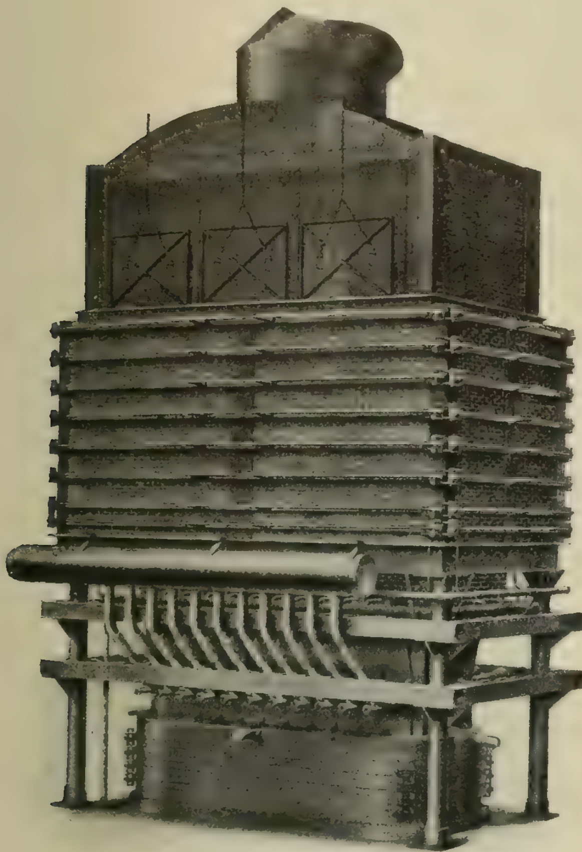


FIG. 3.—RECTANGULAR CAST IRON WATER-JACKETED LEAD FURNACE, 44" x 180".

A main supply pipe, for cooling water for the jackets, encircles the furnace above the jackets, and is provided with all necessary connections to the jackets. Suitable means are also provided for caring for discharge water from the jackets.

The brick shaft above the jackets rests upon a mantle frame built up either of heavy cast-iron sections or of "I" beams, as desired. This is carried by columns at the corners of the furnace. The shaft is bound on outside by heavy corner binders, tied to the opposite corner binders by heavy steel rods. When the furnace is of the "flush-top type" the shaft terminates at the charging floor, the top being covered by cast-iron or steel plates with suitable openings provided in it for charging the furnace, and a connection for the downtake just below the charging floor. When the furnace is of the hood type, as shown in Figs. 1 and 2, the shaft extends 6 to 10 feet above the charging floor level, and is covered by a brick or sheet steel hood, terminating in a brick or sheet steel stack extending up through the roof of the building, with the downtake connecting at a suitable place in the hood or stack. In this style of furnace, charging openings are provided at the sides of the shaft at the feed floor level. These openings are provided with counterweighted steel doors. The stacks of these furnaces are provided with dampers, which are kept closed, except when the furnaces are being blown in, or out, of service.

Cast-iron jackets are preferred by many metallurgists for lead furnaces. Fig. 3 shows a 44-inch x 180-inch modern lead furnace, having cast-iron jackets.

Steel-plate jackets are used on account of their great durability, and the relief they afford from the cracks, blow-holes, shut-downs and complicated construction common to cast-iron jackets. The life of a steel-plate jacket is almost entirely dependent upon the care it receives. Given sufficient cooling water and cleaning from scale, and it will last for years. On the other hand, a cast-iron jacket is liable to crack the first hour it is in service, no matter how well it may be made or cared for, and, at best, its life is very short compared to that of a steel-plate jacket. In addition, as cast-iron jackets are necessarily made in small sections many more of them are required for a furnace of a given size than of steel-plate jackets, hence a more complicated construction and more attenion required in their erection and operation.

The capacity of a lead furnace depends chiefly upon the character of the ores treated, although much also depends upon the man in charge and upon the design of the furnace itself. The furnaces of the Allis-Chalmers Co. are rated ordinarily at a capacity of about 3 1-3 tons of charge per 24 hours per square foot of area at tuyere level; that is, a 36-inch x 120-inch furnace is rated at 100 tons capacity per 24 hours. This capacity is generally exceeded, especially by large furnaces.

The amount of fuel required also depends upon the above-mentioned conditions, and also varies materially with the character of the fuel itself. In small lead furnaces using coke as much as 20 per cent of the weight of the charge may be necessary, while with large furnaces as little as 10 per cent is sufficient. A general average may be assumed, however, at from 12 per cent to 14 per cent of the charge. When charcoal is used, a somewhat larger quantity will be required.

Coke is the most satisfactory fuel for blast furnaces ,as owing to the strength of its structure, it does not readily crush under the weight of the charge in furnace. Charcoal, however, is very satisfactory for small furnaces.

Blast Furnace for Copper Smelting.

Blast furnaces for smelting copper ores are similar in general form to those used for lead smelting, but in detail they differ from them materially. The internal crucible is either shallow or is dispensed with entirely, as in the case of matting furnaces, where the molten materials pass at once from the furnace into an external crucible known as the "settler," where the separation of the matte and slag takes place. The jackets are much higher than in lead furnaces, extending in many furnaces up to the charging floor. In this case they are made in two tiers or sections. Invariably the water-jackets are made of steel plates. The tuyere boxes are of the rigid type and connected with the bustle pipe by light steel tubing. The mantle frame and supporting columns and the structure above the mantle frame are much the same as for lead furnaces. Large copper furnaces are built rectangular in shape; small furnaces round.

Fig. 4 illustrates the standard design of the Allis-Chalmers Co. for rectangular matting furnaces up to 42 inches x 120 inches in size. The bottom of this furnace is formed by a heavy cast-iron plate resting upon jackscrews, which are carried upon a steel truck mounted upon flanged wheels. In many furnaces the truck is omitted, the jackscrew bases resting upon the furnace foundation. The jackets rest directly upon the bottom plate, but are hung from the mantle frame by suitable hangers, so that when the bottom plate is lowered the jackets remain in position suspended from the mantle frame. The jackets are made in sections, of steel plate with riveted joints, and are stayed between the front and back sheets by improved steel stays. No rivet heads appear anywhere on the front sheets of the jacket. One side of the jacket is provided with a recess at the base for a rectangular tapping jacket, used for draining the furnace when blowing out of service. One end jacket is provided at its base with a recess for inserting a rectangular breast jacket, in front of which sets the trap-spout and through which the molten materials flow to the settler. The Allis-Chalmers Co. usually makes the side tapping jacket of cast iron. As this is not used often it does not require a more durable or more expensive material. The breast jacket and trap spout are made of either a special bronze mixture or of steel plate, as they are constantly in use and require very durable materials to give satisfactory service. All jackets are provided at their edges with strong lugs and bolts for securing them to adjoining jackets, and the entire set are bound by steel "I"-beam binders on each side and end of the furnace, held together at the corners by strong "U" bolts. All jackets are provided with inlet and outlet connections for cooling water, hand-holes at the base for cleaning and all other fixtures and fittings complete.

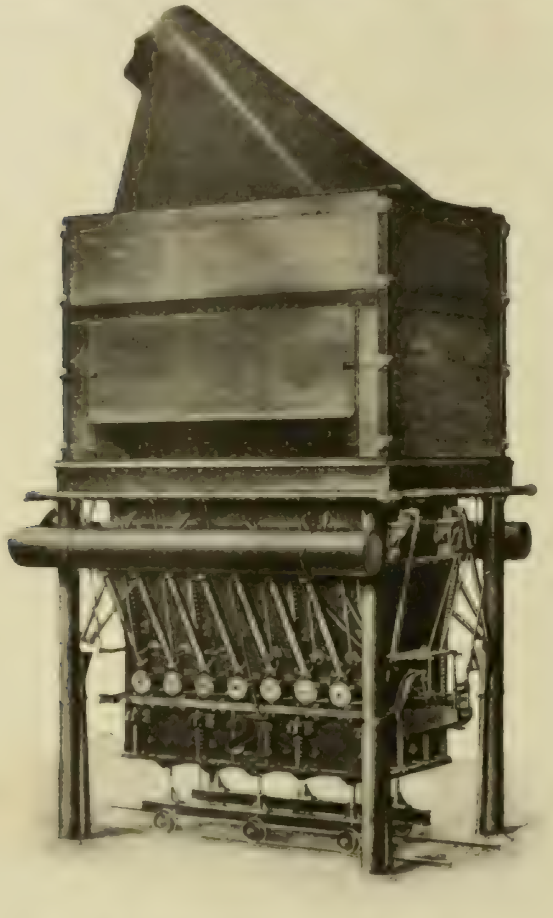


FIG. 4.—COPPER BLAST FURNACE, 42" X 120"

The tuyere boxes are bolted rigidly to the back of the jackets and connected to the bustle pipe by light steel tubing. The bustle pipe is of steel plate, provided with a main flanged nozzle for connecting it with the main blast pipe, and suitable nozzles on each leg for connecting the tuyere pipes.

The mantle frame is of very heavy steel "I", beams, and rests upon four cast-iron or steel columns at the corners of the furnace. A cast-iron plate rests upon the middle frame, forming a base for the brick work of the shaft. The furnace shaft is made of good common brick and lined on the inside with 9 inches or more of firebrick. At the charging floor level openings are provided in each side of the shaft, the full length of the shaft, for charging the furnace. The bottom and sides of these openings are protected by heavy cast-iron plates and the top of the opening is formed by a steel "I"-beam lintel frame, which carries the brick work of the shaft above the charging openings. Stationary light steel curtains from the lintel frame are provided to cover the charging opening to within 12 to 18 inches of the floor level, leaving just opening enough below the curtain to shovel the charge into the furnace. These curtains can be easily removed for barring down, furnace inspection or other purposes. The top of the furnace shaft is covered by a pyramidal shaped steel hood terminating in a connection for a round steel downtake or stack.

Fig. 5 shows the standard design of the Allis-Chalmers Co. rectangular copper furnaces of the larger sizes. These furnaces are much the same as those shown in Fig. 4, excepting that they have a second tier of jackets, directly above the lower jackets, which extend almost up to the feed floor.

Modern copper blast furnaces attain, on an average, a capacity of from 5 to 6 tons of charge per 24 hours per square foot of area at the tuyere level. Many furnaces greatly exceed this capacity, and this rating is only given for approximate calculations.

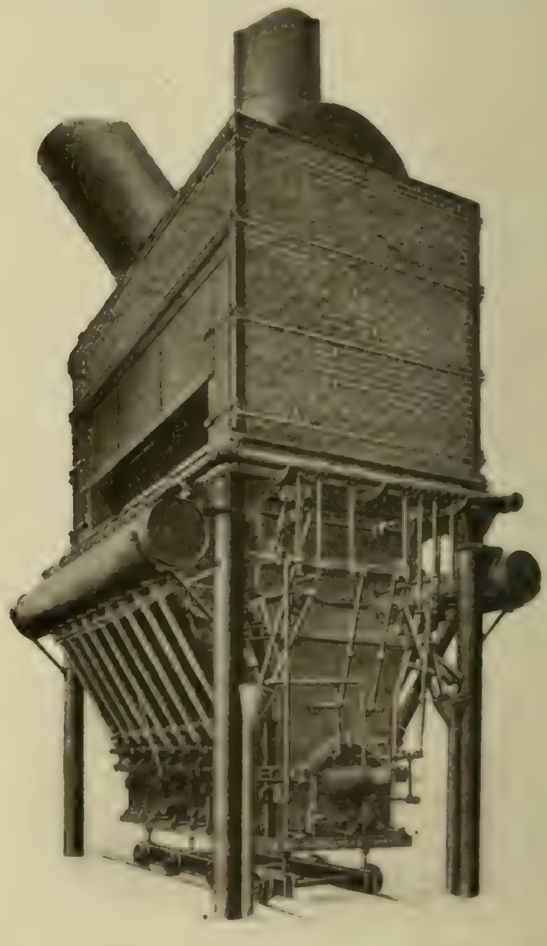


FIG. 5.—WATER JACKETED COPPER FURNACE, 44" X 160"

Coke is, of course, the most desirable fuel for copper furnaces, although charcoal is frequently used for small furnaces. The quantity of fuel required varies, according to conditions, from 7 per cent to 8 per cent of the weight of the charge in large furnaces handling heavy sulphides up to 16 per cent to 18 per cent in small furnaces treating oxide and carbonate ores. This is, of course, using a cold blast. When smelting sulphide ores with a host blast, as low as 2 per cent of coke has been used with satisfactory results.

News and Notes.

American Electrochemical Society.—The next meeting of the Society will be held in Ithaca, N. Y., on May 1, 2, 3, 1906. At the meeting of the Board of Directors, held in Philadelphia in Oct. 7, the following gentlemen were elected members of the Society: Frank T. Colcord, Maurer, N. J.; William F. Oesterle, Waukegan, Ill.; Coloman E. Nutter, Topeka, Kan.; John Meyer, Philadelphia, Pa; Frank K. Cameron, Washington, D. C.. At the next meeting of the Board of Directors, to be held on Nov. 4, the names of the following gentlemen will come up for election to membership: Leonard Waldo, Plainfield, N. J.; W. B. Jadden, Marion, Ind.; William Smith, Philadelphia, Pa.; Carl P. Goepel, New York City.

New York Section Society of Chemical Industry.—The first meeting of the season was held at the Chemists' Club on Oct. 20, the topics of the evening being an account of the annual meeting in London, and of the visits to the English sections of the Society, which have already been reported in this journal. Dr. Parker was presented with a gold watch and fob in recognition of his active work in connection with this trip.

New York Electrical Society.—On the evening of Oct. 23, Mr. Peter Cooper Hewitt lectured before the New York Electrical Society on electrical reactions of gases, with particular reference to the phenomena found in the Cooper Hewitt mercury vapor lamp and rectifier.

Sixth International Congress of Applied Chemistry.—The sixth International Congress of Applied Chemistry will be held in Rome in the spring of 1906. The president of the committee of organization is Prof. Emanuele Paterno, Via Panisperna, Rome, and the secretary, Prof. Vittorio Villavecchia, Central Customs Laboratory, Rome. The subscription fee for active membership is 20 francs, and is to be remitted to the secretary before the opening of the Congress or during its session. The Congress is divided into the following eleven sections: 1, analytical chemistry, apparatus and instruments; 2, inorganic chemistry and industries relating thereto; 3, metallury and mining and explosives; 4, organic chemistry and industries of organic compounds and coloring matters; 5, technology and chemistry of sugar; 6 fermentation and starch; 7, agricultural chemistry; 8, hygiene and medical chemistry, pharmaceutical chemistry, bromatology; 9, photochemistry; 10, electrochemistry, physical chemistry; 11, laws, political economy and legislation in relation to industrial chemistry. Dr. H. W. Wiley, chief of Bureau of Chemistry, Department of Agriculture, Washington, D. C., is the chairman of the American committee. Intending members of the Congress may send their names and dues directly to the secretary of the Congress, together with titles of papers to be presented at the meeting, or if preferred, to the chairman of the American committee, who will undertake to forward names, dues and papers to Rome. In case the dues are sent first to Washington for transmission to Rome, a check for $4.00 should be sent to cover dues, exchange and postage.

Western Electrical Directory.—We have received the 1905 edition of this useful directory, published by the Blanchfield Publishing Co., Rialto Building, San Francisco, Cal. The directory contains a list of the central stations in the following Western States: Alaska, Arizona, California, Colorado, Idaho, Montana, Neveda, New Mexico, Oregon, Utah, Washington, Wyoming, El Paso, Texas, Territory of Hawaii, British Columbia, Yukon Territory, Alberta, Saskatchewan, Assiniboia and a portion of Mexico. States and cities or towns are arranged alphabetically. Following the name of the city the population is given, then the name of the operating company and the names of its officers, capital stock and equipment of plant. The price of the directory, including quarterly reports until the 1906 issue, is $5.00.

Diary.—We have received from Messrs. Geo. G. Blackwell, Sons & Co., a very handsome pocket diary, bound in red morocco. This is the pioneer firm of ferroalloys in Great Britain, and was founded in 1869.

Storage Batteries.—A small folder, recently issued by the Westinghouse Machine Company, deals with the use of storage batteries for electric signal systems. It is chiefly interesting as the first publication of the Westinghouse Machine Company on their storage batteries.

Siloxicon Articles.—According to the Niagara Falls "Gazette" of Oct. 2, "a big new industry for Niagara Falls is forecasted in the form of the Acheson Siloxicon Articles Co. It was recently incorporated in the State of Maine, with a capital of $500,000. The location of the factory will be at Niagara Falls. The company will utilize crude siloxicon and make from it such articles as crucibles, muffles, brick, etc. Siloxicon, as is well known, is a highly refractory material, and it is believed from the present demand for it that it will supersede many of the older forms of refractory materials." Siloxicon is really a series of silicon-carbon-oxygen compounds, for the production of which Mr. Acheson designed an electric furnace with very exact temperature regulation (see our Vol. I., p. 287). An article on the oxidation of siloxicon from the pen of Mr. E. G. Acheson himself may be found on page 373 of our Vol. I., while the use of silico-carbides as refractory material was discussed in an article by Mr. F. A. J. FitzGerald, our Vol. II., p. 442.

Liquid Barretter Patent.—The Fessenden patent for a "liquid barretter" as receiver in wireless telegraphy has recently been adjudicated and sustained, the decision being rendered by Judge Hoyt H. Wheeler. The patent is interesting not only account of the high efficiency of the device, but because it is probably the only application of electrochemistry in wireless telegraphy. The liquid barretter consists essentially of a cell comprising a suitable liquid and suitable electrodes, one of them, usually the anode, having only a minute surface exposed to the liquid; the wireless signals are caused by an increase of the current in the battery circuit in response to the action of received oscillations which pass through the electrodes and cell in parallel with the battery circuit.

Electric Steel Furnace.—According to a report from San Francisco, 1 ton of black beach sand from Guadalupe, Cal., has been treated in Portland, Ore., and over 600 pounds of steel produced therefrom by the use of an electric smelting furnace after concentrating. A small quantity of sand from A. B. Hammond's property, on the Oregon coast near the mouth of the Columbia River, was also put through the process of separating the magnetite from the other minerals. The work was done at the pavilion of the Geological Survey on the grounds of the Lewis and Clark Exposition, under the direction of Mr. David T. Day, chief of the Division of Mining and Mineral Resources. Mr. C. E. Wilson, of Virginia, had charge of arranging the electrical details. A larger electric furnace will be constructed, and Dr. Day will continue his tests until the value of the black sand of the coast has been determined. The most practicable method of making pig iron and steel commercially will be ascertained. The sand can be obtained at several hundred places along the Pacific Coast. Some gold is recovered by the new process.

The Electric Furnace in Metallurgy.—Under this title, Mr. M. Robert Pitaval presented a suggestive paper before the recent International Mining Congress in Liege. "At the World's Fair in Paris in 1900, two or three electric furnaces of small dimensions could be seen. * * * These were curiosities of the laboratory. Now, at Liege, we see in the electrical and metallurgical sections mighty ingots of steel, blocks of ferrosilicon, ferrochrome and other alloys, produced on a large scale in the electric furnace and sold in quantities of many tons to metallurgical works." The paper gives a full review of what has been done in electric steel refining. "The temperature of at least 2000° C. which is obtained in the electric furnace—i. e., 800° C. above that of the open-hearth furnace— permits a very complete deoxidation of the steel, almost perfect removal of sulphur and phosphorus; this high temperature removes any danger that the metal might over-oxidize and become pasty, as is the case if the refining process is prolongated too long in the ordinary metallurgical furnace. Since this high temperature is produced within the mass of the metal itself and not on the surface, as in the open-hearth furnace, there is no possibility of portions of slag intermixed with the metal. Finally, the neutral atmosphere on the surface of the bath in the electric furnace permits the use of very basic slags and fluxes which could not be used in the open-hearth furnace. At the end of the refining process it is thus possible to use a slag by which the latest traces of phosphorus are eliminated." The electric furnace permits an easy and cheap refining of common steel. "L'acier a tué le fer, mais le four élictrique tuera l'acier ordinaire."

Centrifugal Filter.—For the purpose of using lubricating oil over and over again, the oil must be filtered and cleaned after use. A centrifugal filter has been used for this purpose in various forms in this country. In a consular report of Oct. 7 an apparatus of Heine Brothers, in Viersen, Rhineland, Germany, is described. "This oil filter consists of a perforated cylinder-shaped drum, in which the concrete mass is placed. Only 5 minutes' time is necessary by centrifugal force to drive every particle of mucous oil out of the mass placed in the cylinder. This oil is drained into a separate vase, from which it can be removed ready for use. There is a crane attachment, which enables the perforated cylinder to be easily emptied and cleaned."

Lead-Lined Iron Pipes.—We have received the catalogue of the Lead-Lined Iron Pipe Co., of Wakefield, Mass. manufacturers of lead-lined and tin-lined iron pipes. The company was founded some fourteen years ago, and its business has steadily grown, not only in connection with the better recognition of the importance of lined pipes for water-works, but as a result of the marvelous growth of chemical and metallurgical industries in this country. In the latter case lead-lined pipes are an economical necessity wherever acids are to be handled. Acids corrode all kinds of iron pipe, brass pipe and copper, but do not affect lead pipe, provided it is pure lead. The joints on lead-lined iron pipe for acids or chemicals are flanged, thereby ensuring a perfectly tight joint. For hot acids a hardened lead is used. The Anaconda, Amalgamated and United Verde copper mining companies are using this lead-lined pipe where there are acid corrosive properties in the water. The Lead-Lined Iron Pipe Co. received a gold medal at the St. Louis World's Fair.

Assayers' and Chemists' Supplies.—We have received the 1905 catalogue of assayers' and chemists' supplies of the Denver Fire Clay Co., in Denver, Col. The great variety of the laboratory supplies covered in this illustrated catalogue is well-known to assayers and chemists, who often use it as a reference book. Among the novelties in the new edition may be mentioned crucible and muffle gasoline furnaces, now manufactured by this company.

Hancock Jig.—Bulletin 1403 of the Allis-Chalmers Co. deals with the Hancock jig which is of Australian origin, and was invented by Mr. H. R. Hancock, then general manager of the Moonta mines in South Australia. It has proven very successful in treating the low-grade sulphide ores of the Broken Hill district. The present pamphlet gives a fully illustrated description of the jig and the results obtained with the first jig installed in the mill of the Arizona Copper Co., and confirmed by working tests made at other plants. The characteristic features of the Hancock jig are stated to be a very large production; a marked ability to handle an unsized feed (the jig is now opearting in this country on coarse material which averages from ½ inch down to 3 m/m and from m/m to 40-mesh material); small amount of water required in the jigging operation (at least 50 per cent less than in the plunger jig); the jig will handle 100 tons per day as well as 500 to 600 tons.

Cranes.—Section E of the serial bulletins of the Wellman-Seaver-Morgan Co., of Cleveland, on iron and steel works equipments deals with cranes, and gives profusely illustrated descriptions of various types of cranes built by that company. The first part of the bulletin describes details of the Wellman patented double-trolley ladle cranes, which are particularly adapted for steel plant practice, both for handling hot metal to and from the mixers and charging it into the furnaces, and for handling molten steel from the furnaces. The second part of the bulletin deals with standard electric overhead traveling cranes for general purposes, followed by notes on standard locomotive cranes and gantry and yard cranes. The Wellman-Seaver-Morgan Co. also build special purpose cranes for all classes of crane service, including charging and manipulating cranes for steel plant work, ingot extracting cranes, soaking pit cranes, etc.

Saw. Mill Carriages.—Catalogue No. 123 of the Allis-Chalmers Co. deals with saw mill carriages and accessory machinery (Pacific Coast power set works, Pacific Coast carriages, reliance carriages, blocks, knees, spring cushion taper movement, dogs, set works, automatic spring cushion offset, carriage tracks, couplings, buffers). In the manufacture of head blocks and set works, and in fact all their machinery, the Allis-Chalmers Co. use special tools which, together with their system of making every part and piece to gauges or template, insures perfect fitting and perfect work. Mill men, especially mill foremen, will appreciate this system of doing work, as it is a guarantee that when a piece is ordered to replace a worn or broken part it will go to its place without filing or fitting. The catalogue mentioned above describes in detail the various parts that make up the reliance carriages of the Allis-Chalmers Co., and is profusely illustrated.

Rotary Converters.—Special publication 7038 of the Westinghouse Electric & Mfg. Co. deals with rotary converters, their characteristics and construction, with instructions for their erection, operation and care. As the connecting link between alternating and direct-current systems, the rotary converter has naturally found the largest application in sub-stations for direct-current street railway supply in connection with high-tension alternating-current transmission. But its field of usefulness is much wider. Alternating current is now considered "the" system for transmission and distribution over large areas. Wherever at a customer's premises direct current is preferable (as for certain power purposes) or absolutely necessary (as for electrolytic work), the rotary converter steps in. Electrochemical and metallurgical engineers who want to become acquainted with its peculiarities will find a great amount of clean-cut, useful, practical information in this publication of the Westinghouse Co.

Steel and Iron Hardening Metals.—The United States Geological Survey has recently issued the report by Mr.

Joseph Hyde Pratt on the production of steel and iron hardening metals in 1904 (including nickel and cobalt, chromium, tungsten, molybdenum, vanadium, titanium and uranium). The introduction deals with the production of pure metals free from carbon, and discusses the aluminothermic method of Goldschmidt, the electric furnace method of Rossi, with aluminium as reducing agent, and the method of Greene and Wahl. The different metals mentioned above are then discussed in detail with respect to occurrences, localities and production in 1904. The report contains much useful information, especially on ferrochrome and ferrotungsten.

Commercial Examination of Fuel.—A small pamphlet with this title, issued by Messrs. Waller & Renaud, 159 Front Street, New York City, gives a popular and very clear account of the importance of testing fuel before use in commercial plants and of the methods used in such tests. Not only is the heating power of the fuel in B. T. U. per pound to be determined, but since the latter is always determined on a dried sample, it is necessary to know the percentage of moisture in the fuel to make the required correction. The approximate analysis should give the percentages of moisture, volatile combustible matter, fixed carbon and ash. To this is usually added a determination of the sulphur, since when much sulphur is present, the coal will clinker badly. Messrs. Waller & Renaud have equipped their laboratory with a Berthelot-Mahler-Atwater standard bomb calorimeter, and are prepared to make accurate calorimeter tests of coal and other fuels, as well as the usual approximate analyses, sulphur determinations, analyses of ash, etc.

Producer Gas.—A neatly illustrated pamphlet of the Morgan Construction Co., of Worcester, Mass., gives a full description of the Morgan continuous gas producer. This company has developed this new producer in connection with the introduction of continuous heating furnaces to supply hot billets to their continuous rolling mills, since it soon became evident that the success of these special furnaces depended largely upon securing a gas of uniform quality and in uniform quantity. The advantages of the Morgan continuous gas producer are said to be as follows: Complete gasification (the cold ashes containing usually less than 0.5 per cent of the carbon in the original coal); the production of a gas containing the lowest possible percentage of nitrogen and carbonic acid (usually not more than 48 per cent N and 4 per cent CO_2); the delivery of the gas uniform both as to quality and quantity; the automatic disintegration of the clinkers; continuous operation without periodical stoppages for cleaning out; reduction of manual labor to a minimum.

Steam Turbines.—The growing commercial importance of steam turbines is indicated by the following note on one week's business of the Westinghouse Co. in steam turbines. During the week ending Sept. 29, 1905, the Westinghouse companies received orders from the Toledo Gas, Electric & Heating Co., Toledo, Ohio, for two 1000-kw. turbogenerator sets, the Pennsylvania Railroad Co. for four 500-kw. turbogenerator sets, the Water, Light & Gas Co., of Hutchinson, Kan., for two 500-kw. turbogenerator sets, and the Solvay Process Co., of Syracuse, N. Y., for one 500-kw. turbogenerator set. Each turbine will be of the type known as the multiple-expansion parallel flow, with an overhead capacity when running condensing of at least 50 per cent, and the alternating-current generators will be of the turbo-type with rotating fields. The 500-kw steam turbine for the Solvay Process Co. will operate with dry, saturated steam at the throttle of 125-pounds gauge pressure and 28-inch vacuum. The 60-cycle direct-connected turbogenerator will deliver two-phase current at 440 volts. This company already has in operation a number of Westinghouse steam and gas engines, in addition to 3500 hp. in Roney mechanical stokers. Westinghouse apparatus is also installed in the Detroit plant of this company.

Insulating Shellac.—The "Agate Shellac" for insulating purposes, which was noticed in these columns some time ago, has received the highest award in its class at the International Exhibition in Liege, namely, one gold medal in the third class and one silver medal in the second class. Mr. Felix Hamburger, of 90 William Street, New York City, is the representative of the manufacturers of agate shellac in this country.

Personal.

Mr. Herman A. Metz, the well-known importer of chemicals and dyestuffs, has been nominated for the position of Controller of New York City on the regular Democratic ticket. The Chemists' Club, of which Mr. Metz is a trustee, tendered him a reception on the evening of Oct. 28. Mr. Metz has a host of friends among the chemists of this country, who, independent of party affiliations, heartily wish him the fullest measure of success.

Mr. Herbert Haas has opened an office as consulting metallurgical engineer at 218 California Street, San Francisco, Cal. Mr. Haas gained his technical experience as chief chemist and superintendent of leading silver-lead and copper custom smelters in the United States and Mexico, also as superintendent of construction of smelting plants. A paper by Mr. Haas was published on page 101 of our present volume. Mr. Haas will make a specialty of the metallurgy of copper and lead, the design and erections of copper and lead reduction works along modern and approved lines, and will give special attention to pyritic smelting.

Dr. Karl Goldschmidt is at present in this country in the interest of the rapidly expanding business of the Goldschmidt Thermit Co., which is the American branch of the chemical and tin smelting works of Theodor Goldschmidt, Essen-Ruhr, Germany. The two brothers, Dr. Karl and Dr. Hans Goldschmidt, are the owners of this firm, founded in 1847, and widely known through its effective pioneer work in electrolytic detinning, and in recent years through Dr. Hans Goldschmidt's celebrated aluminothermic process. Dr. Karl Goldschmidt intends to sail for Germany on Nov. 7, on the "Kaiser Wilhelm II."

Mr. George G. Blackwell, head of the firm of George G. Blackwell, Sons & Co., Ltd., of Liverpool, manufacturers and dealers in ferro-alloys, rare ores and supplies for chemical and metallurgical manufacturing plants is making his annual visit to this country. He makes an extended trip throughout the East and Middle West and to the principal cities in Canada, and will sail early in December for Liverpool. This concern is one of the oldest in the ferro-alloy business, having been established by Mr. Blackwell himself in 1869.

Dr. P. Heroult, Mr. R. Turnbull and Mr. J. Sejournet left New York on Oct. 18 to start the plant at Sault Ste. Marie, in which, with the coöperation of the Canadian Government and of the Lake Superior Power Co., the use of the electric furnace for the reduction of pig iron from ore will be studied. The electric furnace, which has already been completed, has a capacity of 2 to 4 tons a day. The intention is to carry out the operation in such a way as to oxidize all the fuel, used for reduction, to CO_2 within the furnace, so as to utilize its calorific value to the utmost, water-power being so abundant that its consumption compared with that of fuel is considered secondary. It is suggested to use charcoal and charred peat as fuel,

Digest of U. S. Patents.

Compiled by Byrnes & Townsend, Patent Lawyers, National Union Building, Washington, D. C.

Electric Smelting and Reduction Processes.

(Concluded.)

No. 677,207, June 25, 1901, Charles M. Hall, Niagara Falls, N. Y.

Purifies bauxite and other oxides of aluminium containing as impurities oxides of silicon, iron, titanium, etc., by fusing them with an electric current, with little or no flux, and then reducing the impurities in the fused bath. Example: Bauxite, containing alumina, 60; ferric oxide, 18; silica, 2-3; titanic acid, 3-4, and water, 17, is calcined to remove the water. Before calcination, 5 to 10 per cent of powdered charcoal or coke may be added, to remove the water more completely and partially reduce the iron oxide. The calcined product is then mixed with powdered carbon to give a mixture containing 8 to 10 per cent of carbon and fused in an electric furnace. Less carbon may be added if it be supplied by the carbon of the electrodes and furnace lining. With a furnace of 8 inches internal diameter, a direct or alternating current of 1500 amps. at 28 volts is suitable, effecting the fusion and reduction in 1 hour. The reduced iron, silicon and titanium unite to form a fused alloy, the bulk of which settles to the bottom. The mass is then allowed to cool and the alloy separated, the particles which are distributed through the alumina being picked out by a magnet. Or the molten alumina and alloy may be tapped out. The purified alumina is pulverized and is then suitable for solution in fused baths for the production of aluminium. During the operation the molten alumina is covered with a layer of unfused bauxite, which in turn melts. At the end the bath may be covered with a layer of carbon or purified alumina. The furnace shown is a carbon-lined pot, serving as one electrode, into which depends a carbon-rod electrode. This rod may be raised slightly above the molten material, apparently forming an arc. Two depending electrodes of opposite polarity may be used. For a furnace having a chamber 16 inches in diameter and 24 inches deep, an alternating current of 1500 to 2000 amps. at 60 volts is suitable. Incandescent electric furnaces may also be used, the heat being generated either in the material or in a separate carbon resistance.

No. 677,208, June 25, 1901, Charles M. Hall, Niagara Falls, N. Y.

Purifies bauxite and other oxides of aluminium containing as impurities oxides of silicon, iron, titanium, manganese, vanadium, etc., by mixing with powdered aluminium or ferro-aluminium and melting the mixture. The reduced impurities combine into an alloy. Example: Calcined bauxite, containing alumina, 72.9; silica, 1.3; ferric oxide, 22.7, and titanic acid, 3.1, is mixed with sufficient powdered aluminium or ferro-aluminium to reduce the oxide impurities, the amount of aluminium preferably used being equal to 37.5 per cent of the ferric oxide, 65 per cent of the silica, and 50 per cent of the titanic acid. The mixture is fused in a carbon-lined electric furnace, and is maintained in fusion for one and one-half hours, when the charge contains 300 pounds of bauxite. The reduced metals combine as a molten alloy which generally sinks to the bottom, but may collect in globules or buttons. The heat evolved in the reaction assists in the fusion. The furnace may be a carbon-lined pot constituting one electrode, into which depends a carbon-rod electrode, which is gradually raised to keep it above the surface of the fused mass. As the charge mixture fuses more may be added until the furnace is nearly full, purified alumina being added to cover the mass at the end of the operation. A potential difference of 60 volts, with sufficient amperage to give 150 to 200 hp. has been employed. After the operation the mass in the furnace may be allowed to solidify and the alloy at the bottom detached, or the molten alumina and alloy may be tapped out. The mixture may be fused by making it a resistance conductor, electrolysis then aiding in the reduction. When the bauxite is low in iron, iron may be added, preferably as an alloy with aluminium. For example, calcined white bauxite, containing alumina, 88.53; silica, 4.3; ferric oxide, 1.57, and titanic acid, 3.6, is mixed with powdered 50 per cent aluminium in such amount that the contained aluminium equals 5.8 of the bauxite. Where the bauxite contains much iron, carbon may be added to the charge to assist in the reduction.

No. 681,367, Aug. 27, 1901, Hugh A. Irvine, Niagara Falls, N. Y.

Electrically generated heat for the reduction of ores or compounds by employing a resistor of molten slag. In the furnace shown, the hearth and sides are lined with carbon. Carbon-rod electrodes of opposite polarity depend through the arched roof. In using this furnace for the production of phosphorus, an initial resistor of coarsely-granulated coke is placed on the hearth of the furnace in contact with the lower ends of the electrodes. The charge, *e. g.*, a mixture of phosphates of calcium and aluminium, mix with crushed coke or anthracite coal, and a silicious, basic or silicate flux, is then gradually added and melted. As fusion proceeds, a body of molten slag accumulates on the hearth, and it carries the heating current as soon as it rises in contact with the lower ends of the electrodes. The excess of slag can be removed through tap-holes. The current may be direct or alternating. The carbon lining of the furnace may constitute one electrode.

No. 685,043, Oct. 22, 1901, William T. Gibbs, Buckingham, Canada.

Effects fusion and reduction by the heat radiated from a carbon-rod resistor, extending horizontally through the furnace above the charge. The furnace chamber may consist of firebrick or carbon, preferably having an arched roof to reflect the heat downward. One resistance rod, or several connected in series, may be used, extending between massive carbon blocks, which are set in iron sockets embedded in the side walls. The furnace has a charging opening through one side wall and a tap-hole and gas outlet through the other.

No. 693,482, Feb. 18, 1902, Edward G. Acheson, Buffalo, N. Y.

Reduces ores, for example, silica, by coating the particles of ore with graphite and employing a mass of the coated particles as a resistor. Acheson graphite is preferably employed, and is applied by mixing the ore and graphite and subjecting the mixture to a tumbling or rubbing process, by which the graphite is spread upon the surface of the particles. Example: Silica, 150 parts, is mixed with Acheson graphite 55 parts, and the mixture is rubbed in a mortar or tumbled in a barrel. The charge is then placed in an electric furnace between suitable terminals, fine silica being placed beneath, at the sides and on top of the charge. In one experiment the terminal electrodes had an exposed end surface of 4 square inches, and were separated 3 inches, the intervening space measuring 3 x 2 x 2 inches. The current, beginning with 3 amps. at 100 volts, rose in 7 minutes to 18 amps., and was turned off after 25 minutes. Globules of pure silicon were found scattered through the unreduced portions of the charge. Suggests that the furnace product may be removed intermittently, in a fluid or solid condition, or continuously in a solid, liquid or gaseous state. When the product is a volatile metal, such as zinc, condensers are provided. Alloys may be made by employing a mixture of graphite-coated particles of ores of two or more metals. Carbides may be made by employing an excess of graphite.

No. 701,718, June 3, 1902, C. M. Hall, Niagara Falls, N. Y.

The alloy of iron and silicon, titanium, aluminium, manganese, etc., produced by reducing the oxide impurities of bauxite and other impure oxides of aluminium, as described in the Hall patent 677,207, is used as a pigment, being pulverized in a mill and then mixed or ground with oil.

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Our Third Volume and Our Regular Departments.

This issue, which closes our third volume, is accompanied by the yearly index, which we trust may prove a useful and efficient key to the contents of the volume. An index is, in itself, a most concise summary. It would serve no useful purpose to try and summarize it again, but a few words may be said here concerning the serials and concerning the regular departments in our columns during the past year. Mr. F. A. J. FitzGerald's serial on electric resistance furnaces, which was started in our second volume, is now almost completed, and will form a most valuable treatise of an authoritative character in view of the author's long connection with the two most important electric resistance furnace industries in this country. In our March issue of the present volume, the first installment appeared of Dr. J. W. Richards' serial on metallurgical calculations, which has found an enthusiastic reception everywhere in metallurgical circles. It will be continued during the next year.

* * *

What may be called more properly regular departments are our Analysis of Current Electrochemical Patents, the Digest of United States Patents, the Synopsis of Periodical Literature, the Recent Metallurgical Patents and the Notes on Electrochemistry and Metallurgy in Great Britain. The last-named department appeared for the first time in the first issue of the present volume; it serves the important purpose of maintaining the continuity between the old and the new metallurgical and electrochemical world. The department dealing with Recent Metallurgical Patents and the Synopsis of Periodical Literature have remained the same as in the second volume, except that in the Synopsis more space has been allotted to the discussion of metallurgical subjects. To the many new readers which have been won for this journal during the last year an explanation, however, appears to be necessary concerning the difference between the Analysis of Current Electrochemical Patents and the Digest of United States Patents prior to July, 1902. The "Analysis" always covers those patents which have been issued during the past month or so, and covers all fields of electrochemistry and electrometallurgy. In this department the electrochemical United States patents issued since July, 1902, have been covered in our first three volumes. Of course, since the patents had to be analyzed as soon as issued, no detailed classification has been possible in this department. The case is absolutely different with the Digest of United States Patents prior to July, 1902. This department intends to cover all electrochemical and electrometallurgical United States patents which were issued before this journal was started. In this department the patents are classified according to subject matters. To get a complete record of all patents on any one subject, both the "Analysis" and the "Digest" should therefore be consulted.

Accuracy of Chemical Analysis.

We conclude in this issue an article by Mr. Thorn Smith, of the Ducktown Sulphur, Copper & Iron Co., Ltd., on a cooperative analysis of copper slag. This is a subject of great general importance, for analytical chemistry is the basis of all metallurgical work, as it is of all chemical knowledge in general. An accurate method of analysis of a properly prepared sample is a sine qua non in a smelting plant. Not only are the purchases of the ore made from the certificates of the chemist, but analyses of the ores, slags, matte, metal, coal, coke, etc., are the only sure guide to the superintendent when things are going wrong in the plant. Now, it is apparent that inaccurate and false data can disturb the whole line of argument by which the trouble is diagnosed. For instance, in the operation of a lead furnace excessive care must be observed to keep the slag of right chemical composition. If the conditions are not just right and all do not fit together like the stones of an arch, "a thousand dollars worth of lead and silver" can easily be put over the dump each day in the form of "foul slag." So it is seen that the results of poor analytical work can easily cause great and direct financial loss.

* * *

Mr. Smith has, in his own initiative, taken up this work of investigating the methods and checking one method against another, and has also eliminated the personal equation by checking work of different analysts against each other. While it would be irrelevant to enter into the discussion here of the various methods, yet we can say that as the result of investigations "the true has been sifted from the false"—a process described by Kant, the great philosopher, as the right way to approach the truth. We know that all chemists must feel indebted to Mr. Smith for his unselfish labor to put technical chemistry on a surer foundation. The work should be taken up broadly and elaborated by the Geological Survey in conjunction with the Carnegie Institute and the committee appointed by the American Chemical Society. It is a field for organized effort by several public institutions rather than by any one person. The fact that Mr. Thorn Smith devoted so much time from the busy life of a works-chemist to take this important subject in hand, is a fact that evidences his American public spirit.

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Business Activity.

There is unquestionably sweeping over this country a wave of prosperity which is the reaction from the period of slight inactivity in 1904. Perhaps the best indication of this is the fact that all the metal markets, especially that of copper, have shown decided strength during the past summer before the usual October purchases. Crop reports from the West assure us of a continuance of this state for at least a year, for it is founded on a safe and sure foundation of elemental wealth. Mr. James J. Hill, on a recent tour in the Northwest, where the fresh air from the prairie loosened his tongue, so silent in the vicinity of Wall Street, proclaimed prosperity for these States, because "all wealth comes originally from farms, mines, seas and forests, and all these are now abundant producers." These, of course, are the basic wealth of all countries, only needing development by an energetic and intelligent race. The old "spelling book" rhyme, ending with the refrain, "the farmer he must feed them all," is as true as ever, but in an age of metals and steam engines, mineral wealth is of almost equal importance. There is a curious concatenation of circumstances that makes for prosperity, well illustrated by a story from the breezy, virile West. A rich old farmer drove a herd of hogs to the station, to be converted into a variety of products, varying from spare-ribs to perfumed toilet soap, at one of the large packing houses of the West. When asked what he was going to do with the substantial check he received he said, "Buy more land." But they said, "What are you going to do with that land?" "Raise more corn," was the answer. And "with the corn?" "Feed more hogs." "But with all those hogs?" "Sell 'em." "And with that money?" "Buy more land." And then he repeated that series of monosyllabic words, "Land, Corn, Hogs, Money," to his infinite satisfaction.

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Such a bourgeois anecdote finds direct and concrete application in the metal business. It is mines, metals, money, more mines, more metals, more money. This is the essential reason of the expansion of the metal industry. And figures which do not prevaricate show this. Consider iron, copper and lead the most important commercial metals. The percentage increase in these from 1895 to 1904 has been as follows, according to the statistics of the United States Geological Survey: The output of pig iron in the United States increased in those nine years by 75 per cent, and we should here consider the relative inactivity in the iron industry in 1904. If for 1905 we assume an output of 20,000,000 long tons only, the increase in the output from 1895 to 1905 will amuont to 110 per cent. The cash value of the output of pig iron increased from 1895 to 1904 by 102 per cent. The output of copper increased in the same nine years by 111 per cent, while the increase in value of the copper produced amounted to 178 per cent. The lead production increased by 80 per cent, the value by 136 per cent. With great improvement in the size and the mechanical features of metallurgical apparatus, have come great advances in the knowledge of the proper metallurgical conditions—to operate a plant at its maximum efficiency. Our national prosperity is due originally to striving for ideals of perfection, just as Germany has turned from a nation of philosophers to a nation of engineers. Sentiment, thought and reason govern men's action, and unless this strain is held, our prosperity cannot long endure, and the spirit of the American pioneer flowering in these material achievements will perish. But the reserve democratic spirit of our country is stronger than ever, and it will be an ever-present help in time of trouble.

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Novelties in Water-Cooling Furnaces.

There is no better way to preserve a refractory material than to place in proximity to it an iron backing which is continuously cooled by a stream of water. This was originally done by laying between the courses of brick, water pipes. This had the practical result in the old brick shaft-furnaces of making extremely long campaigns. The final evolution of this idea was the water-jacketed copper furnace, where no refractory lining was provided between steel jackets and melting zone. The chilled slag immediately formed a natural refractory lining to the furnace. There was considerable talk at the time of the wide introduction of this innovation some fifteen years ago, that the water cooling would increase heat losses largely, and thus increase the amount of fuel per ton of ore. However, Mr. J. B. F. Herreshoff published some tests on the amount of heat lost in feed-water, and proved that only some 5 per cent of total heat was lost in water from jackets.

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The usual way of applying this water on a lead or copper furnace is by circulating it through metallic jackets. These jackets are usually cast iron or pressed steel, though cast steel and cast copper have been employed. Signor E. Ferraris, a brilliant metallurgist, of the Monteponi Co., of Sardinia, has built a lead furnace with the crucible made of cast-steel bolted segments cooled by a spray of water. Provided the film of water is always on this steel plate, the temperature must be less than 100° C., and thus the result is the same as if there was a 2-inch stream of water there, or even the Atlantic Ocean. This construction of Ferraris, used in iron blast furnaces to some extent, is better than jackets, mechanically, moreover, it is cheaper. There is also no danger of steam pockets or mud filling up the bottom of the jacket. We expect to see this scheme adopted in the United States to some extent in the future.

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In the iron blast furnaces much use has been made of the so-called "bosh plates," which are in essence a series of diminutive water jackets. They are often placed so near together that a modern furnace is one large mass of plumbing. The construction is extremely complicated, and means that much care is necessary to stop leaks. In Germany, Burgess has seized the bull boldly by the horns and has constructed at Geismar a furnace cased entirely by bolted cast-iron sections and lined with only some 3 inches of firebrick. This furnace is water cooled by a spray of water from top to bottom. It has been operated continuously for four years, and there is no reason to believe that it will ever burn out its lining. Its slightly increased fuel consumption is compensated for by long campaign. This idea will be taken up by prominent steel companies of America, since the refractory lining of an iron blast furnace is uncertain and unsatisfactory.

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Metallurgical Book-Keeping.

In the past ten years there has been a friendly invasion of this country by the class of "chartered accountants" from England and Scotland. Their reception here has been due quite largely to the formation of the large corporation. In a big concern, organization is essential. Cost-keeping and accounting has been developed as a science abroad, and those who practice it belong to a most important profession. The art of the statistician must be mixed with good, common sense. Strikingly similar to conditions in practical work, theory and practice must be harmoniously wedded. In a large metallurgical works there is present the sum of the concrete experience of all those who at furnace or desk have labored to bring the plant to its maximum efficiency. The result is a human institution more or less imperfect. Now, we believe that it would not be impossible to apply the methods of the "constructive accountant" to preserve these records in definite and "get-atable" form. Of course, the personal factor is important, and the keen eye of the expert steel roller or old furnaceman could never be mumified and placed in a vault for purposes of record. But this very example indicates important possibilities. The trained eye of the furnaceman employs by intuition some sort of optical pyrometry. By the commercial use of optical pyrometers or of pyrometers of other kind—there are now quite a number of reliable instruments available—it becomes possible to record the proper sequence of the temperatures in a process and to duplicate it exactly from the record, after it has once been determined. The secret is then no longer locked up in the personal experience of a few men. In the same way it would be well in general to bring all this accumulated experience to some definite system, so that, to assume an impossible case, the entire metallurgical force could be blotted out of existence, and their successors would have some actual tangible basis on which to build future results.

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Of course, such a set of metallurgical books explaining the irregularities of operation of the different departments would, unless opened by an expert, be clumsy and unwieldy. For following the thought of the foregoing paragraph, it is impossible to create Pygmalion-like flesh and blood out of a graven statue. Nevertheless, even to the best of practical men, systematic records are invaluable. And this scheme, if adapted to meet actual conditions, to be flexible and accurate, would preserve in concrete facts and figures the results of expensive experiments so that they could be applied to daily work. In short, what is needed is a metallurgical memory, if we consider that the smelting plant is an organism like a man, following the metaphorical philosophy of Schoepenhauer. Of especial value would such a department be in connection with the department of metallurgical research, which subject we discussed editorially in our July issue. Many thousands of dollars are expended each year at large smelting works in experiments. Most of these naturally are unsuccessful. But these unsuccessful experiments are important, for, as that veteran civil engineer, William J. McAlpine, said, "We learn by our failures rather than by our successes." But in too many instances these are forgotten, especially if the plant suffers a change of regime. In this case, the value of the work is carried to a rival plant, as actually as if the equivalent sum in money was transferred.

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One thing is certain, that engineers who publish as freely as possible in technical journals, put their experience in good shape for themselves and receive suggestions usually more valuable to them than the information they impart. It is an example of growing rich by giving away. But all this valuable knowledge, which Herbert Spencer alluded to in his "Synthetic Philosophy," should be preserved in the manner we have described, for the primary benefit of the owner of that experience. The task is a hard one, for it has been tried and found to be so by more than one concern. But its very difficulty should be an inspiration to some one to combine the qualities of the accountant with those of the engineer and leave in permanent record human thought. We feel sure that this will be the general practice in the future, as it is at present the gratifying exception.

New York Section Meeting of the American Electrochemical Society at the Electrical Testing Laboratories.

The most successful and enjoyable meeting ever held by the New York Section of the American Electrochemical Society took place on the evening of November 23, at the Electrical Testing Laboratories, 80th Street and East End Avenue, New York City. The attendance reached almost the hundred mark, and proved that in order to attract to the meetings the rank and file of the members it is necessary to offer them something interesting to see. Some of the nearby places, like Perth Amboy, had sent remarkably strong delegations. Not enough credit can be given to the officers of the Electrical Testing Laboratories for the splendid arrangements which they had made and for the indefatigable courtesy with which the visitors were entertained.

The laboratories are situated in the far east of Manhattan Island—so as to be perfectly free from all the disturbances of trolley cars, etc. For this reason some of the members were late in arriving, and it was after 9 o'clock when the chairman of the section, Dr. Charles A. Doremus, called the meeting to order. He introduced the president of the Electrical Testing Laboratories, Mr. J. W. Lieb, Jr., who reviewed the history of the institution. The laboratories were founded with the support of a number of the largest Edison companies in this country, who recognized the necessity of such an institution, not only for their own purposes for testing lamps, but for the electrical industries in general. The main work of the laboratories is still the testing of lamps, not less than 17,000,000 having been tested last year. But the equipment of the laboratories is so complete that almost any electrical tests desired may be easily performed there. Mr. Lieb pointed out that without the aid of the Edison companies it would have been impossible to equip and maintain the laboratory in its splendid condition, but he emphasized forcibly that the laboratories in the testing work are absolutely independent of any commercial relations or of any company.

Dr. Clayton H. Sharp and Mr. W. S. Howell then explained in very interesting speeches the leading principles of the arrangement and management of the laboratory. In order to make the labor cost of testing a minimum, the plan has been followed of laying out all the frequently recurring tests in advance. All requisite apparatus for such tests is put permanently in position and all electrical connections made, so that it is necessary only to connect the material or instrument to be tested into the circuit, close a switch or two and proceed to make measurements. In cases where tests are required less frequently, so that it is not found possible to set apart all the necessary apparatus for their exclusive use, a place has been assigned to the tests, and the electrical connections have been made so that but little preliminary work remains to be done. Mr. Howell then invited the visitors to a thorough inspection of the laboratories, beginning with the top floor and ending in the basement, where refreshments were served.

The chief impression of the visit was the remarkable size of the laboratories—far beyond anything in size and equipment which is ordinarily understood under a laboratory. It is an old electric generating station remodeled for its new purpose. It is a three-story building, 50 x 120 feet, with a two-story addition, 30 x 120 feet. The building is thus amply large for all the requirements of an extensive testing laboratory. Electrical energy is available in any required amount, both in form of direct current and alternating current. Further, with respect to apparatus and instruments, the equipment is so complete that everything that might be needed is at once available. If inventors wish to carry on secret investigations they are at liberty to do so in the rooms on the second floor, which are being rented for such purposes.

There can be no doubt that when the splendid equipment and facilities of the Electrical Testing Laboratories become better known, they will be made much use of for the most varied purposes. For experimental electric furnace work the basement would appear to offer excellent facilities.

The following notes give only a general idea of the arrangement of the laboratory. For more detailed information the reader may be referred to an illustrated pamphlet of Dr. Clayton H. Sharp, which may be had from the Electrical Testing Laboratories, 80th Street and East End Avenue, New York City, for the asking.

The lowest floor, slightly below the level of the street, is concreted and is used as a machinery room. The non-electrical equipment of this room includes a sulphur dioxide refrigerating machine, having a capacity of a ton and a half of ice per day, together with a brine storage tank and a triplex pump for circulating the cold brine through pipes reaching to the top of the building. Besides this there is a motor-driven airpump, which compresses air in a reservoir from which pipes lead to outlets in many parts of the laboratory. This compressed air is used for operating blast lamps, and is especially useful in removing the dust from articles, particularly delicate electrical apparatus. For executing repair and construction work on apparatus a small but excellent equipment of light machine tools has been installed on this floor. Electrical energy is supplied to this floor from the New York Edison Co., either in the form of direct current at 120-240 volts, or of three-phase alternating currents, 25 cycles 6,600 volts. There are installed also several motor-generator sets and a storage battery. The range of currents at the disposal of the laboratories is very great, particularly since the cables of the Edison system leading to the building are of such size that 500 amps. at 240 volts are available from this source alone. For electrochemical work heavy currents at low voltage are available.

The second floor is devoted principally to the purpose of photometry. It has been said—and this statement is undoubtedly true—that the photometric testing rooms are the finest of their kind in the world, the equipment for making tests of all possible kinds being very complete. On this floor there are also the rooms which are rented to investigators who wish to make secret researches.

The top floor is occupied by offices and contains the laboratory for general electrical testing, the standardizing laboratory for resistance and conductivity tests, instrument tests, high-voltage tests, magnetic tests, etc., etc.

There were quite a number of employees of the Electrical Testing Laboratories present, who, together with the officers of the institution, were indefatigable in courteously showing the visitors around and explaining the equipment. As said before, the occasion was altogether delightful and was greatly enjoyed by all who attended it.

Notes on Electrochemistry and Metallurgy in Great Britain.

(From Our Special Correspondent.)

The Autumn Meeting of the Iron and Steel Institute.

The Second Day's Proceedings.

The only paper which was discussed in any detail on the second day of the Sheffield meeting was that by Mr. Talbot, on "Segregation in Steel Ingots." After referring to the remarks of earlier writers on segregation, the author observed that Pourcel, in 1893, had pointed out that aluminium has a decided tendency to promote segregation, but that no systematic series of tests had been published concerning such results. Mr. Talbot's paper, therefore, contains a record of a series of parallel tests made on ingots from the same heat, with and without the addition of a small amount of aluminium to the ingot.

The ingots taken for examination were from both acid and bacic open-hearth furnaces, and varied in weight from about

1½ tons to 3½ tons, and drillings were taken over the whole surface of the divided ingot.

As a rule, the results show that in the case of ingots to which no aluminium had been added, excessive segregation down the central line of the ingot occurs from about 6 inches from the top to about half-way down the ingot, that sulphur is the element which tends to segregate most, phosphorus next, followed by carbon, and finally manganese, the segregation of which latter element is so slight as to be almost negligible. No silicon determinations were made, as the amount of this element present was extremely small.

An examination of the results obtained shows clearly that by the use of aluminium a billet of a much more regular composition is obtained. This is especially important in the case of carbon, especially if this steel had, without aluminium, been intended for rail purposes, as the surface of the rail would probably have shown considerable irregularities in the carbon percentage, with a consequent want of uniformity in its wearing properties.

In cases in which the carbon has segregated to the center it is obvious that corresponding areas will be found at the sides in which the carbon is less than the mean, through the carbon having migrated to the center to a greater or less extent.

As regards the homogeneity of the steel, Mr. Talbot's analysis tended to confirm Pourcel's view, that the addition of aluminium tended to promote this. His experience on the addition of aluminium in the ingot mould during casting had always been that the aluminium appears to make the metal set quicker. This he was aware was against the view usually held by metallurgists. Thus, in Mr. Harbord's recently published book on steel, it is stated: "The addition of very small amounts of metallic aluminium to such metal"—*i. e.,* metal containing dissolved oxides—"is found to cause a marked increase in the fluidity of the molten metal, to stop the evolution of gas, and to allow of the production of sound ingots without blow-holes."

Not only does the addition of a little aluminium to the metal, as it is run into the ingot, have a marked effect in setting the surface, but it also, in the author's experience, tends, when added above a certain quantity, to form cavities in the upper part of the ingot, so that the amount added has to be strictly regulated. The setting effect on the top of the ingot is so marked that at the works with which the author was connected, after it had become the custom to add aluminium regularly, the ingots were never sanded over or stoppered down, as no such treatment was necessary, either with acid or basic open-hearth steel. With mild steel, also, it was found that the moulds could be stripped sooner when aluminium had been added. It was also observed that when the same quantity of aluminium, viz.: some 3 to 4 ounces per ton of steel was added to the metal as it ran into the ladle, its effect was not so pronounced as when added in the ingot mould as the ingot was being teemed. As to what is the precise action of the aluminium, the opinions of experts differed, some considering that increased fluidity was afforded, others that the reverse action took place. Theoretically, one would undoubtedly expect some increase in temperature, owing to the reaction between the dissolved oxides in the metal and the aluminium, an action akin to the well-known thermit process. If an increased temperature is obtained, with the consequent increased fluidity, this would cause the steel to take longer to solidify, and would consequently tend to increase the segregation, provided that the aluminium has no special action of its own on the metal, whereas the numerous analyses made by the author prove that there is a considerable diminution in the amount of segregation. As the metal appears to set quicker, and as, consequently, segregation would be expected to be less, due to this quicker setting, the analyses seem to agree with this view. The author's usual practice was to add about 3 ounces to 4 ounces of aluminium per ton of steel in the ingot, but this was never added until the ingot mould was approximately two-thirds full. Assuming the aluminium to be all concentrated in this top third of the ingot, it would then only be at the rate of about 12 ounces per ton, or about 0.033 per cent aluminium.

The author concluded with the suggestion that it would be worth while for other investigators interested in the manufacture of higher carbon steel, such as for rail, tire and similar purposes, to follow up these results, with a view of proving whether a more uniform and regular steel is not thereby obtained, a result well worth the few pence per ton the aluminium would cost. Perhaps the chief result to be looked for would be the decreased amount of crop end that it would be necessary to cut off from the top of the ingot, due to the greater solidity of the top and the lessened amount of segregation in this top part of the ingot. This alone would undoubtedly pay for the cost of the aluminium added, without considering the more regular quality of the finished product as a whole.

A very keen discussion followed, Mr. J. E. Stead being the first speaker. As one who has studied the phenomena of segregation for fifteen years and was still puzzled at some of its manifestations, he had several questions to propound. Why, for example, was there segregation at all? Why did aluminium reduce it? Why was the top of an ingot purer than the rest? It was found that the carbon in the upper layer of an ingot was 18 per cent less than in the center, and sulphur and phosphorus were 30 to 50 per cent less. Why were there less impurities in the top than in other parts? To these questions fairly satisfactory answers could be given. There were a few axioms on the structure of steel to be kept before the mind. First, rapid cooling produced small crystals; secondly, agitation had the same effect; thirdly, the converse was true, slow cooling and stillness resulted in the formation of large crystals. After explaining the growth of "fir tree crystallites" in a cooling mass of steel, Mr. Stead went on to show that the layers first "frozen," those at the top and bottom of the ingot, must be purer than the remainder, hence the diminution of carbon and phosphorus in these parts. The introduction of aluminium had, he thought, the same effect as tranquillity. It prevented circulation and increased the rapidity of cooling, and, by the quietness that it caused, was instrumental in the production of large crystals. Some time ago he (the speaker) made tests of the reputed increased fluidity of steel containing aluminium. Two long spiral moulds of small bore were made. Into one untreated steel was poured. Aluminium was then added to the same ladle, and the other mould poured. On examination it was found that the metal had filled nearly three times as much of the second as of the first mould. But the result was not due to greater fluidity. It was due to the fact that the steel in the first honeycombed, frothed up, cooled and choked the bore; in the second, the flow was solid, it did not cool so rapidly, and hence went further into the mould.

Prof. Harbord was in general agreement with Mr. Stead as to the increased "facility of flow" caused by aluminium, a term that he has adopted in the second edition of his work, and which he considers preferable to "fluidity," and he thought that it was due to a mechanical action of some kind. Mr. Stead had suggested that they should assist segregation. That might do for guns and shafts, but it would not do for rails, and if the introduction of a small quantity of aluminium provided a means of obviating segregation for a few pence per ton, the method should receive the attention of steel makers.

Mr. A. Windsor Richards said that he had found that in those ingots which set most quickly there was least segregation, and therefore they had given up sanding or shuttering down. He believed that if steel was properly blown and recarburized it set so quickly that no aluminium was needed to prevent segregation, and he thought the introduction of it would cause piping. He not only agreed with Mr. Stead that segregation was not injurious to rails, but he went so far as to suggest that it might even be an advantage, since the wearing parts were rendered purer by its presence.

After succeeding speakers had ranged themselves for and against the use of aluminium, Mr. Talbot, in reply, urged that the results which he had put before the meeting showed that the addition of aluminium was at any rate worth trying, the quantity necessary being too small to have any effect on the final product.

Electric Furnaces at the International Electrical Exhibition

It seems that unwittingly I did this exhibition an injustice in my last letter in stating that there were no electric furnaces exhibited. It was only on a third visit that I came across two of these, exhibited by Messrs. Marryat & Place.

The first, of the Moissan type, consists of a chamber of refractory material; in this case firebrick, having a hole at either end for the carbons carrying the current. The upper half of the box can be removed, showing the carbons lying along their channels with a small space, or hollow, around their tips, so that a crucible may be placed between them. When working, the top box is replaced, and investigation can be made through a spy-hole opposite the arc. The carbons are 2 inches long. The box or furnace measures about 2 feet square. The cavity for the crucible would not be more than about 6 inches x 6 inches x 8 inches deep at the start, but, of course, the sides burn away during the operation; 600 to 800 amps. (alternating) are used.

The second furnace exhibited was of the two-pole Siemens type. The arrangement is similar in appearance to an ordinary well with cross-beam for winding. The "well" is a chamber of firebrick about 1 foot 6 inches square by 2 feet deep. Into this descend the two electrodes, hanging from the cross-bar by screwed rods, provided each with a hand-wheel, so that the distance apart of the electrodes is adjustable. The electrodes are carbon blocks 4 inches x 4 inches x 2 feet long. The method of working is to place scrap in the well, and allow the arc to play in the middle of it; 800 amps. (alternating) is the current employed.

A Municipal Electrochemical Undertaking.

Municipal activities are highly developed in England, the latest departure being the action of the Poplar Borough Council, who are seeking estimates of the cost of a plant wherewith to manufacture their own disinfectants. They propose to employ electrolytically-produced solutions of the hypochlorites, and to use the liquid yielded by the electrolizer installed for the purpose of washing streets, flushing sewers and for general distribution by the officials of the Local Public Health Department. The use of low-strength solutions is contemplated, the manufactured product to have the low strength of only 2 grammes of available chlorine per liter, and the distributed product one-fourth of this strength. Tenders have been received for the installation of a "Hermite Electrolyseur," using a solution containing 40 per cent of sodium chloride and 10 per cent of magnesium chloride, and producing per hour 250 liters of a strength of 1 gramme per liter. The electrolizer would require a current of 30 amps. and a potential of 110 volts. To put it mildly, the electrochemical efficiency offered of about 76 grammes per kw-hour would be dear if the apparatus were a gift. In a paper just read before the Faraday Society, there are recorded efficiencies of hypochlorite production at strengths of 3 grammes per liter, which are 350 per cent greater than those mentioned above.

Prof. Ayrton on Power Distribution

When I referred to the British Association's meeting in South Africa as a picnic, I was better advised in the choice of a word than I then realized. The picnic spirit dominated certain papers, with the result that one past-president of the Institution of Electrical Engineers gave the hall-mark of the past-presidential sanction to the use of the word "enthused" in a paper on a subject of scientific interest. Another, Prof. Ayrton, lectured on power distribution, in a manner most charitably described as "popular." Discarding the excrescences due to the holiday environment, the professor announced himself as a heretic on the subject of power distribution, and urged the advantages of high-tension direct-current transmission as against alternating currents. A leading article in the *Electrician* hits off the subject very happily in the following words: "The heresy—or shall we not rather say the reactionary movement—is not a new one; it was, like a greater movement, fostered at Oxford, but, strange coincidence, it has thriven most at Geneva." The Geneva allusion is, of course, to M. Thury's experiments between St. Maurice and Lausanne at 23,000 volts.

I need not enter into Prof. Ayrton's specific pleas for continuous-current distribution, but I rather wonder whether his pose as a heretic is that of a mental exercise and recreation or is meant in all seriousness. I may add that on good local evidence the professor does not seriously regard the Victoria Falls as likely to become a second Niagara in regard to electric power distribution. Within the immediate neighborhood of the Victoria Falls there are no possibilities of electrochemical or electrometallurgical industries being established, while the distance to Johannesburg, measured in an absolutely straight line, is 586 miles. Therefore, if electrochemical industries arise in South Africa it is exceedingly probable that, as in England, these can only center around collieries.

Market Quotations During October.

There are one or two small changes to chronicle in regard to chemicals. Arsenic has risen to £14.17.6 per ton, and sulphate of ammonia to £13 per ton. Among the coal tar products carbolic acid crystals have receded ¼d. per pound, but naphtha (crude) has risen from 3d. to 3¼d. per pound, and naphtha solvent from 9½d. to 10½d. Copper sulphate has increased in price to £22.15. Potassium carbonate and caustic potash are unchanged. Shellac is still quoted at £9 per cwt.

Copper closed at £71.5 per ton, a retrogression compared with the price of £72.18.9, which it reached on Oct. 23. Tin fell from £147 to £145.15 per ton on the 9th, the subsequent rise has been a fairly steady one, the closing price being £149.15. Hematite pig iron touched 73s. per ton on Oct. 13, and although now lower at 70s., this price represents a rise for the month of nearly 8s. Cleveland pig iron opened at 51s. 6d., reaching 54s. 6d. by Oct. 18. The price on Oct. 31 was 52s. 9d. Lead is firm in price, fetching £15.5 per ton. Quicksilver is unchanged, at 142s. 6d. to 145s. per flask.

London, Nov. 8, 1905.

CORRESPONDENCE.

The Huntington-Heberlein Process.

To the Editor of Electrochemical and Metallurgical Industry:

Sir:—With reference to your notes on page 363 of the October issue permit me to express the opinion that you appear to overestimate the chemical effect of the lime addition in the Huntington-Heberlein process.

In my opinion the characteristic feature of this process is best brought out by its analogy to the Bessemer process. The distinguishing point in the case of roasting lead ores is the easy fusibility of the galena, and the principal object of the addition of lime is to keep the charge open and accessible in all parts to the air blast and to prevent the sintering and melting of galena, which would result in making the air blast ineffective. It, therefore, seems to me that whatever the chemical reactions of the lime may be, its chief function in the Huntington-Heberlein process is not to be found in the chemical effect but rather in a physical one, namely, to keep the charge in the right physical condition for the application of the air blast.

Lead Smelter.

[In connection to this subject attention may be called to several articles reviewed in the "Synopsis of Periodical Literature.]

Electrically-Heated Carbon-Tube Furnaces.

A recent valuable Faraday Society paper, by R. S. Hutton and W. H. Patterson, deals with electric furnaces constructed with a carbon tube through which an electric current is passed. This type of furnace is considered to be the most readily available for the very highest temperatures; despite some inherent disadvantages, due to the presence of carbon, it is capable of very wide application. One of its chief advantages lies in the fact that the temperature can be very easily regulated and kept constant.

Messrs. Hutton and Patterson do not claim any superiority of their types of carbon-tube furnaces over those constructed by other workers. In particular, they state that the construction of some of the furnaces of Dr. H. N. Potter (see our Vol. I., pp. 187, 188, 250; Vol. II., p. 203; Vol. III., p. 14) is obviously more perfect in many of the essential details; although, probably, they could not very easily be made and installed for ordinary laboratory work. Messrs. Hutton and Patterson have been able to get satisfactory results with a very simple type of construction.

General Considerations.

The most important points in the construction of carbon-tube furnaces are, firstly, the provision of a satisfactory device for leading the current to the ends of the tube, and, secondly, the protection of the tube from contact with the air or other material capable of reacting with the carbon to burn it away. It is, moreover, a very great advantage to closely jacket the tube with some substance, which, whilst not tending to combine with the carbon, forms an efficient heat-insulator. Even with the most carefully constructed system of exterior tubes with intermediate gas spaces a great amount of heat is carried away by convection currents in the gases, and consequently much more power has to be expended in the tube to attain any required degree of temperature than when the tube is closely jacketed by some material of low conductivity.

The connections at the ends of the tubes should be so constructed as to remain sufficiently cool to prevent any oxidation of that part of the carbon which is exposed to the air. Moreover, provided the ends are kept cool, it is an easy matter to make a gas-tight joint for the passage of any desired gas through the tube.

Graphite Tube Furnace.

The first furnaces made by Messrs. Hutton and Patterson were constructed of Acheson graphite, a tube being bored from a solid rod of about 3 centimeters diameter, as shown in Fig. 1. This method of construction is considered to be advantageous when only a few experiments have to be carried out.

The tube is bored, turned from a solid rod, and is screwed at the ends into graphite plates (2.6 by 5.2 centimeters), which are clamped to copper holders (not shown in the figure). The glass extensions, which are fixed into the graphite tube at A and B by asbestos, serve for the passage of gas through the tube; the plate-glass ends enable the progress of the experiment to be observed from either end. The central portion of the tube is surrounded with carborundum, or other heat-insulator, not shown in the figure. The material to be heated is placed in small carbon boats, which are advanced to near the center of the tube. These boats are so shaped as to touch the tube at only a few points, so that they do not conduct, themselves, any appreciable amount of the current.

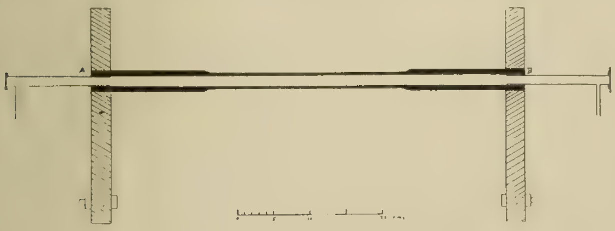

FIG. 1.—GRAPHITE TUBE FURNACE.

As a jacketing material calcined magnesia was not found satisfactory, on account of very great shrinkage. Carborundum is considered to be a very excellent jacketing material. It is a fairly good heat insulator and has the advantage of being a reducing agent, consequently protecting the carbon from burning away. Messrs. Hutton and Patterson used carborundum in a fine granular condition, the grains just passing through a sieve, 180 meshes to the linear inch. As soon as the graphite has attained a fairly high temperature these grains frit together, forming an adherent tube-shaped jacket around the graphite which serves to strengthen and protect it.

Agglomerated Carbon-Tube Furnace.

For most of their experimental work, Messrs. Hutton and Patterson use agglomerated carbon tubes, which can readily be obtained of almost any desired size from manufacturers of arc carbon. The construction, as finally adopted, is spoken of as very excellent in actual use.

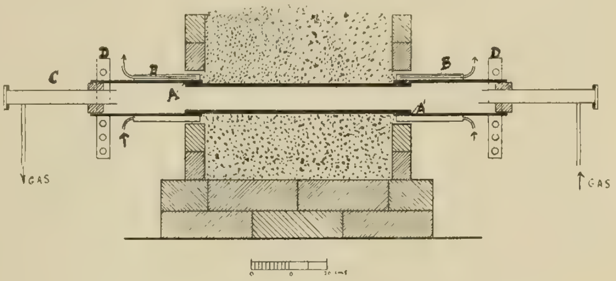

FIG. 2.—AGGLOMERATED CARBON TUBE FURNACE.

The ends of the carbon tube, which may with advantage be somewhat roughened by filing, are electro-coppered, and then soft-soldered to thick copper tubes, just large enough to slip over them. The copper tube is provided for a short distance with a brazed-on water jacket, which effectively cools the joint and protects it, even when a very high current density is necessary; at the further end of the copper tube a copper clamp connected with flexible cables leads in the current. By soft-soldering the carbon to copper tubes with brazed water jackets, it is always possible to easily replace the carbon without any fear of injuring the joints of the water jacket. The length over which the soldered joint has to be made obviously depends on the size of carbon tube and the strength of current to be sent through it, but with this water cooling the necessary surface of contact is surprisingly small.

For the earlier experiments, carbon tubes 30 cms. long, 15 mm. internal, and 20 mm. external diameter, were employed; these were jacketed as before with carborundum, which was generally kept in place with an external asbestos tube and asbestos washers fitting tightly up against the ends of the water jackets.

A similar construction has been used with much larger tubes (60 cms. long, 67 mm. internal, and 82 mm. external diameter), as shown in Fig. 2. The ends of the carbon are coppered and soldered at A and A^1 to copper tube extensions provided for a short distance with water jackets B, which keep the joint cool. The current is led in by copper clamps D. Glass tubes C are connected to the ends of the copper tubes by rubber stoppers, and serve for the passage of gas and for observation of the progress of the experiment.

With these large tubes it is advisable to use some sort of diaphragm near the ends to prevent the radiated heat from burning the surface of the rubber or cork stoppers, which, as the copper is effectively cooled by the water jacket, can serve to attach the gas inlet and outlet extensions; with the small tubes there is no need to take this precaution.

Owing to the porosity of the carbon tubes at high temperatures, it is difficult, even with a fairly rapid current of hydrogen, to keep the gas in contact with the heated material free from carbon monoxide. Since this was highly desirable in some of the experiments of the author, a modified form of construction was adopted for this special case. In the first modification the carbon tube was surrounded with a water-cooled, gas-tight metal jacket, through which hydrogen was passed, but with this arrangement the convection currents in the external gas carry off an enormous amount of heat, a very high current density is required to reach even moderately high temperatures. On the other hand, when the carborundum-jacketed tube is, as a whole, surrounded with the gas jacket, the gas in the inner tube still contains 1 or 2 per cent of carbon monoxide, which is probably due to the carborundum containing a small amount of unreduced silica. The arrangement finally adopted (see Fig. 3) has worked well, and is, moreover, considerably more efficient than when a bare carbon tube is surrounded with a large gas enclosure.

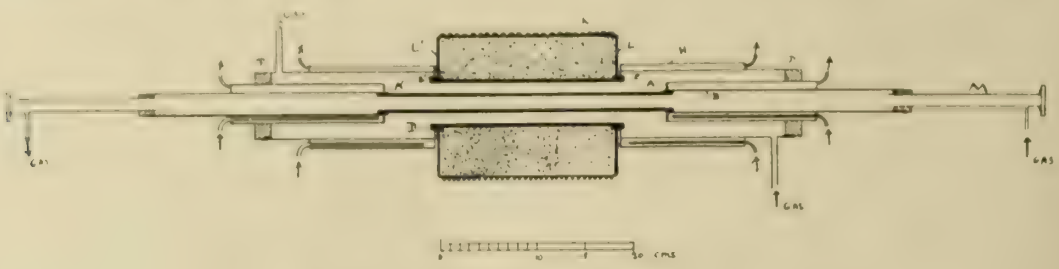

FIG. 3.—SPECIAL FORM OF AGGLOMERATED CARBON TUBE FURNACE.

The central carbon tube A A^1 carries the current, and is the heating tube proper; it is provided with copper water-cooled extensions B C, to which the current is led by clamps not shown. To insure the purity of the gas in the tube A A^1 it is surrounded by a concentric carbon tube E E^1, which is also provided with copper water-cooled extensions F H, a gas-tight joint between the two tubes being made by the rubber stoppers P. The outer carbon tube is jacketed with a granular heat-insulator to prevent, as far as possible, the radiation of heat. This granular material is held in place by asbestos washers L and an asbestos tube K. Glass extensions M serve as in Fig. 2 for the passage of gas through the heated tube, and for observing the progress of the experiment.

Hydrogen is passed through the space between the two carbon tubes, as also through the inner carbon tube, and in this way it is easy to keep the gas in contact with the heated material almost absolutely free from oxygen compounds.

For a large number of purposes these precautions are quite unnecessary, and the simpler form of construction is all that is required.

The following table gives some of the results obtained by the authors:

Type of Furnace.	Jacketing Material.	Observations
Graphite; 28 cms. long 2.0 external, 1.5 internal diameter	Carborundum	295 amps. at 8.2 volts melts nickel in 26 mins.; 320 amps. at 9.6 volts melts nickel in 13 mins.; platinum in 16½ mins.
Agglomerated carbon; 27 cms. long, 2 cms. external, 1.5 cms. internal diameter.	Carborundum	140 amps. at 7.7 volts melts nickel in 19 mins.; platinum in 28 mins.
Agglomerated carbon; 27 [illegible]	None; bare tube surrounded with hydrogen.	235 amps. at 16.4 volts required to melt nickel (5½ mins.)
Agglomerated carbon; dimensions, etc., as Fig. 3.	Carborundum outside outer carbon tube.	200 amps. at 21 volts melt nickel in 3 mins.; current increased to 240 amps. at 25 volts, melting platinum, in further 2½ mins.
[illegible] ternal, 6.7 cms. internal [illegible]	Carborundum.	600 amps. at 8.6 volts gave 1,200° C. [illegible] 850 amps. at 13.0 [illegible] 12 mins.; platinum in further 8½ mins.; [illegible] 860 amps. at 11.1 volts.

Important differences manifest themselves according to the type of furnace construction adopted. With a bare tube surrounded with a gas jacket, a given current very soon brings the carbon up to a temperature which remains more or less constant, or at the most only continues to rise very slowly; on the other hand, with carbons closely jacketed with a granular material, which is a good insulator, the temperature continues to rise for a very long time, and in this way, with a much smaller current and expenditure of power than is required in the previous case, a high temperature can be gradually built up. As the temperature rises, however, the jacketing material becomes an electrical conductor, and even pure oxides at or near their fusing point become good conductors, and by carrying some of the current tend to limit the temperature attainable.

A Co-operative Analysis of a Copper Slag.

By Thorn Smith.

(*Concluded from page* 413.)

The results on lime vary between the highest and lowest to the extent of 1.51 per cent. A discussion as to the causes of the various differences is somewhat difficult. The writer's result was obtained by a double precipitation, after the elimination of manganese, and ignition to oxide. Nos. 5, 12 and 14 agree closely, and two of them at least are by chemists whose complete analyses are far superior to the rest of those coöperating. Nos. 3 and 6 are fair, while Nos. 2, 7, 8, 9 and 10 are not so satisfactory on an element so comparatively simple of estimation. No. 4 is not worthy of consideration.

No. 5 determined the line in the acid filtrate from the silica determination by adding ammonium oxalate and titrating with permanganate. This is the usual rapid method used in smelter practice. No. 12 added ammonium oxalate to a solution free from iron, alumina, zinc, copper and manganese, boiled and allowed to settle. Then filtered and titrated with permanganate. His permanganate was standardized against iron wire of a purity assumed as .996, and also against an oxalic acid solution of known strength. He was able to check his result closely by the same method as used by No. 5.

No. 14 secured his result by a method which the writer has never seen in print. In brief, he weighed 1¼ grams of the slag into a No. 2 beaker, treated with aqua regia until complete solution was effected, then ran to dryness twice, exactly as in a silica determination, took up with dilute HCl and filtered off the silica. The silica was ignited and treated with hydrofluoric acid and the residue added to the main filtrate. This solution was neutralized with ammonia and 3 c. c. of HCl added in excess, after which hydrogen sulphide was run in until the copper was precipitated. The volume of the solution at this point should not exceed 200 c. c. The solution was then filtered into a 250-c. c. flask and well washed with hydrogen sulphide water nearly to the mark. At this point 15 c. c. of strong ammonia was added and hydrogen sulphide again passed for ten minutes. The solution was then made up to 3 c. c. above the mark, to allow for the precipitate, and well shaken. Two hundred c. c. were then filtered off into a beaker and boiled down to a low volume. A little HCl and Br were added at this point to clear up any turbidity. Fifty c. c. of a saturated solution of ammonium oxalate were added to the boiling solution along with

10 c. c. of strong ammonia. The precipitate was allowed to settle for two or three hours and then filtered into a No. 6 beaker followed by thorough washing. The clean precipitate was next dissolved in HCl and reprecipitated. It required much longer standing, and even then a little lime settled out after long standing. This precipitate was titrated against permanganate.

From the result obtained the writer believes this method to be accurate, but not more so than the usual method.

Of the fair results No. 3 precipitated the lime from the alkaline solution after separating the iron, alumina, zinc, copper and manganese in the usual manner. He dissolved the first precipitate, as did No. 14, and reprecipitated. His slightly low result is due probably to the slight loss liable to occur at this point, as mentioned by No. 14. No. 6 made two separations of the iron and alumina and one ammonia oxalate precipitation. He titrated with permanganate, presumably using the same solution as he did in his iron titration. Inasmuch as his result on iron is much too high it would follow that his lime would be correspondingly high. The contrary, however, is the case, and is due in all probability to the lime held up by the iron and alumina, which may be considerable. This point was brought out forcibly by Mr. Rueger in the *Engineering and Mining Journal* of April 28, 1904. Manganese was not removed and would be precipitated in part. No. 2 used a method suggested by the writer. The latter neglected to state that the manganese should be removed before precipitating the lime, hence this high result must, in the absence of other data, be ascribed to this factor. It was hoped that one or more of those coöperating would call attention to this point, but not one did so. No. 7 used the ordinary rapid method. No. 8 used the same method as No. 2, and the criticism of that result will apply. No. 9 failed to state the method used. No. 10 precipitated in the acid solution from the silica determination and titrated with permanganate. Inasmuch as a rapid method does not secure all of the iron and as this chemist's iron is practically correct, it follows that his permanganate solution is weak. Using the same methods Nos. 5 and 10 differ by .6 per cent. The method is the usual rapid smelter control, and gives fair results, if too great haste is not employed.

An accurate determination of lime demands primarily that all of the lime be separated from the iron and alumina. It further demands a second precipitation, and in this the zinc and magnesia are eliminated. If titrated the standard permanganate must be correct. In the absence of this knowledge the lime must be ignited to oxide. Lime is not difficult to estimate, especially when not present in too great quantity, but it does require a strict attenion to details and manipulation.

A considerable variation is apparent in the results on magnesia. The standard was secured by two precipitations, previous to which the lime had been twice precipitated. Of the good results Nos. 2, 14 and 15 are beyond criticism, while No. 17 is practically so. Nos. 3, 6 and 8 are not satisfactory. No. 4 found a trace. No. 2 followed the method of the writer and checked exactly. No. 12 may be considered satisfactory, and was made by one precipitation, but previously two separations of lime had been made. The absence of much ammonia salts might perhaps make two precipitations of magnesia unnecessary. It is possible that this chemist lost a trifle by igniting too strongly. Fifteen minutes, as in his case, may cause a loss when the heat is intense. No. 15 followed the method of the writer. No. 14 was secured after throwing out the lime and other bases as previously described at length. This chemist ignited in porcelain, though why is not apparent. However, such does not invalidate his result. Of the unsatisfactory results, No. 3 was obtained by practically the same method as the writer's, and no cause for the low result can be found from a study of other parts of his report. No. 6 obtained his result by one precipitation, preceded by two ammonia separations of iron and alumina and but one for lime. It is evident that in addition to the aforementioned causes of low results he did not secure a complete precipitation of magnesium phosphate. No. 8 used the same method as the writer. The latter has repeatedly obtained results of this tenor when but one precipitation has been made, and this may be the cause of No. 8's high figures. No. 4 found but a trace. He evidently overlooked the fact that persistent stirring is often necessary, and often a few grains of sodium phosphate must be added in order to start the precipitation.

The chemists of the Association of Official Agricultural Chemists are able to secure closely agreeing results on phosphoric acid which calls for practically the same precipitation. Many of the assistants in the experiment stations are men of very little experience outside their work as students. Hence an accurate determination of magnesia cannot be considered as more than ordinarily difficult.

Manganese is present in this sample of slag to the extent of .37 per cent. When present in such small amounts, in a material of this nature, it does not seem to submit to rapid methods. The determination of manganese is not usually considered difficult, but a glance at the results obtained by the various chemists makes it appear difficult in this case at least. The writer secured his result by precipitation as phosphate after a thorough preliminary separation.

Of the eight results reported but three can be considered as satisfactory. Nos. 12 and 14 agree with that of the writer exactly, while No. 4 has a fair result. The remaining five are not at all satisfactory. No. 12 secured his figures by tritration with permanaganate after separating the manganese from the basic acetate filtrate with bromine. No. 14 used the same method as the writer. No. 4, whose complete analysis is so exceedingly poor, reports that in his laboratory the separation of manganese by the use of bromine has never been found a decided success, and that often no precipitate at all is formed when manganese is known to be present. This is another remarkable statement, and demonstrates the need of a little study. He precipitated with hydrogen sulphide after taking out the zinc and copper in an acetic acid solution. This method is, of course, accurate if properly performed.

Of the unsatisfactory results No. 2 reports his result as obtained by the method suggested by the writer when the samples were sent out. It consists in weighing the oxide precipitated by bromine. There is, in this method, too much danger of contamination, and while it often gives good results it just as often comes out wrong. No. 3 used exactly the same method, but instead of his result showing contamination it is apparent that at least one-half was lost in the course of its separation. No. 6 used the Volhard method, which was demonstrated in the first report to be unsatisfactory in the case of this slag. It has also since been condemned by the committee on methods of zinc analysis. Poor chemicals is probably the chief cause of failure. No. 8 used the same scheme as Nos. 2 and 3, and evidently did not secure all of the manganese. No. 10 used the same method and jumped to the other side of the standard. The majority of the poor results were obtained by weighing the manganese oxide as secured by precipitation with bromine. One was secured by Volhard's method. Further comment does not seem necessary.

The results on sulphur as a whole are fairly satisfactory. An accurate determination demands that the iron be removed before precipitating with barium chloride. Methods of oxidization may differ, at least in a material of low sulphur content. The writer secured his result by oxidizing with potassium chlorate and nitric acid. A second evaporation with HCl removed the nitric acid. Without taking out the silica a double ammonia separation of iron and alumina was made, and after the addition of a little HCl the sulphur was precipitated by adding barium chloride drop by drop to the boiling solution. This method, particularly the adding of the barium chloride in this manner, is essential to the securing of accurate results. As a check on the above method several determinations were made by the fusion method of oxidization, and these checked the first given method exactly. Mr. Rueger also checked his result by the fusion method.

Of the results that may be considered satisfactory Nos. 3, 6, 7, 8, 9, 10, 12 and 14 are reported, with Nos 6 and 8 as the best. No. 6 oxidized with potassium chlorate and nitric acid, but failed to state whether the iron was removed, but it is assumed not. No. 8 used the same method as the writer. He secured one higher result by not throwing out the iron. No. 3 also used the same method. No. 7 used a rapid smelter method. Such methods vary, hence we are at a loss to know how his figure was obtained. No. 9 used the method suggested by the writer. No. 10 oxidized with potassium chlorate and nitric acid, but made only one evaporation to remove the nitric acid. The iron was left in the solution. No. 12 used the method of the writer. No. 14 likewise.

No. 2, 4, 11 and 13 are not satisfactory, but No. 13 is not far from some so considered. No. 2 used the same method as the writer, but seems to have failed in some detail. No. 4, while he does not positively so state, probably used the aqua regia method of oxidization. Sulphur in this method is liable to escape oxidization unless the analyst has had considerable experience. No. 11 oxidized with potassium chlorate and nitric acid, but did not remove the iron. No. 13 used the ordinary Colorado practice, whatever that may be. It can readily be seen that the removal of the iron gave the most uniform results. True it is that one or two secured good results without taking this step, but high results are the rule. The subject is a difficult one to discuss, owing to the variable results. The method of precipitating with the barium chloride is of greater importance than is generally considered, and unless strict adherence is held to the standard method, which has been published several times, good results will not be the rule. The determination of sulphur in simple materials should not be considered difficult. It is more of a case where exact adherence to tried methods and manipulations is necessary.

Zinc is the element in which the writer expected to find the greatest variation. A mere glance will show that he was not disappointed. The chief cause of the lack of agreement seems to lie in the use of the Von Shulz and Low method. For accurate work this method is worse than useless, and for even satisfactory results it is doubtful if the method so serves the majority of chemists. It has been the accepted method for so long that the writer's statement may be considered too strong.

A perusal of the results reported to the committee on methods of zinc analysis will soon convince the closest friend of Von Shulz and Low's method what may be expected in everyday work. The estimation of zinc by any method is difficult, owing to the small amount of iron which seems almost impossible to remove. Whether it is estimated as sulphide or carbonate a little iron is almost invariably found with the ignited precipitate, and a correction is difficult. The method of Mr. W. George Waring is said to give accurate results. It consists in precipitating the zinc as sulphide in a formic acid solution and under pressure. This method, in the writer's hands, has failed to give a precipitate free from iron, but this may be due to some defect in his manipulation. However the zinc is finally estimated, Mr. Waring's method is most decidedly the best for a preliminary separation, as it is not necessary to first remove the iron.

Of the ten results reported, Nos. 3, 10 and 13 are the best. Nos. 2, 9, 12 and 14 are fair but hardly satisfactory. Nos. 4, 7 and 8 are very poor. The writer's result was obtained by a preliminary separation of the zinc as sulphide in a formic acid solution, but without the use of a reducing agent other than the hydrogen sulphide employed for the precipitation. No pressure was employed. The zinc was then separated from the copper and lead with hydrochloric acid, the little iron remaining taken out by a basic acetate separation and the zinc finally precipitated as carbonate. The paper containing the moist precipitate was ignited in a platinum crucible at a very low heat until the carbonaceous matter was entirely destroyed, and after the addition of a few drops of nitric acid and careful drying ignited over the blast for one-half hour. The results were checked by using a porcelain crucible and careful removal of the precipitate from the filter paper. This method is somewhat lengthy, but accuracy was the aim. In view of the writer's success with the samples sent out by the committee on uniformity in zinc analysis, he believes his results correct.

Of the three good results reported, No. 3 precipitated as sulphide and ignited. He checked his result by dissolving this ignited precipitate and reprecipitating as carbonate and ignition to oxide. His preliminary separation of iron and alumina was repeated three times by ammonia. The objection to the use of ammonia is that no matter how many times the separation is made hydrogen sulphide will generally show zinc in the filtrate. No. 10 used the Von Shulz and Low method and secured acceptable results, although a little high. No. 13 used a Colorado method, which means the same as No. 10 probably.

Of the fair results No. 2 made a preliminary separation of the iron and alumina by the acetate method followed by two ammonia separations. The filtrate was then made acid with acetic and the copper and zinc thrown out as sulphides. After separating the copper, the zinc was precipitated as carbonate, ignited to oxide and weighed as such. In view of the fact that his result is high it is probable that a little iron ran through the filter paper in the final acetate separation. No. 12 used practically the same method. His result is low, due, probably, to an incomplete acetate separation. No. 14 used a method differing but little from Von Shulz and Low's. He believes his modification gives accurate results.

Of the very poor results No. 4 was obtained by Von Shulz and Low's method. In view of the fact that this chemist had difficulty with practically the whole of his work it would hardly be fair to lay the blame for poor work on the method used. No. 8 used the same method as did No. 2. It is evident that his zinc precipitate was badly contaminated with iron. It would appear that a majority of those using a gravimetric method did not take the precaution to know that the zinc, or a portion of it, was not retained with the precipitates of the other bases. Not enough results are at hand to criticize the Von Shulz and Low method. The method will undoubtedly receive its share of attention in the reports of the committee on uniformity in zinc analysis.

It was greatly desired in this work to secure information regarding the make, quality and ash content of the various filter papers used, but no satisfactory reports were made.

The writer, on several occasions, has found a so-called high-grade filter paper to be but little better than a very cheap one. Often he has found such paper to contain two and three times the quantity of ash which the makers claim. Such papers may produce an error up to one-tenth of a per cent. The writer believes it a good plan to determine the ash in each new purchase if they are to be used for accurate work.

A question brought up by one correspondent, although not a contributor, is regarding the probable error in sampling and parceling the various samples. This might prove a clever device for excusing a poor analysis were it not for the fact that a majority of chemists are honest. This slag sample was taken by the writer and occupied but a few seconds in the taking. A sample high in copper was purposely chosen. It was mixed in its coarse, granulated condition, then coarsely ground and again mixed by sieving. This grinding, mixing and sieving was repeated until the desired fineness was secured. Consequently, the writer believes that all samples had exactly the same composition. Furthermore, much of his work was done on different samples, as many as five different samples being used on the iron alone.

In concluding this study the conclusion must be drawn that failure to secure satisfactory results, in several instances, is due to incompetence, neglect, over confidence, or all three.

Our methods for the determination of silica and copper have been brought to a high degree of accuracy, and it is not probable that material improvement will be made beyond the use of better glassware and chemicals. The writer has several times stated that the methods of teaching were at fault, and his statement has been abundantly borne out through numerous letters received from his coöperators. A professor of chemistry, whose hobby is organic chemistry, is liable to be a poor teacher of inorganic, and especially of analytical chemistry. Too often he is desirous of following out an interesting derivative, and in such cases his classes in inorganic are taught by inexperienced instructors. The student graduates, and in turn becomes an instructor. While this man's theoretical knowledge may be unquestioned something more is required to make a working analytical chemist. The writer knows of more than one case where exactly this condition of affairs exists and every year it becomes worse. In short, too much is delegated to an instructor whose training is but little above that of the student he is endeavoring to teach. Professors are frequently employed whose chief recommendation is a doctor's degree.

Nothing, in the writer's opinion, is doing more harm to the teaching profession than the indiscriminate publishing of student theses in the leading chemical publications. It makes no difference whether the work is done by a man seeking a master's or a doctor's degree. This applies particularly to applied chemistry. False conclusions, the result of poor analytical work, are often drawn, and these must be reflected to the school from which they emanated. Not only that but such articles are often a source of annoyance to the working chemist who may wish to apply the conclusions. The writer trusts that these remarks will not offend the teaching profession to whom he owes valuable training and inspiration.

Owing to a press of other work the writer is compelled, for the immediate present, to give up this coöperative work, but he looks forward to the time when it can be again taken up and pushed to a point where greater good may come of it.

Ducktown Sulphur Copper & Iron Co., Ltd., Isabella, Tenn.

The Microstructure of Silicon and Alloys Containing Silicon.*

By A. B. Albro.

(*Concluded from page* 426.)

Ferro-Silicon.

In the steel industries silicon is used in the form of an alloy with iron under the name of ferro-silicon.

With the exception of two samples purchased from Messrs. Eimer & Amend, New York, all ferro-silicons used were made especially for this investigation. In these the silicon content ranged from 30 to 88 per cent, but as ferro-silicon is generally sold under three classifications, viz.: 25 to 30, 50 to 55, and 70 to 75 per cent silicon content, only these grades of the alloy are shown and described herein.

The physical structure of these grades of ferro-silicon is very interesting. As the percentage of silicon increases from 30 to 50 the brittleness of the alloy increases rapidly, and the 50 per cent alloy can be crushed between the fingers to octahedra about ½ a millimeter on an edge. After 50 per cent is passed the ease of fracture remains practically the same in the vertical plane, but becomes more and more difficult in the horizontal plane. After 75 per cent is passed the fracture becomes more difficult in both planes until the peculiarities noted with 92 to 98.8 per cent silicon are attained.

In the fracture of ferro-silicon containing 60 per cent of silicon, plates were obtained measuring 2 centimeters along their vertical axis, with a width of 1 centimeter and a thickness of from 1 to 2 millimeters.

The etching of the ferro-silicon sections was done by immersing the sample in dilute mixed hydrofluoric and nitric acids for 10 seconds; washing and drying and then immersing in concentrated *aqua regia* until details were brought out clearly.

As a basis of comparison, Fig. 21 shows a practically carbonless area of Swedish iron magnified 118 diameters, with vertical illumination.

Fig. 22 shows 50 per cent ferro-silicon magnified 8 diameters, but does not give the detail presented by this section to the unaided eye, as it then showed distinct laminations running at various angles in different grains.

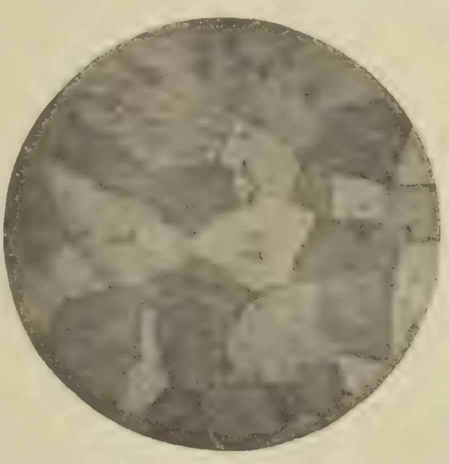

FIG. 21.—CARBONLESS IRON V × 118.

Fig. 23 is 52 per cent ferro-silicon, 8 diameters, and shows the large grain formation characteristic of silicon. These grains, like those of silicon, are plainly visible to the unaided eye. The incipient fractures can also be seen.

Fig. 24 shows ferro-silicon having 75 per cent silicon content, 8 diameters, and shows the approach to the structure of pure silicon. The matrix seems to be composed of octahedra, with incipient fractures extending in all directions between them.

No figure is shown of 32 per cent ferro-silicon at 8 diameters, owing to the lack of detail. With a magnification of 34 diameters the details of its structure begin to appear, as is seen in Fig. 25. Irregular areas, surrounded by wide channels which have been etched away, are characteristic of alloys of about this percentage. Continued etching brings out this structure more clearly, but at the expense of flatness of field.

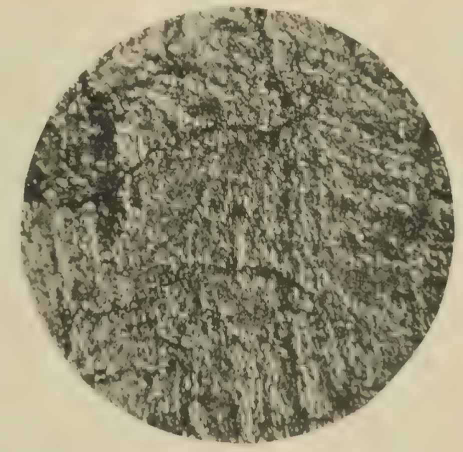

FIG. 22.—50 PER CENT FERRO-SILICON × 8.

Fifty per cent ferro-silicon, 34 diameters, is shown in Fig. 26. Owing to its extremely pulverulent nature it was impossible to obtain a plane surface for examination, but this figure shows its characteristic crystalline structure, and many octahedra can be distinguished despite the unevenness of the surface and consequent lack of sharpness in the photograph.

Fig. 27 shows 52 per cent ferro-silicon, 34 diameters. It presents a rough-grained matrix, showing some octahedra and laminæ of silicon. These latter show white on the dark background of the matrix.

With the same magnification, 34 diameters, the 75 per cent alloy possesses a very characteristic structure, as is seen in Fig. 28. Numerous irregular-shaped interlocking grains of silicon are seen with the octahedra of the alloy between them and sometimes extending in laminæ of considerable length.

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* A paper read before the American Electrochemical Society, Boston meeting.

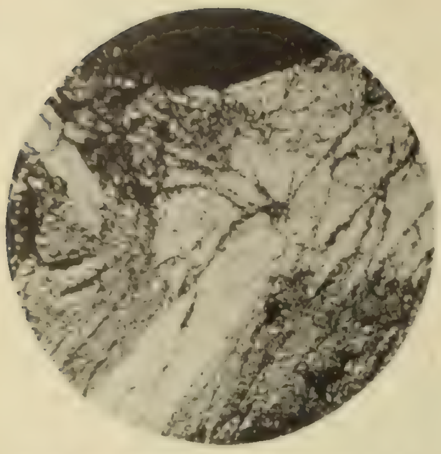

FIG. 23.—52 PER CENT FERRO-SILICON × 8.

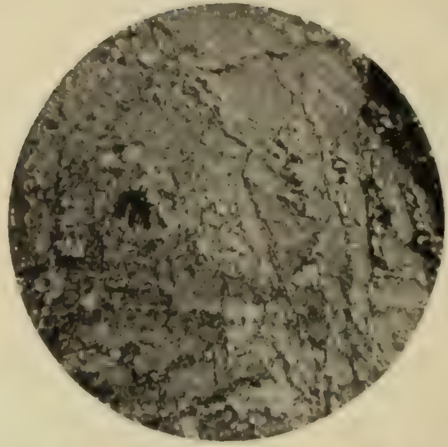

FIG. 24.—75 PER CENT FERRO-SILICON × 8.

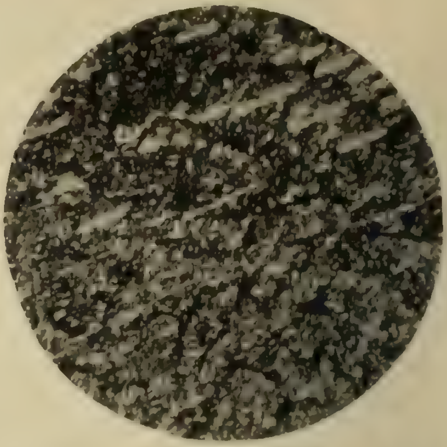

FIG. 25.—32 PER CENT FERRO-SILICON × 34.

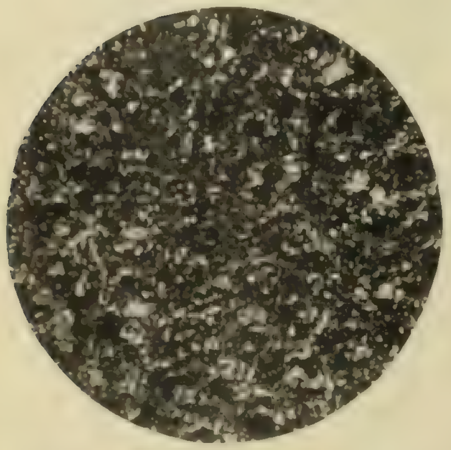

FIG. 26.—50 PER CENT FERRO-SILICON × 34.

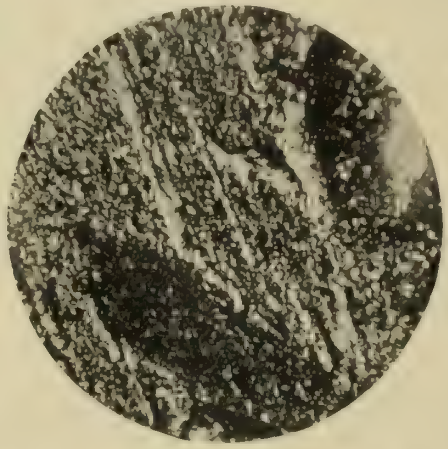

FIG. 27.—52 PER CENT FERRO-SILICON × 34.

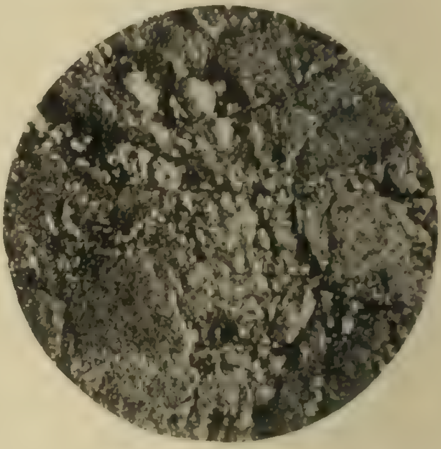

FIG. 28.—75 PER CENT FERRO-SILICON × 34.

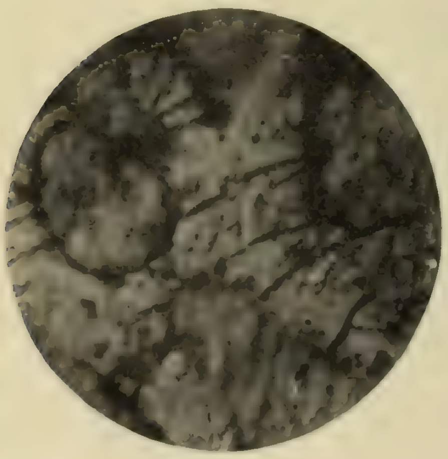

FIG. 29.—32 PER CENT FERRO-SILICON V × 167.

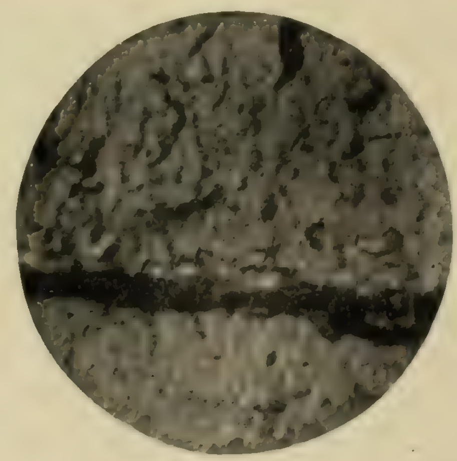

FIG. 30.—52 PER CENT FERRO-SILICON V × 167.

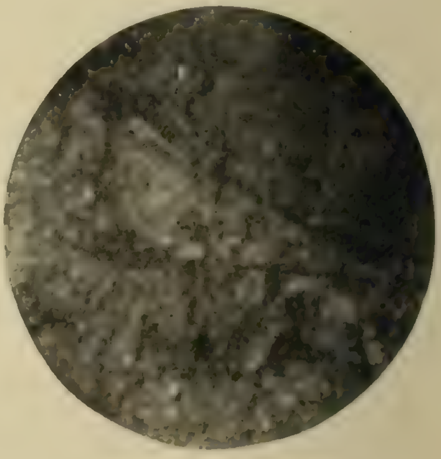

FIG. 31.—75 PER CENT FERRO-SILICON V × 167.

With vertical illumination and a magnification of 167 diameters, 32 per cent ferro-silicon exhibits a very interesting structure, as shown in Fig. 29. The etching was carried far enough to bring out the details of the matrix, and, as a result, the irregular prominent grains cannot be brought into focus at the same time. The matrix shows somewhat the structure of impure iron, and seems to have grains of silicon imbedded in it at irregular intervals. The prominent irregular-shaped areas, when brought into focus, show the same structure as that of 52 per cent ferro-silicon with the same magnification, 167 diameters, as shown in Fig. 30.

In this we have the surfaces of two grains and the dividing line between them. The section seems to be homogeneous in structure, with the exception of one or two large irregular grains standing in clear relief above the surrounding material. These probably are grains of silicon.

Fig. 31 shows 75 per cent ferro-silicon, also 167 diameters. Here we have, in the band running through the center of the field, a structure resembling that shown in the 52 per cent material of the last figure, while the surrounding portions of the field have a structure resembling impure silicon. The fact that the central band is etched slightly deeper than the other portions would tend to prove this.

These figures show that the structure of iron is decidedly different from that of silicon or any silicide of iron which may exist, and that the presence of silicon or a silicide in iron

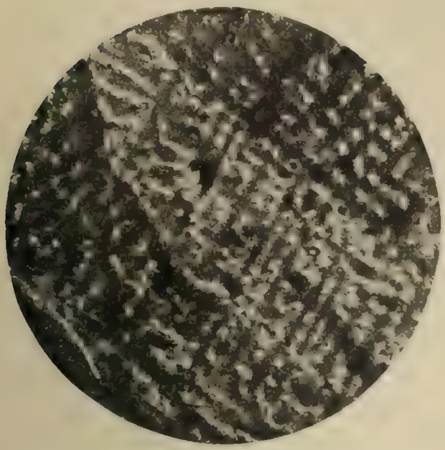

FIG. 32.—Si 2.3 PER CENT, Cu 97.7 PER CENT × 34.

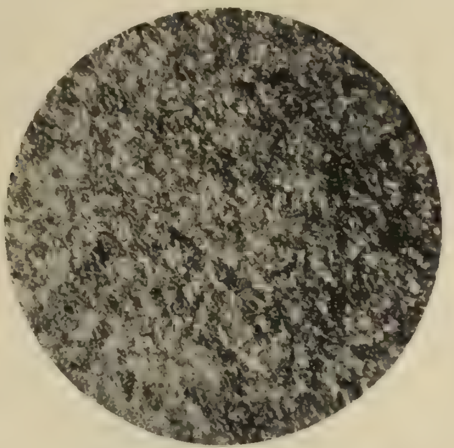

FIG. 33.—Si 10.12 PER CENT, Cu 87.0 PER CENT × 34.

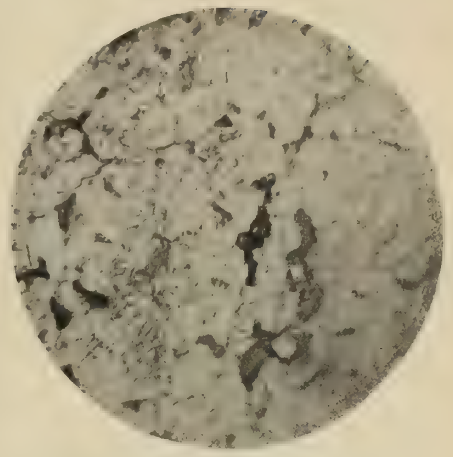

FIG. 34.—Si 10.12 PER CENT, Cu 87.0 PER CENT v × 167.

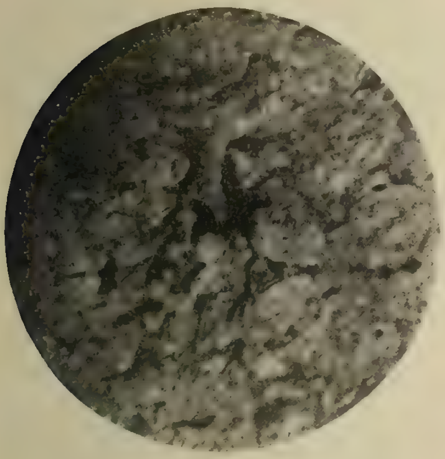

FIG. 35.—Si 10.12 PER CENT, Cu 87.0 PER CENT v × 167.

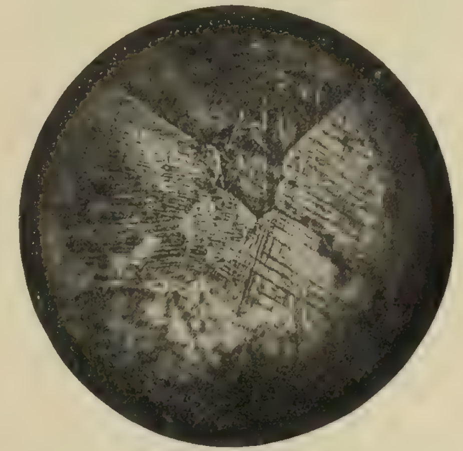

FIG. 36.—Si 2.3 PER CENT, Cu 97.7 PER CENT × 8.

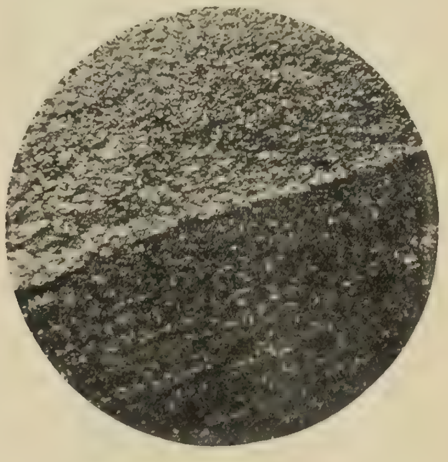

FIG. 37.—(SEE TEXT) × 34.

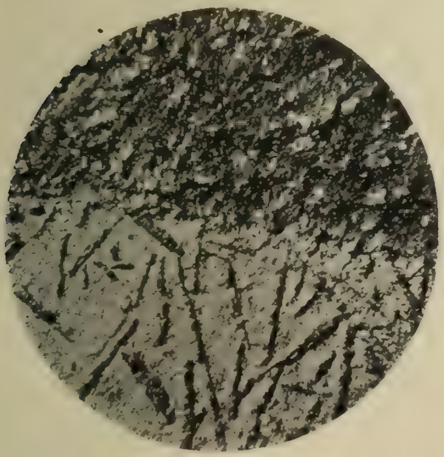

FIG. 38.—(SEE TEXT) × 34.

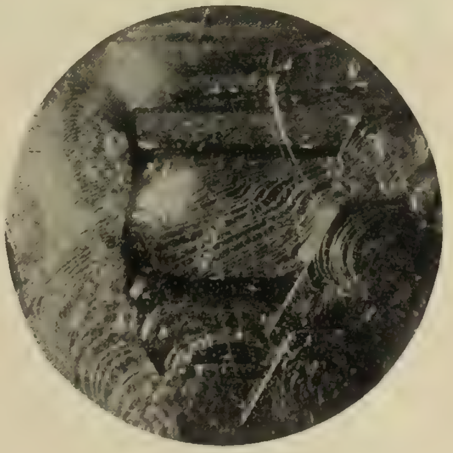

FIG. 39.—SiC CRYSTALS × 8.

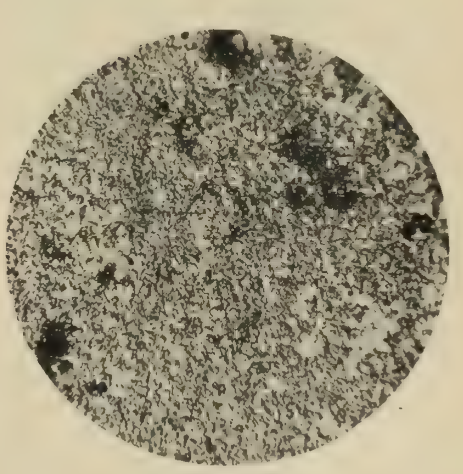

FIG. 40.—Si 80 PER CENT, SiC 20 PER CENT × 8.

should be distinguishable in iron as clearly as that of phosphorus or sulphur. Further, the composition of ferro-silicon can be determined, at least roughly, by means of the microscope with very little practice, and especially when one has only to distinguish between two samples whose silicon content differs by from 5 to 10 per cent. The 30 per cent, 50 per cent, and 75 per cent can readily be distinguished from each other by their fractures.

Copper-Silicon.

In studying this alloy three grades of material were used, containing respectively 2.3 per cent, 10.12 per cent and 23.4 per cent copper. The 10.12 per cent silicon copper was furnished from stock purchased by the Westinghouse Electric & Mfg. Co., while the other two were specially made.

In regard to physical properties these samples differed greatly, the low percentage alloy having almost the same appearance and malleability as ordinary ingot copper, while the high percentage alloy was steel-gray in color, hard and very brittle. The 10.12 per cent alloy possessed properties intermediate between these two, being silver white in color when freshly fractured and turning to a straw-yellow on standing 24 hours. The fracture was markedly conchoidal, and in some cases in breaking a piece from the mass it would separate with two and sometimes three cleavage surfaces, all of which were conchoidal.

The low percentage alloy was etched with dilute, the other two samples with concentrated *aqua regia*.

Fig. 32 shows 2.3 per cent copper-silicon, magnified 34 diameters after prolonged etching. In the large grains two sets of laminæ at right angles are noticeable. In the smaller and darker grain the laminæ are at a different angle and somewhat indistinct. The surface appearance of this alloy to the unaided eye was very striking, presenting irregular polyhedral grains, sometimes 2 or 3 millimeters across.

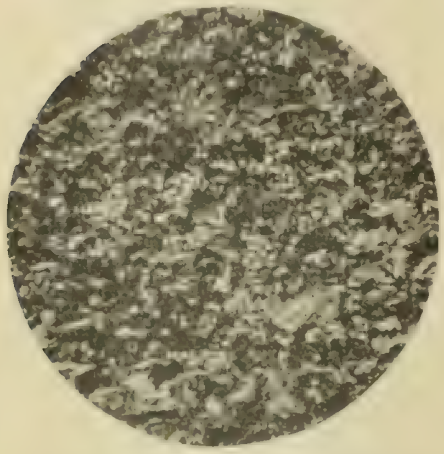
FIG. 41.—Si 80 PER CENT, SiC 20 PER CENT × 34.

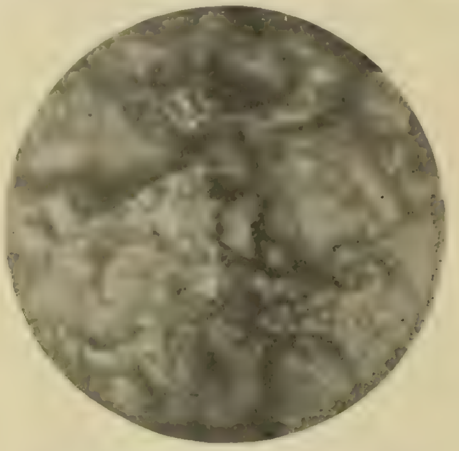
FIG. 42.—Si 80 PER CENT, SiC 20 PER CENT × 167.

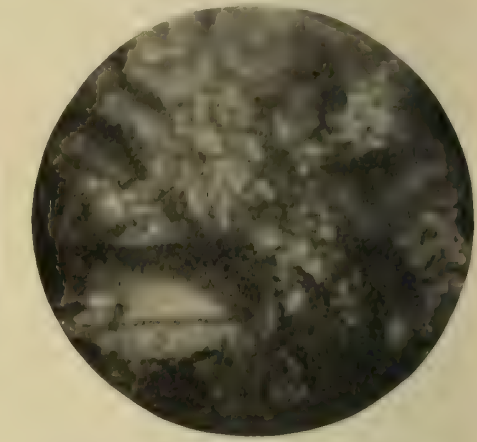
FIG. 43.—Si 80 PER CENT, SiC 20 PER CENT × 167.

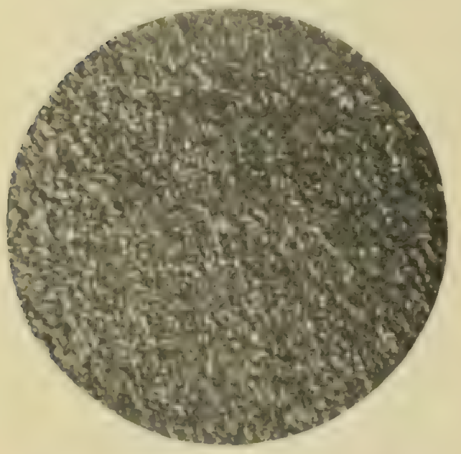
FIG. 44.—CARBOSILICON V × 34.

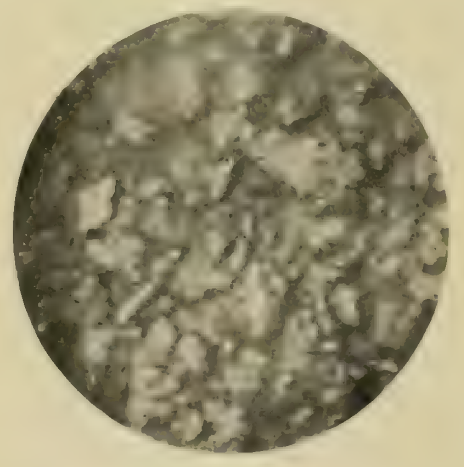
FIG. 45.—CARBOSILICON V × 167.

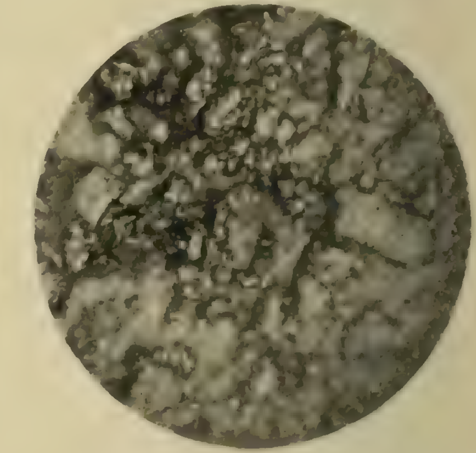
FIG. 46.—CARBOSILICON V × 167.

Fig. 33 shows the 10.12 per cent alloy, also 34 diameters. The white areas show the presence of silicon, as they are composed of silica, which results from the action of *aqua regia* on copper silicide, Cu_2Si.

Figs. 34 and 35 show the same sample with vertical illumination and a magnification of 167 diameters. The dark areas are pits caused by the action of the etching fluid, and are, I believe, copper areas. The main body of the section is made up of minute irregular grains overlapping each other to some extent. Some of these are eaten away more rapidly than others, and are a straw-yellow color, while the latter appear white.

The 2.3 per cent alloys, after being repolished and very slightly etched with dilute *aqua regia*, had, when magnified 8 diameters, the appearance shown in Fig. 36. This sample shows some scratches on its surface, but has, nevertheless, the same laminated appearance shown with higher magnification in Fig. 32.

As the high percentage alloy showed nothing in addition to that already shown by the 10.12 per cent copper-silicon, no figures are presented.

The Action of Silicon Towards Carbon and Silicon Carbide.

The facts recorded in this division of the subject are presented, because of the peculiar microstructures and physical properties observed.

Silicon may be melted in a carbon container without introducing carbon or silicon carbide. Fig. 37 shows the junction formed between silicon and Acheson graphite under a magnification of 34 diameters. The dividing line is sharply defined, and the silicon is homogeneous in structure, showing no areas in relief, which are always seen when silicon carbide is present.

Fig. 38 shows a cross-section of the tip of an arc electrode used in fusing silicon. In this case the silicon was subjected to a very high temperature, and the carbon to the action of silicon vapor. The silicon here, in distinction from the former sample, shows the presence of its carbide, as indicated by the long, narrow, raised portions. However, it will be noticed that the dividing line between the graphite and silicon is still sharply defined, but analysis of the silicon shows no free carbon content, and the graphite can be cleanly cut away from the silicon surface. Magnification, 34 diameters.

Solid carbon and molten silicon do not necessarily react. In the large number of samples examined, made under the greatest range of temperatures, the silicon has never contained any free carbon.

The characteristic silicon carbide crystal is described as a hexagonal plate; in other words, a hexagonal prism in which the shortest axis is the distance between the two hexagonal surfaces. With silicon crystals this axis is generally the longest one, and, when crystals of silicon have been found whose form was that of the accepted silicon carbide crystal, they have been called silicon pseudomorphs of silicon carbide, despite the fact that in the dissociation of the carbide at high temperatures the carbon, not the silicon, remains.

The author has examined many grottoes in silicon in which hexagonal plates have been found, measuring from 2 to 20 millimeters on the longest diagonal. These on analysis were found to be pure silicon, containing no trace of carbon or silicon carbide.

From the powerful influence on the structure of metals in general, it is possible that it profoundly influences the crystal type of its carbide SiC, about 70 per cent of which is silicon, and that hexagonal plates of silicon, instead of being silicon carbide pseudomorphs, are merely another form for the hexagonal prisms or octahedra observed as characteristic of silicon.

Fig. 39 shows crystalline silicon carbide magnified 8 diameters. The laminated structure is very clearly seen together with several hexagonal plates. This sample consisted of one large crystal, to which several smaller ones were attached.

Silicon containing about 20 per cent of silicon carbide presents the peculiar structure in Fig. 40. The white portions contain amorphous silicon carbide, while the dark areas are silicon. Magnification, 8 diameters.

Fig. 41 shows this same magnified 34 diameters, and reveals a softer matrix from which irregular grains stand out in relief. This structure is shown equally well by polishing in bas-relief on cloth or by etching with mixed nitric and hydrofluoric acids.

The next two slides (Figs. 42 and 43) show different sections of the same sample magnified 167 diameters. The large, prominent grains are a silicon-silicon carbide material; while the hexagonal plates are crystals of carbide which have crystalized from solution of the amorphous material in molten silicon. It will also be noticed that the matrix has the characteristic silicon structure.

In addition to the above reaction, Dr. Potter discovered that when silicon and silicon carbide are saturated with each other the resulting material exhibits remarkable properties. Analysis proves this to contain the element and its carbide in proportions approximating very closely the formula—

$$SiC + Si = \text{(perhaps) } Si_2C$$

The material is non-porous, harder than corundum, and so tough that a 3-inch cube withstood heavy blows from a sledge on all its faces without fracture. It is evident that this substance, termed by its discoverer "carbo-silicon," is very different from either of its components in regard to toughness, and that the method of combining the element and its carbide has much to do with the properties of the resultant substance.

In comparison with the previous figures, which showed a solution of amorphous silicon carbide in silicon, Fig. 44 shows the structure of "carbo-silicon" under a magnification of 34 diameters. It will be seen to be much more homogeneous than the ordinary solution.

The next two figures (Figs. 45 and 46) show this material magnified 167 diameters. All portions of the mass have the same structure as the parts shown in sharp focus in these photographs. No characteristic silicon structure is visible, only that characteristic of silicon saturated with its carbide or vice versa. Its complete freedom from large grains and the absence of any symmetrical arrangement of the grains obviously accounts for its toughness.

It thus appears that silicon has a remarkable influence upon the structure of anything with which it enters into solution or combination, and further investigation would seem almost certain to be rewarded by the discovery of novel and valuable properties which are particularly alluring in view of the certain cheapness of silicon as soon as a demand develops.

These studies were made at the suggestion of Dr. Potter, and I wish in conclusion to express my great indebtedness to him for his direction and counsel throughout.

NOTE.—After the completion of the photomicrographs for this paper, two samples of ferro-silicon were received from the Roessler & Hasslacher Chemical Co., New York, according to their analysis, containing 57.78 and 75.60 per cent of silicon respectively.

Their appearance, fracture and microstructure are the same as samples of like composition made especially for this paper.

The microstructure of the 75.60 per cent material is characteristic, while that of the 57.78 material shows the distinguishing octahedra of 50 per cent ferro-silicon, with intergranular areas of silicon.

Metallurgical Calculations.—X.

By J. W. RICHARDS, PH. D.

Professor of Metallurgy in Lehigh University.

ARTIFICIAL FURNACE GAS. (Continued.)

2. MIXED GAS PRODUCERS.

This class of producers are those most commonly used. In them a moderate amount of steam or vapor of water passes with the air into the fire, and is decomposed, producing carbon monoxide and hydrogen gases by the reaction:

$$\begin{array}{ccccccc} \text{I} & & & & \text{I} & & \text{I} \\ H_2O & + & C & = & CO & + & H_2 \\ 18 & & 12 & & 28 & & 2 \end{array}$$

which may be read as follows: One volume of steam forms one volume of carbon monoxide and one volume of hydrogen; or eighteen parts by weight of water vapor act upon twelve parts of solid carbon, producing twenty-eight parts of carbon-monoxide and two parts of hydrogen. If we speak of kilograms as the above weights, then we can call each "volume" spoken of 22.22 cubic meters; or if we call the weights ounces avoirdupois, each "volume" represents 22.22 cubic feet.

The water vapor is admitted either automatically, as in the old Siemens type of producer, where water was run into the ash pit to be evaporated by the heat radiated from the grate or by hot ashes falling into it, or as in the modern water-seal bottom producer, where the ashes rest upon water in a large pan, and so are continually kept soaked by capillary action; or, finally, steam is positively blown under the grate, either as a simple steam jet or, more economically, by using it in a steam blower, so as to have it produce by injector action an air blast sufficient to run the producer. In the latter case the proportions of air and steam may be regulated with precision, and the blast action produced makes the production and delivery of the gas practically independent of chimney draft. The use of steam also rots or disintegrates the ashes, preventing or breaking up masses of clinker, and so facilitating the removal of the ashes.

Steam or water vapor cools down the fire in the producer, so that it runs cooler; at the same time gas is produced which is rich in hydrogen, and, therefore, of higher calorific power. This saves unnecessary waste of heat in the producer, and increases the efficiency of the gas in the furnace in which it is burnt. The scientific reasons for these facts are to be found in a consideration of the thermochemistry of the reaction by which steam is decomposed.

$$\begin{array}{lll} H_2O + C = CO + H_2 & & \\ -69{,}000 \quad\quad + 29{,}160 & = -39{,}840 & \text{Calories.} \end{array}$$

This would be the deficit in decomposing 18 kilograms of water if it starts in the liquid state. If, however, it is used as steam at 100° C., each kilogram contains 637 Calories of sensible heat, making $18 \times 637 = 11{,}466$ Calories altogether, leaving the deficit 28,374 Calories, or the deficit

= 1,576 Calories per kilogram of steam decomposed.

= 2,364 Calories per kilogram of carbon thus burnt.

In making this calculation it might be objected that the steam used is often at three or four atmospheres pressure, and its temperature, therefore, over 100° C.; but it must not be overlooked that this steam expands suddenly to atmospheric tension, and that in so doing it cools itself to an amount roughly proportional to its excess pressure, so that the expanded steam at atmospheric pressure is usually close to 100° C. When mixed with air, the temperature of the mixture is usually below 100°, some 40° to 50° C., and in this condition some of the steam is possibly condensed to fog, but it must not be forgotten that the heat of condensation thus gives out has been absorbed in raising the temperature of the admixed air, and therefore goes as sensible heat into the bed of burning fuel. For the purpose of calculation, we will, therefore, be very nearly right in assuming that we are dealing in each case with steam at 100° C., requiring the above calculated deficits to be made up, in order for the decomposition to proceed.

It will be evident that any heat thus used in the producer must be sensible heat of the hot carbon, which, if not so used, would be lost as sensible heat in the gases produced or radiated and conducted away from the producer. The basic process of the running of the producer is the burning of carbon to carbon monoxide, liberating 2,430 Calories per kilogram of carbon, which is 2,430 ÷ 8,100 = 30 per cent exactly of the calorific power of the carbon. This heat, if no water vapor gets into the producer, is lost as sensible heat of the hot gases and by radiation and conduction, and is thus largely a dead loss. In Problem 14, for instance, these items figured out 28.25 per cent of the calorific power of the coal used. Now, the facts are, that while small producers need that much heat to keep them up to working temperature, large producers need very much less, and run far too hot if no steam is admitted to check the rise of temperature. In the largest producers the sensible heat in the gases, plus the losses by radiation and conduction, does not exceed 10 per cent of the calorific power of the fuel. Out of the 30 per cent of the calorific power of the carbon inevitably generated, only some 10 per cent is, therefore, needed to supply the losses in a large producer, leaving 20 per cent applicable to decomposing steam. We therefore have:

Per 1 Kilo. of Carbon Gasified.

Heat generated	2,430	Calories
Necessary loss in a large producer...........	810	"
Useful for decomposing steam...............	1,620	"
Required to decompose 1 kilo. steam at 212° F. (69,000 — 11,466) ÷ 18 =	3,196	"
Maximum steam decomposable 1,620 ÷ 3,196 =	0.507	kilo.

Calculation, therefore, shows that about one-half of a unit weight of steam is the maximum which can be used per unit weight of carbon burnt to carbon monoxide, consistent with keeping the producer at proper working temperature. This would be equal to

0.088 kilos. of steam per kilo. of air used.
0.088 pounds of steam per pound of air used.
0.114 kilos. of steam per cubic meter of air used.
0.007 pounds of steam per cubic foot of air used.

The above discussion is on the assumption that the bed of fuel in the producer is thick enough, and its temperature always high enough, to burn all the carbon to carbon monoxide. Such is only an ideal condition, for the irregular charging, descent and working of the fuel always allow of some carbon dioxide being produced, and the best regular gas usually contains 1 to 5 per cent of dioxide, representing some 3 to 20 per cent of the total carbon oxidized in the producer.

Assuming that on an average 10 per cent of the carbon oxidized inevitably forms carbon dioxide, we can calculate under these more usual conditions how much steam can be used, because for each kilogram of carbon oxidized, 0.1 kilo. then gives us 8,100 Calories per kilogram instead of 2,430 in the producer, a surplus of

$$0.1 \times (8{,}100 - 2{,}430) = 567 \text{ Calories}$$

over the conditions when only monoxide is formed. Instead of the 1,620 Calories available for decomposing steam we will now have 2,187 Calories, which will decompose

$$\frac{2{,}187}{3{,}196} = \mathbf{0.684} \text{ kilo. of steam.}$$

Which reckoned on the air used would be

0.108 kilos. of steam per kilo. of air used.
0.108 pounds of steam per pound of air used.
0.140 kilos. of steam per cubic meter of air used.
0.009 pounds of steam per cubic foot of air used.

The above proportions are those which cannot practically be exceeded, if producing gas as low as is usually possible in carbon dioxide.

[Working the producer comparatively cold, with excess of steam, much larger proportions of carbon dioxide are formed and correspondingly larger proportions of steam are decomposed, but this manner of working is abnormal, is not unusual, and will be discussed under the next heading, treating of Mond gas.]

Making gas containing any given proportion of CO^2 to CO, by volume, the ratio thus obtained is identical with the proportionate weight of carbon oxidized to CO^2 and CO in the producer, and by the application of the principles just described it can be calculated how much steam can be used per unit of carbon oxidized, making proper allowance for losses by radiation, etc.

Illustration: A Siemens producer with chimney draft produced gas containing, by volume, 4.3 per cent CO^2 and 25.6 per cent CO. How much steam could be used per pound of air used, assuming 50 per cent of the heat generated by the oxidation of the carbon to be needed to run the producer?

Solution:

$$\text{Per cent of carbon burnt to } CO^2 = \frac{4.3}{4.3 + 25.6} = 14.4 \text{ per cent}$$

Heat generated by C to CO^2 = 0.144 × 8,100 = 1,166 lb. Cal.
Heat generated by C to CO = 0.856 × 2,430 = 2,080 "
Heat generated per kilo. of C = 3,246 "
Heat lost by radiation, etc. (50 per cent) = 1,623 "
Heat available for decomposing steam = 1,623 "

$$\text{Steam decomposable} = \frac{1{,}623}{3{,}196} = \mathbf{0.508} \text{ pounds.}$$

The above is expressed per pound of carbon oxidized, but the same proportion is true per kilogram. The air required per pound of carbon oxidized is found from the oxygen required to form CO and CO^2:

Oxygen for C to CO^2 = 0.144 × 8-3 = 0.384 lbs.
Oxygen for C to CO = 0.856 × 4-3 = 1.141 "

Total = 1.525 "
Air = 6.608 "

$$\text{Volume of air} = \frac{6.608}{0.0808} = 81.8 \text{ cu. feet}$$

$$\text{Steam per cubic foot of air} = \frac{0.508}{81.8} = \mathbf{0.006} \text{ lbs.}$$

$$\text{Steam per pound of air} = \frac{0.508}{6.608} = \mathbf{0.077} \text{ ''}$$

Air required per pound of steam = **13.0** "

It should easily be seen that whatever heat is absorbed in the producer in decomposing steam, is entirely recovered when the hydrogen thus produced is burnt and steam is reproduced. If then, in any case, it is possible to absorb in the producer, in the decomposition of steam, an amount of heat equal to say, 20 per cent of the total calorific power of the fuel, then that 20 per cent is regained and capable of being utilized when the hydrogen so produced is burnt in the furnace. In other words, 20 per cent *less* of the calorific power of the fuel will be lost in the process of conversion into gas in the producer, and 20 per cent *more* will be obtained in the burning of the gas when it is used. The great advantages of using steam judiciously are thus clearly evident.

Problem 15.

R. W. Hunt & Co. report the following tests made of the running of a Morgan continuous gas producer. Coal used, "New Kentucky" Illinois coal, run of mine. Composition:

Fixed carbon	50.87	per cent
Volatile matter	37.32	"
Moisture	5.08	"
Ash	6.73	"
	100.00	

The ultimate composition was:

Total carbon	69.72	"
Hydrogen	5.60	"
Nitrogen	2.00	"
Total sulphur	0.94	"
Oxygen	11.00	"
Moisture	5.08	"
Inorganic residue (less sulphur)	6.66	"

The ash, on combustion, contains 1.12 per cent of its weight of sulphur (as FeS); the ashes obtained from the producer contain 4.66 per cent of unburnt carbon.

The gas produced, dried, contained by volume:

Carbon monoxide (CO)	24.5	per cent
Marsh gas (CH^4)	3.6	"
Ethylene (C^2H^4)	3.2	"
Carbon dioxide (CO^2)	3.7	"
Hydrogen (H^2)	17.8	"
Oxygen (O^2)	0.4	"
Nitrogen (N^2) (by difference)	46.8	"

[The moisture and sulphur compounds in the gas not having been determined, we can calculate the former, and, for the purposes of calculation, are justified in assuming the sulphur present in the gas as H^2S, and in subtracting it from the hydrogen. We will also assume that the moisture in the coal goes unchanged into the gases as moisture, and that all the steam used is decomposed. The 0.94 per cent of sulphur in the coal will furnish 0.86 per cent to the gases, because 6.73 × 0.0112 = 0.08 per cent will go into the ash as ferrous sulphide. The 0.86 pound of sulphur would produce 0.91 pounds of H^2S, equal in volume to 9.56 cubic feet per 100 pounds of coal used, or (since we will see later that 53.83 cubic feet of gas are produced per pound of coal) there will be 0.2 per cent of H^2S in the gases, leaving 17.6 per cent of hydrogen.]

On the basis of above data and assumptions:

Required: (1) The volume of gas produced per pound of fuel used in the producer.

(2) The weight of steam used per 100 cubic feet of air blown in, assuming the air dry.

(3) The proportion of the total heat generated in the producer which is utilized in decomposing steam.

(4) The percentage of increased economy thus obtained reckoned on the calorific power of the fuel.

(5) The efficiency lost by unburnt carbon in the ashes.

(6) The efficiency of the gas, burnt cold, compared with the coal from which it is made.

Solution: (1) Per pound of coal burnt, there remains in the ashes 0.0673 × (0.0466 ÷ 0.9534) = 0.0033 pounds of unburnt carbon, leaving 0.6972 — 0.0033 = 0.6939 pounds gasified. One cubic foot of gas, at 32° F., contains the following weight of carbon:

$$\begin{aligned}
\text{C in CO} &= 0.245 \times 0.54 \text{ ounces} \\
\text{C in } CH^4 &= 0.036 \times 0.54 \quad " \\
\text{C in } C^2H^4 &= 0.032 \times 1.08 \quad " \\
\text{C in } CO^2 &= 0.037 \times 0.54 \quad " \\
\text{Total} &= 0.382 \times 0.54 \quad " \\
&= 0.20628 \text{ ounces Av.} \\
&= 0.01289 \text{ pounds Av.}
\end{aligned}$$

Gas produced, measured dry, per pound of coal, at 32° F.:

$$\frac{0.6939}{0.01289} = 53.83 \text{ cubic feet.} \qquad (1)$$

(2) If the moisture of the coal is assumed to pass unchanged into the gas, as moisture, then all the hydrogen in the dry gas, in any form or combination, must have come either from hydrogen in the coal or in the air blast. The hydrogen in the coal is given as 5.60 per cent. The hydrogen in the gas is calculated as follows, per cubic foot:

$$\begin{aligned}
H^2 \text{ in } H^2 &= 0.176 \times 0.09 \text{ ounces.} \\
H^2 \text{ in } H^2S &= 0.002 \times 0.09 \quad " \\
H^2 \text{ in } CH^4 &= 0.036 \times 0.18 \quad " \\
H^2 \text{ in } C^2H^4 &= 0.032 \times 0.18 \quad " \\
\text{Total} &= 0.314 \times 0.09 \quad " \\
&= 0.02826 \text{ ounces Av.} \\
&= 0.00177 \text{ pounds Av.}
\end{aligned}$$

Hydrogen in gas from 1 pound of coal:

$$0.00177 \times 53.83 = 0.0953 \text{ pounds.}$$

Hydrogen from decomposition of steam:

$$0.0953 - 0.0560 = 0.0393 \text{ pounds.}$$

Weight of steam decomposed per pound of coal used:

$$0.0393 \times 9 = 0.3537 \text{ pounds.}$$

To express this weight relatively to the air blown in, we must calculate the air used per pound of coal, as follows:

Nitrogen in gas per cubic foot

$$\begin{aligned}
0.468 \times (14 \times .09) &= 0.5897 \text{ oz. Av.} \\
&= 0.036856 \text{ lbs. Av.} \\
\text{Per pound of coal} = 0.036856 \times 53.83 &= 1.9840 \quad " \\
\text{Subtract } N^2 \text{ in 1 pound of coal} &= 0.0200 \quad " \\
\text{Leaves } N^2 \text{ from air} &= 1.9640 \quad " \\
\text{Weight of air} = 1.9640 \times \frac{13}{10} &= 2.5532 \quad " \\
\text{Volume of air} = 2.5532 \times 16 \div 1.293 &= 31.61 \text{ cu. ft.}
\end{aligned}$$

Steam used per 100 cubic feet of air blown in:

$$= \frac{0.3537}{31.61} \times 100 = 1.119 \text{ pounds.} \qquad (2)$$

(3) The heat utilized in decomposing steam has been found to be 3,196-pound Calories per pound of steam at 212° F. We, therefore, have the heat so used per pound of fuel used:

$$0.3537 \times 3{,}196 = 1{,}130\text{-pound Calories.}$$

This quantity must now be compared with the total heat generated in the producer, and the latter quantity can be determinated in two ways: (1) We may subtract from the total calorific power of the coal the calorific power of the gas produced and of the unburnt carbon in the ashes; the difference must be the *net* heat generated in the producer, *i. e.*, the total heat generated minus that absorbed in decomposing steam. The total heat generated is the net heat thus calculated plus the heat absorbed in decomposing steam. (2) We may calculate the heat of formation of the CO and CO^2 in the gas, and assume that as the total heat generated in the producer. This method is not so accurate as (1).

The total calorific power of the coal is given as practically 7,747-pound Calories per pound, water formed being condensed, which would be decreased by the latent heat of vaporization, if the latter is assumed uncondensed. The deduction is 606.5 × [(0.056 × 9) + 0.0508] = 337-pound Calories, leaving 7,410-pound Calories as the practical metallurgical calorific power of the fuel.

Calorific power of the gas (dried) per cubic foot:

$$\begin{array}{llrrl} CO & 0.245 \times & 3{,}062 = & 750.2 & \text{ounce Cal.} \\ CH^4 & 0.036 \times & 8{,}598 = & 309.5 & \text{"} \\ C^2H^4 & 0.032 \times & 14{,}480 = & 463.4 & \text{"} \\ H^2 & 0.176 \times & 2{,}613 = & 459.9 & \text{"} \\ H^2S & 0.002 \times & 5{,}513 = & 11.0 & \text{"} \end{array}$$

$$\text{Total} = 1994.0 \text{ "}$$
$$= 124.6 \text{ pound Cal}$$

Calorific power of gas per pound of coal:

$$124.6 \times 53.83 = 6{,}707 \text{ pound Cal.}$$

Calorific power of carbon in ashes:

$$0.0033 \times 8{,}100 = 27 \text{ "}$$

$$\text{Sum} = 6{,}734 \text{ "}$$

Calorific power of 1 pound coal = 7,410 "

Net heat lost in conversion = 676 "

Used in decomposing steam = 1,130 "

Gross heat generated in producer = 1,806 "

Proportion of this utilized in decomposing steam

$$\frac{1{,}130}{1{,}806} = 0.626 = 62.6 \text{ per cent.} \qquad (3)$$

(4) We can state this result in another way, by saying that 1,806 ÷ 7,410 = 24.40 per cent of the calorific power of the fuel is generated in the producer, of which 1,130 ÷ 7,410 = 15.25 per cent is utilized to decompose steam, and 9.15 per cent is lost by radiation, conduction and sensible heat in the gases. The calorific power of the gases represents 6,707 ÷ 7,410 = 90.50 per cent of the calorific power of the coal of which 9.15 per cent, however, is clear gain from the employment of steam. Reckoning on the total calorific power of the coal, the increased economy from the use of steam is **9.15** per cent (4).

(5) The loss of calorific power by the unburnt carbon in the ashes is 27-pound Calories, or, on the whole heat available,

$$= \frac{27}{7{,}410} = 0.36 \text{ per cent.} \qquad (5)$$

This loss is exceptionally low, and may be very profitably compared with the analogous loss of 18.6 per cent occurring in the case discussed in Problem 14.

(6) This has already been calculated as

$$\frac{6{,}707}{7{,}410} = 90.50 \text{ per cent,}$$

on the assumption that the gases are burnt cold. If they are burnt hot, say issuing from the producer at 1200° F. (649° C.), and are burnt when at 1,000° F. (548° C.) their sensible heat at 1,000° F. will be added to their efficient heating power, and can be calculated with exactness, using the principle explained and used in requirement (6) of Problem 14. Under such conditions the total efficiency of the producer, reckoned on the calorific power of the coal used, approximates 95 per cent.

To be fair to everybody concerned, however, we must deduct from this the coal required to be burnt to raise the steam used. There is a little over one-third pound of steam used per pound of coal used in the producer. This requires in ordinary boiler practice $\frac{1}{3} \times \frac{1}{8} = \frac{1}{24}$ pound of coal, or about 4 per cent of the weight of fuel burnt in the producer. The results of the previous calculation must, therefore, be diminished in this proportion, if the steam has to be raised by burning coal under boilers. Under these conditions (6) become

$$90.50 \div 1.04 = 87 \text{ per cent efficiency.} \qquad (6)$$

If the steam can be obtained from waste gases of a blast furnace, or from the hot producer gases themselves (in case they are going to be burnt cold), then no such deductions need be made. It is very evident, however, that if 9.15 per cent greater efficiency is gained by burning 4 per cent more coal to raise the steam, that the net gain would be only 5.15 per cent. Even under these conditions it pays to use steam, because of the greater calorific intensity of the richer gas, the usefulness of the steam for supplying air blast, and the rotting of the clinkers therewith obtained.

[The next instalment will discuss Mond gas and water gas.]

Report of the Karlsruhe Meeting of the German Bunsen Society.

By H. Danneel, Ph. D., and J. K. Clement, Ph. D.

The following is the continuation of the report, the first part of which was published on page 327 of this volume:

REACTION KINETICS OF THE PROCESS OF ORGANIC CHEMISTRY.

A paper on this subject was presented by Prof. Dr. Goldschmidt, of Christiania. The organic chemist can ascertain analytically only the beginning and end products of a reaction. As to intermediate products he can only set up hypotheses. Information about intermediate products can be obtained from measurements of velocities of reactions.

The well-known formula $v = A^m B^n B^p$ is often right. When it does not hold it is a proof that the reaction does not take place as was assumed. It has been found, for example, that in the formation of a dye from a diazo compound and an amine, the amine salt does not react itself, but that by hydrolitic dissociation a base is formed, which then reacts with the diazo compound. The author has measured the velocity of the reduction of nitro-compounds to hydroxyl amine compounds by means of stannous oxide. The paper discusses at great length the reactions in alkaline and acid solution.

ACCUMULATORS WITHOUT LEAD.

Prof. Dr. K. Elbs, of Giessen, discussed non-lead accumulators. The reactions in the lead accumulator and in the Edison battery are respectively

$$Pb + PbO_2 + 2H_2SO_4 \rightleftarrows 2PbSO_4 + 2H_2O$$
$$Fe + 2Ni(OH)_3 \rightleftarrows Fe(OH)_2 + 2Ni(OH)_2$$

Ordinary Fe when polarized as anode in KOH becomes passive. The active mass of iron must, therefore, be especially prepared. The best way is to glow iron filings in the open air, then reduce them with H_2, and finally polarize them at the cathode. Fe, which has been prepared in this manner, can be depolarized at the anode fairly well; though it is impossible to make the whole mass of the electrode active. There is always an Fe core which is not acted upon and which does not disappear even after repeated polarization and depolarization.

It is probable that a hydroxide of divalent Fe is formed at the anode. The hydroxides of trivalent Fe could be easily identified by the yellow-brown color, and are not formed until later by oxidation in the air. Furthermore, unless they are subjected to considerable compression in KOH, $Fe(OH)_3$ and $FeO(OH)$ can scarcely be reduced at all, and Fe_3O_4 can be reduced only with difficulty. The reaction at the Fe electrode follows the equation

$$Fe \rightleftarrows Fe(OH)_2$$

or

$$Fe \rightleftarrows Fe^{\cdot\cdot} + 2(-)$$ where (—) means a negative charge.

The product of the oxidation of $Ni(OH)_2$ in alkaline solutions is always $Ni(OH)_3$. No higher oxide is ever obtained. The product of the reduction of $Ni(OH)_3$ in operation of the accumulator is always $Ni(OH)_2$. The equation for the reaction is

$$+\ (+)$$

$$Ni(OH)_3 \rightleftarrows Ni(OH)_2 \text{ or } N\overset{\cdots}{i} \rightleftarrows N\overset{\cdots}{i}$$

The complete equation is

$$Fe + 2Ni(OH)_3 \rightleftarrows Fe(OH)_2 + 2Ni(OH)_2$$

The process of charging and discharging the positive plates is more complete than that of the negative plates, though even on the positive plates, with moderate current density O_2 is evolved. Both of the nickel hydroxides are only very slightly soluble in KOH. The hydroxides of cobalt are quite soluble in KOH, and cobalt is, therefore, unsuitable for use in the accumulator.

The e. m. f. of the combination Fe, KOH, $Ni(OH)_3$ is 1.42 to 1.48 volts when the accumulator is freshly charged. While standing, the e. m. f. falls slowly on discharging rapidly to 1.35 or 1.37. It is independent of the concentration of KOH.

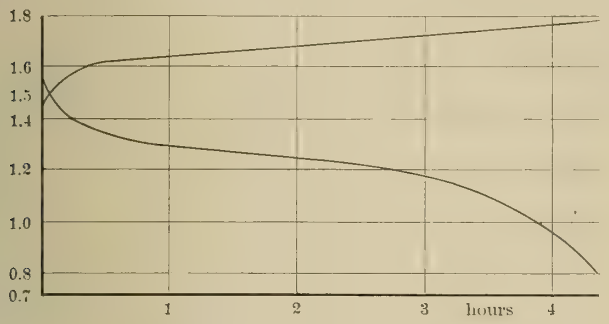

FIG. 1.—CHARGE AND DISCHARGE CURVES OF Ni-Fe CELL.

Dr. Gräfenberg, of Kalk (near Cologne), spoke on the Junger battery, as made by a German storage battery company. (This is essentially the same as the Edison battery.) The positive electrode is made of black nickel hydrate. A hydrate of the lower oxide is obtained electrolytically and is converted to the higher oxide by chemical oxidation. The negative electrode is composed of iron, which is obtained by the reduction of iron scales by H_2 at 380°. To the nickel oxide 40 per cent nickeled graphite is added and to the iron 10 per cent.

These masses are then compressed into small bricks, which are introduced into boxes of thin sheet nickel and further compressed. Ten such boxes are formed into one plate.

The speaker exhibited an accumulator consisting of six pairs of plates in a pressed steel case, the total weight of which was 3 kilograms. The distribution of weight is active mass, including admixtures 900 grams, frames of plates 750 grams, electrolyte 625 grams, case, etc., 725 grams.

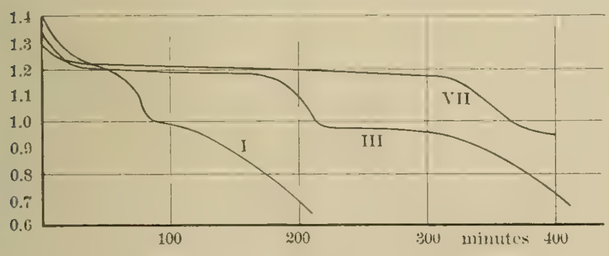

FIG. 2.—DISCHARGE CURVES OF Fe ELECTRODE.

The electrolyte is 20 per cent potassium hydroxide, which is free from acid radicals, organic substances and carbonic acid.

With a normal discharge of 4 hours the cell has a capacity of 35 to 40 ampere-hours, with an average e. m. f. of 1.23 volts. That is 16 to 18 watt-hours per kilogram of aggregate weight. The volume is 2.9 liters per 100 watt-hours.

The resistance increases during the discharge. The average value for the small cell was 0.0035 ohm.

The e. m. f. of the cell at rest is 1.35 volts. On charging it rises rapidly to 1.6 and then gradually to 1.68. Fig. 1 shows the curves for the whole battery. The potential curve of the Fe electrode (Fig. 2) shows two distinct stages of discharge, one at 1.2 volts and the other at 0.9 volts. The formation of the active material of the Fe electrode is complete after seven charges.

Fig. 3 shows that the Ni electrode has only one stage of discharge. The Ni electrode reaches its full capacity in five discharges. Theoretically, the iron electrode has the greater capacity, but the efficiency of the reaction at the Ni electrode is 75 per cent, while that of the Fe electrode is only 10 per cent.

The capacity decreases only slightly with increasing current density. The accumulator will be useful principally where heavy currents are required.

The iron electrode is affected by the air, so it is difficult to transport it. When charged it oxidizes, when discharged it is further oxidized to ferric oxide, and is afterwards difficult to reduce. The accumulator is very insensitive to rough handling, and is not easily injured while in transportation. After 100 discharges it had lost 12 per cent of its efficiency, after 200 discharges 17 per cent. This loss has not yet been explained.

In the discussion, Dr. Roloff said that Elb's equation is correct, but there are other reactions which take place. This is proved by the step-shaped fall of the e. m. f. of both electrodes. There are also several stages in the charge. By measuring the potentials against an Hg electrode, Roloff identified the electrode potentials as potentials of the various oxides of Ni and Fe. He found a black $Ni(OH)_2$, which was previously unknown. Its formula is $Ni(OH)_2 + 2H_2O$. The hydrate $Ni(OH)_2 + H_2O$ is gray. Amhydrous $Ni(OH)_2$ is green.

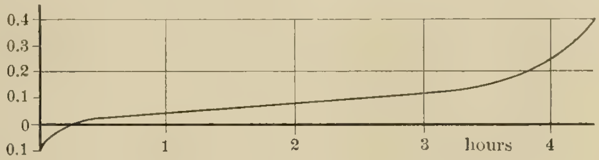

FIG. 3.—POTENTIAL CURVE OF Ni ELECTRODE.

THE ACTION OF THE PHARMACEUTICAL REAGENTS ON LIVING ORGANISMS.

A paper on this subject was presented by Prof. Dreser, of Elberfeld, who studied the effect of increasing quantities of salicylate of soda on the evolution of CO_2 from barm. An interesting result is the experimental and mathematical demonstration of the homeopathic doctrine that small doses of poison are invigorating, while large ones are fatal.

KINETICS OF EXTREME STATES OF MATTER.

Dr. Emil Bose, of Göttingen, presented a paper, an interesting result of which is that through kinetic considerations on rarified gases conjointly with the electron theory, an advance into the uninvestigated field of the kinetic theory of solid substances is made possible. From experimental results obtained for quartz, fluorspar, sylvite and rocksalt, the author concludes that the amplitudes of the molecules in the solid state are considerably smaller than the mean distance between molecules.

They also indicate a parallelism between the cofficient of expansion and amplitude of oscillation.

SUGAR SYNTHESIS; SILENT DISCHARGES.

Dr. W. Löb, of Bonn, presented a paper on the assimilation of carbonic acid. It is unfortunate that this paper, which is perhaps the most interesting of the whole congress, cannot be reviewed as completely as the importance of the subject demands. The author has imitated in laboratory experiments the formation of sugar from carbonic acid and water, as it occurs in plants.

He followed, in the opposite direction, the same course which nature takes in the breaking down of sugar molecules by fermentation. He first reviewed exhaustively the literature on the subject and the various hypotheses which have been set up.

The following is the most probable scheme:

$$C_6H_{12}O_6 = 2CH_3CHOHCOOH = 2CH_3CH_2OH + 2CO_2$$

$$2CH_3CH_2OH + 2O_2 = 2CH_3COOH + 2H_2O = 2CH_4 + 2CO_2 + 2H_2O$$

$$2CH_4 + 2O_2 = 2CH_2O + 2H_2O$$

$$2CH_2O + 2O_2 = 2H_2O + 2CO_2$$

The final result is:

$$C_6H_{12}O_6 + 6O_2 = 6H_2O + 6CO_2$$

The speaker enumerated the facts which lead to this scheme and described the method with which he obtained his sugar synthesis.

Carbonic acid and water, under the influence of the silent electric discharge, form carbon monoxide, CO, and a small amount of formic acid, HCOOH. If oxygen is excluded from the CO, formaldehyde is formed:

$$CO + H_2 = CH_2O$$

The H_2 is the product of the reaction:

$$CO + H_2O = CO_2 + H_2$$

which always takes place in a mixture of CO and H_2O vapor at high temperatures.

At the same time formic acid is formed. Its production is considerably increased by adding H_2 to the gas mixture. It is probable that in plants the formation of formic acid is the principal reaction:

$$CO + H_2O = HCOOH$$

since formaldehyde is never found in plants. If a depolarizer which will remove oxygen, *e. g.*, benzaldehyde, salicylic acid, even chlorophyl is added, the production of CO and HCOOH becomes much greater. Part of the formaldehyde unites with water vapor to reform $CO + H_2$. At the same time methan is formed:

$$CH_2O + 2H_2 = CH_4 + H_2O$$

Formic acid and water vapor act similarly. Methan and water form acetic acid. Since $CH_4 + CO$ gives acetaldehyde, the formation of acetic acid is similar to that of formic acid.

It is known that ethyl alcohol is formed from acetic acid by reduction; perhaps alcohol is formed directly from aldehyde and methan. On the other hand, it is known that alcohol is the regular product of the fermentation of sugar, and it is frequently found in plants. The experiments on the formation of alcohol are not yet completed. Alcohol and water alone do not form hydrocarbons. Hydrocarbons are obtained, however, when the silent discharge is passed through $CH_3COOH + H_2O + CO_2$. The speaker obtained a liquid which gave all the reactions of a sugar, in particular of a hexose. After evaporating, there remained an acid (due to lactic acid?), syrupy liquid, whose reactions were described in detail by the speaker. With phenylhydrazine he obtained an osozon whose melting point was 148° to 155°—perhaps *β*-acrose.

It is probable that the way in which the sugar synthesis takes place varies in detail according to the nature of the different plants. So much is certain that the addition of energy alone is sufficient to carry out the synthesis of sugar from $CO_2 + H_2O$.

The speaker closed with some interesting remarks concerning assimilation in plants. Chlorophyl alone is not capable of producing the sugar synthesis—in addition a supply of energy (mostly in the form of light) is required. Chlorophyl probably serves only to remove the oxygen—this is necessary to produce a good yield.

Fixation of Atmospheric Nitrogen by Electric Discharge.

Dr. Johannes Brode spoke on the oxidation of nitrogen in the high-tension electric arc. When high-tension electricity is discharged in the air with a current of about 0.1 amp., an arc, having three zones, is formed: I., a brightly luminous, bluish band arched upwards; II., above that a bluish-green part; III., a dim brownish layer, bending upwards. The appearance of the arc is independent of the material composing the electrode. It is, therefore, not due to small particles broken off from the electrodes.

The arc varies only slightly when the current is varied. When the distance between the electrodes is increased, part III. becomes larger.

In air and in nitric oxide the same reaction takes place, but in opposite directions, *i. e.*, equilibrium sets in. The eqiulibrium depends on the number of watts used, on the kind of current, the distance apart of the electrodes, and, when the electrodes are close, on the material of the electrodes and the cooling of the electrodes. In the hot part of the arc a higher concentration of nitric oxide is formed; in the cooler part it is partly decomposed.

The speaker showed that the flow of current is principally in Part I. When a glass rod is introduced into Parts II. and III., current and voltage remain unchanged. But when it is held in Part I. the voltage changes. Part I. has, therefore, the highest temperature.

From the Van't Hoff equation, Nernst calculated the curve NO — concentration — temperature. The author found that in zone I. a gas mixture containing 8 vol. per cent NO is formed, from which a temperature of 3,700° is calculated. In Part III. the concentration of NO in equilibrium with $N_2 + O_2$ is less. Therefore, when the NO formed in Part I. enters Part III. it is partly decomposed. Zone II. could not be investigated.

The speaker described other experiments in which he found that ozone is formed in oxygen by the heat of the electric arc. He found no indication that O_3 is formed by the light rays of the arc.

When the arc is broken up by a magnet or other means, the cooling is more rapid, and the loss of NO is small. This fact is known to industrial electrochemists.

Zinc and Manganese as Anodes.

Dr. W. J. Müller, of Mühlhausen, spoke of the anodic behavior of zinc and manganese. The author used anodes with different current densities and measured their potentials against a normal electrode. The capillary syphon which lead to the normal electrode was introduced in immediate proximity to the electrode.

In NaOH the polarization of zinc is normal, *i. e.*, the same potential curve is obtained with rising as with falling current density. The same is true of zinc in neutral and acid solutions of Na_2SO_4.

In alkaline Na_2SO_4 solutions the result is quite different. In a solution of n — Na_2SO_4 and 0.5n NaOH the polarization rose normally at first. At 4 to 6 milliamperes there is an enormous falling off of the potential; on increasing the current density beyond this point the polarization rises further in a straight line, and is accompanied by the evolution of gas. With decreasing current density the polarization falls in a straight line to 0.9 milliamperes, at 0.7 milliamperes the polarization reverts momentarily to its initial value.

The following general results were obtained: First, the polarization of Mn and Zn is normal in all acid solutions of sodium sulphate and sodium phosphate and in neutral sulphate solutions; the polarization of Zn is normal in neutral phosphate solutions. Both metals polarize normally in solutions in which concentration of NaOH is less than 0.01 normal.

Secondly, with abnormal polarization there are two distinct cases: either the e. m. f. sets in at once as with Zn in alkaline sulphate solutions, or the abnormal polarization requires a certain length of time to develop, as with Zn in alkaline solutions of phosphate, and with Mn in all alkaline solutions.

Thirdly, with normal polarization the metal always goes into solution with its smallest valency. In cases where the polarization is abnormal one would infer that the metal changes into a passive modification. The author could detect no change in the reflecting power of a zinc mirror which had been subjected to anodic electrolytic action.

If a passive layer were formed its passivity would continue for some time, but such is not the case. The author used his experiments to support a theory of passivity which he had advanced previously.

Contact Process of Making Sulphuric Acid.

Prof. Dr. Max Bodenstein dealt with the determination of equilibrium in contact sulphuric acid. This subject, which was first investigated in the classical paper of Knietsch, is important to both scientific and technical chemistry.

Though it has been the subject of a number of thorough experimental investigations, the results of the various experimenters are in part quite different. This is due to the unusually great experimental difficulties to be encountered.

From his experiments at 500°, the working temperature of the Baden Anilin & Soda Factory, at which the equilibrium is close to 100 per cent SO_3, Knietsch concluded that the law of chemical mass action is of no importance in technical chemistry. This is certainly not true for higher temperatures, such as are required when ferric oxide is used as the contact substance.

The author determined, therefore, the constant of equilibrium, K of the reaction: $2SO_2 + O_2 = 2SO_3$

$$K = \frac{[SO_2]^2 \times [O_2]}{[SO_3]^2}$$

in which the symbols in brackets denote the pressures of the gases in question.

The measurements were made at different temperatures and were very exact. His ingeniously designed apparatus avoids the sources of error inherent in previous experiments, especially the uncertainty of the temperature.

The gas (SO_2 distilling at room temperature and electrolytic O_2) having been previously analyzed, was passed through a quartz vessel, which was filled with platinum sponge.

The quartz vessel, which was held in a modified Heraeus platinum foil furnace, was composed of two compartments. From these the gas passed very rapidly through a fine capillary tube, thus avoiding any appreciable displacement of the equilibrium by the fall of temperature.

The form of the quartz vessel made it possible to obtain the equilibrium from opposite direction also ($2SO_3 \rightarrow 2SO_2 + O_2$).

Measurements made at 727°, under the most varied conditions of experiment, proved that the constant of equilibrium obtained is independent of the direction of the reaction (*i. e.*, whether the reaction proceeds from SO_3 or from $SO_2 + O_2$), independent of the velocity of the flow of gas, of the relation of SO_2 to O_2 in the original gas mixture, and independent of the presence of nitrogen.

The experiments furnish a verification of the law of chemical mass action which for range and accuracy has been equalled probably by no other reaction. The results of the experiments are:

Temperature C°	528	627	727	832	897
K	1.55×10^{-5}	3.16×10^{-4}	3.54×10^{-3}	2.80×10^{-2}	8.16×10^{-2}

From these figures the following values for q, the heat of reaction of the Van't Hoff formula, were calculated:

Temperature	500° — 600°	600° — 700°	700° — 800°
q	21,900	21,700	21,500

The value of q at room temperature is found by extrapolation to be 23,100, which is in excellent agreement with the value obtained by Berthelot, 22,600. For the specific heat of SO_3 at 100° C. 10.6 was obtained. This result accords with the values for other similar gases.

The formation of sulphuric acid fulfills the requirements of the law of chemical mass action so completely that it forms one of the most instructive examples of the application of the law. The constant K can be calculated from the equation (T = absolute temperature):

$$\log K = -\frac{10373}{T} - 2.222 \log T + 14.585$$

in close approximation to the observed values. The author exhibited tables and curves from which the yield at different temperatures and with gases of different compositions could be read off directly.

The conclusion of the report will be published in our next issue.

Faraday Society Meeting.

The sixteenth ordinary meeting of the Faraday Society was held on Oct. 31, the president, Lord Kelvin, being in the chair.

Alternating-Current Electrolysis.

Prof Ernest Wilson gave a resumé of his paper on alternating-current electrolysis. Experiments made with platinum plates in dilute sulphuric acid show that of the total energy supplied to the cell in a given time, more is returned to the source when the frequency is high than when it is low—the maximum coulombs being of the order .0006 per sq. cm. in each case. If the quantity of electricity be plotted co-ordinately with the e. m. f. of electrolysis it is found that at the higher frequency, for about the same maximum coulombs, the curve has relatively a smaller area, such reduction being probably brought about by the greater reversibility. An experiment made at an intermediate frequency when the maximum coulombs were .0000023 per sq. cm. gave a still higher value for the proportion of the total energy which is returned to the source—demonstrating that the magnitude of the maximum coulombs has an important effect.

Further experiments made with aluminium plates in dilute potash, soda and ammonia-alum solution showed a similar effect.

Judging from the above experiments, one might be led to expect that when a metal is dissolved in an electrolyte by alternate-current electrolysis, the amount dissolved in a given time at a given current density would be smaller at high than at low frequency. This is found to be the case.

Besides this chief conclusion there remain indications of other important effects. So great is the number of variables, and so great the difficulty of keeping all but one constant during an experiment, that from the few experiments performed it is dangerous to attempt a generalization.

A series of experiments was made to determine the influence of current density, the frequency being constant = 92. The weight of lead dissolved in dilute H_2SO_4 increases with the current density. Amalgamated zinc is difficult to deal with as the degree of amalgamation influences the result. Iron is dissolved in ferrous-sulphate at a greater rate the greater the current density when the respective experiments are started with fresh solutions. This is not the case when a new experiment is started in a solution whose density has been increased by electrolytic action in a previous experiment. A difficulty with ferrous-sulphate solution is its known tendency to oxidize. In sodium chloride more iron is dissolved at the higher current density. A complete investigation would need to take accounts of the density and temperature of the electrolyte, and possibly of other conditions.

Mr. W. R. Cooper then presented an abstract of his paper on alternating-current electrolysis, as shown by oscillograph records. The author pointed out that although polarization is of the nature of a capacity in an alternate-current circuit, there is a considerable difference. What might be termed the e. m. f. of a condenser rises and falls as rapidly as the applied pressure, but although the e. m. f. of polarization may rise as rapidly as the applied pressure, it falls more slowly, with the result that under suitable conditions the current curve may depart very considerably from the sine form. Actual oscillograph records were reproduced in support of this view. In considering the subject it has been very generally assumed that the current follows a sine curve. Since the curve obtained depends very much on the conditions of the experiment, it is necessary to define the conditions very carefully before conclusions can be drawn from different experiments. Oscillograph records of electrolytic rectification were also shown.

In the discussion which followed, Mr. A. E. Trotter referred to the bearing of work on alternate-current electrolysis on the corrosion of water and gas pipes, caused by stray currents from alternate-current systems. He described some practical experiments he had made with lead pipes buried in the earth; these experiments showed that there was a minimum current density or a minimum voltage—he was not sure which—below which no corrosion took place. More work was required to be done in order to indicate the exact conditions necessary for corrosion, but great caution was necessary in the application of scientific experiments to practice.

Mr. James Swinburne thought that the greater absorption of energy found by Prof. Wilson at the lower frequencies was simply due to diffusion, and this was borne out by Mr. Cooper's curves.

Dr. F. M. Perkin said he had found that platinum plates electrolyzed alternately in dilute sulphuric acid were more corroded at low than at high frequencies; a colloidal solution of platinum seemed to form, and the electrodes had an etched appearance.

Crystalline Structure of Electrodeposited Copper.

Prof. A. K. Huntington presented a paper on this subject, which was illustrated by microphotographs.

In Mr. Cowper-Cole's process for making copper wire electrolytically (see our October issue, p. 392) a spiral scratch or groove on the mandril causes the copper deposited on it to part so easily that a long ribbon can be obtained. The author's explanation is that the direction of the lines of crystallization of an electrodeposited metal are the same as in a casting made on surfaces having the same inclination, *i. e.,* the crystals form at right angles to the surface on which the deposit or the casting is made. It follows that when the crystals which form on surfaces more or less at an angle to one another meet, there will be want of continuity in the two sets of crystals, and a line of weakness will be developed. Therefore, in castings and deposits when strength and not rupture is aimed at, it is important to avoid sharp angles.

In the case of a cathode plate which has been deposited on a thin strip of electrodeposited copper the crystals of the original strip continued in some cases in the cathode plate. On annealing the electrodeposited copper the long crystals of the deposit are completely broken up and become largely twinned. Hence brittleness, due to the direction in which crystals form in castings and electrodeposits, may be modified and probably completely removed by suitable annealing, unless the continuity is entirely broken.

In the discussion which followed, Mr. J. F. R. Rhodin thought that structure of the copper deposited in a groove, such as Mr. Cowper-Coles employed, depended on electrostatic repulsion rather than on crystalline form. The fringing at the edge of a deposit, even where the current density was not high, seemed to him to confirm that explanation.

Electrolytic Production of Hypochlorite.

A paper by Mr. W. Pollard Digby dealt with some observations respecting the relation of stability to electrochemical efficiency in hypochlorite production.

The author commenced by drawing attention to the fact that in all electrolytic methods of producing hypochlorite solutions, only a small portion, rarely more than 18 per cent, of the chlorine usually present in the form of chloride is converted into hypochlorite, and suggests that the amount of available chlorine produced from a sodium chloride solution depends upon the relation which the amount of unconverted sodium chloride actually present between the electrodes bears to the current density.

As regards stability, particulars are given of two tests in which the rate of depreciation, although only 0.3 to 0.4 grammes in 12 hours per liter for solutions containing 8 grammes of available chlorine per liter, caused a very great depreciation in the yield per kw-hour supplied. The suggestion is advanced that no test of the efficiency of any apparatus for the electrolytic production of the hypochlorites can be regarded as complete without an estimation of the losses through instability, and a mention of the volume of the contents of the electrolyzing tank.

While stability of solution does not greatly affect the efficiency of any continuous type of apparatus (in which the sodium chloride is fed in at one end and drawn off at the other end ready for continuous use), it will materially affect the efficiency in any non-continuous apparatus, or in any apparatus in which the electrolyte is circulated through an external cooling tank and returned to the electrolyzer to be further worked up. It is suggested that the presence of minute quantities of iron in the solution have also a bearing on the depreciation.

ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS

Electric Furnaces.

Reducing Oxides.—A. J. Rossi, 802,941, Oct. 24. Application filed Nov. 3, 1902.

To reduce highly refractory oxides, as that of titanium, the inventor uses aluminium as reducing agent with heat supplied from an outside source, preferably—if very high temperatures are required—in form of electrical energy. In the latter case he obtains the heat of the electric furnace combined with the heat evolved by the oxidation of the aluminium. First a fused bath of aluminium is produced, into which the metallic oxide is then introduced, not necessarily in powdered form, but merely in their natural condition, or such coarse sub-division as will tend to accelerate the reduction to the required state. Any impurities in the charge, whether derived from the materials of the furnace walls or from the gangues of the oxygenated metal, will, according to the inventor, on the sufficient continuance of the high temperature required, enter into the alumina slag. (See also our Vol. I., p. 523.)

Calcium Carbide Furnace.—E. Appleby, 803,147, Oct. 31. Application filed Jan. 28, 1905.

The inventor has devised an arc furnace for the production of calcium carbide. The main object is to provide a limited zone of intense heat into which powdered lime and coke are continuously fed, and from which the formed carbide is continuously withdrawn. The construction is clearly indicated in claim 1: "In the electric furnace, the combination of a vertically-disposed tubular receptacle open at the bottom; a second receptacle surrounding the first, and having a bottom movable in alinement with said first receptacle; a pair of electrodes mounted in said frame and extending into the furnace near the lower end of said first receptacle; mechanism for gradually moving said bottom downwardly away from said electrodes, and mechanism for agitating the walls of said first receptacle."

Electrolytic Processes.

Electrolysis with Insoluble Anodes.—A. G. Betts, 803,543, Nov. 7. Application filed Feb. 16, 1905.

The yield from anodic oxidations is increased, the life of the anodes is prolonged, and the necessary voltage is reduced by very greatly increasing the speed with which the layer of solution in contact with the anode surface is changed. The inventor uses anode rods and gives them a reciprocating motion in a direction perpendicular to their length. For instance, in electrolyzing a solution of ferrous and cupric sulphates, with a diaphragm, depositing copper on a cathode and oxidizing ferrous to ferric sulphate at the anode, and with a circulation of electrolyte that was previously considered amply sufficient, the e. m. f. required to work the cell was considerably reduced, the evolution of gas at the anode was entirely stopped, and the current efficiency raised from 50 to 100 per cent on giving the anodes a reciprocating motion of 100 complete cycles per minute, with an amplitude of about 1 inch. Other examples of the application of this method are the oxidation of organic substances and the manufacture of chromic acid from chromium sulphate.

Treatment of Anode Slimes.—E. F. Kern, 803,601, Nov. 7. Application filed March 27, 1905. (Assigned to A. G. Betts.)

The inventor treats lower-grade slime from the treatment by electrolysis of lead alloys containing a large quantity of antimony or arsenic, as follows: He oxidizes the slime by a gentle roast, driving off most of the arsenic as arsenious acid and converts the antimony into trioxide, so that it can be subsequently dissolved from the product with a suitable solvent, preferably hydrofluoric acid. Moistening the slime with dilute or concentrated sulphuric acid facilitates the roasting and prevents sintering of the slime. A temperature below a red heat is preferred, since if the temperature reaches a red heat the antimony goes into tetra and pent oxide, which is not easily soluble. The process is also applicable to the treatment of copper anode—slime containing much copper sulphide converted by roasting into copper oxide and sulphate. Before extracting the antimony from the product the inventor treats it with an acid solution of ferric sulphate or other oxidizing acid solution to complete the oxidation of antimony, arsenic, etc., and to remove any copper. After extracting the antimony the slime is ready for fire treatment. Silver may be extracted with the copper, if desired, or left to be recovered by fire processes, in whole or in part.

Tank for Refining Lead.—A. G. Betts, 803,544, Nov. 7. Application filed March 18, 1904.

For his lead refining process the inventor uses a tank with a wooden body, lined with copper plates upon its bottom and sides. These plates are joined together along their contiguous edges by solder. To the central portions of the copper plates thus soldered together is applied a coating of insulating paint which extends to within a short distance of the line of solder which lies along the joint of the lining, leaving a narrow strip of bare copper exposed along the opposite sides of the lines of solder. The tank is then filled with acid copper sulphate solution, and is connected in an electric circuit as a cathode. The soldered surfaces thereby become covered with a coating of pure copper, so that there is a substantially integral copper lining over the soldered surface. Finally, the whole interior of the tank is covered with an insulating paint. But it appears that the insulating-paint cover is not strictly necessary, since "the copper lining of the tank, although capable of being dissolved by the electrolyte, is protected from attack by the same by reason of its potential relatively to that of the anode." Mr. Betts has demonstrated by experiment with such an apparatus, using an anode from which lead was dissolved by the electrolyte, that the copper lining of tank assumes a potential intermediate between the anodes and cathodes, and is not in the slightest degree attacked or dissolved by the electrolyte, and by the use of this apparatus he has electrodeposited lead which contains less than one part of copper to one million parts of lead.

Electrolysis of Sodium Sulphate.—H. S. Anderson, 803,263, Oct. 31. Application filed June 21, 1905.

The general problem of the electrolytic production of caustic soda and sulphuric acid from sodium sulphate was discussed on page 286 of our August issue. H. S. Anderson uses for this purpose a combination of a mercury cathode cell and a diaphragm cell. If simply a diaphragm cell would be used, the recombination of the anodically-formed sulphuric acid and of the cathodically-formed caustic soda could not be effectively prevented. For this reason a mercury cathode is used, the sodium ions when discharged forming an amalgam which is decomposed in a different compartment. It is also necessary, however, to prevent the anodically formed sulphuric acid from passing over to the mercury cathode. The inventor, therefore, provides a diaphragm above the mercury cathode, and below the anodic compartment and further passes a continuous stream of sodium sulphate solution between this diaphragm and the mercury cathode, so that any acid which might pass through the diaphragm is carried away with this stream of solution and cannot reach the cathode. A cross-section of the cell is shown in Fig. 1. **A** is a reservoir containing sodium sulphate, **B** is the electrolytic cell proper, and **C** the cell in which the amalgam is decomposed for the production of caustic soda. In **B** there is the anode 17, the diaphragm 16 and the mercury cathode 12. A continuous stream of sulphate solution is passed between 16 and 12; it flows from the supply tank **A** to the inflow-passage **a**, then passes between 16 and 12, and is taken off through the outlet-passage **b** by a syphon 22.

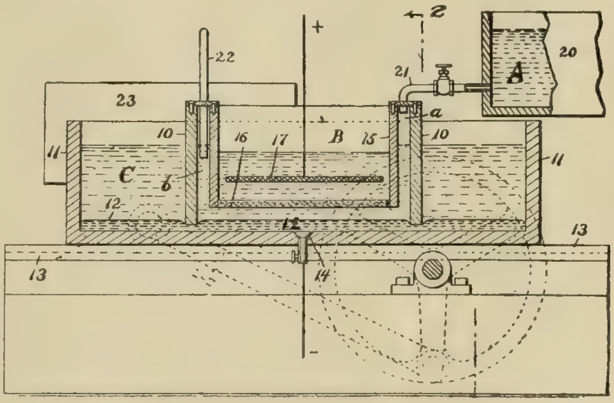

FIG. 1.—CELL FOR ELECTROLYSIS OF SODIUM SULPHATE.

Collection of Gases from Electrolysis.—G. E. Cassel, 804,931, Nov. 21. Application filed May 4, 1905.

To prevent the gases developed either at the anode or at the cathode from passing over to the other electrode, he constructs each electrode as follows: Each electrode is made of two conducting end plates, facing each other. The outside surfaces of these end plates are insulated so that no gas can develop on them. The two inside conducting surfaces are provided with a number of plates inclined upwards, above one another; the ends of these inclined plates face each other, but there is a free space between them. The top surfaces of these inclined plates are covered by plates of glass or otherwise insulated. All the gas formed develops on the under sides of the inclined plates, and in striving to ascend the gas is compelled to pass to the free space in the center, where it rises upwards into a closed hood provided with a pipe.

Apparatus for Electrolysis of Chlorides of Alkalies.—F. J. Briggs. 802,060. Oct. 24. Application filed Jan. 19, 1905.

Mechanical details of construction, the object of which is the improvement of details of his electrolytic cell, patented May 12, 1903 (patent 727,889, described in our Vol. I., p. 392). He reduces the weight of his cell by doing away with his former cathodes by balancing the "dialyzers" by means of a system of levers and comparatively small weights, the tank being made of iron and the perforated iron sheets which make parts of the dialyzers operating together with the iron tank as cathode. There are also some other modifications of his former cell construction.

Manufacture of Seamless Hollow Articles.—F. A. Voelke, 789,342. May 9. Application filed Jan. 20, 1904.

Mechanical details of a method for making seamless hollow articles, such as floats, cylinders for holding compressed air, etc. In a proper mould a core or pattern of the desired form is first made from paraffin or wax, etc.; for this purpose the mould is heated and revolved, after molten paraffin, etc., has been poured into it; compressed air is simultaneously introduced to force the paraffin against the face of the mould. After the paraffin core has thus been formed it receives a graphite coating and then an electrolytic deposit. It is now suspended with the passage, through which air was introduced before, downwards, and is then heated above the melting point to paraffin. The paraffin core is thus melted within the metallic shell, and is in part reduced to vapor, which forces the paraffin out through the former air hole. The shell is finally thickened by alternating dipping into a fused bath and electrolytically depositing other layers.

Electric Discharges Through Gases.

Nitrogen Oxides by Electric Discharges.—E. Marquardt and H. Viertel, 804,021. Nov. 7, 1905. Application filed May 2, 1903.

A mixture of oxygen and nitrogen gases performs a cycle, continuously passing through an electric arc discharge, and thence to a cooling or absorbing apparatus, then returning to the arc, etc. "Thus a definite quantity of gas is gradually enriched with oxides of nitrogen until a concentration is reached, at which every excess of product is retained in the cooling or absorbing apparatus." The principle is shown in Fig. 2, where **b** is a system of pipes forming a closed cycle; **c** represents the electrodes when the arc discharge takes place; **d** is a vessel containing a cooling substance, like solid carbonic acid; **e** a condenser in which the formed product collects, and from which it may be drawn off by means of the spigot **f**. **A** is a vessel containing the raw material. A sufficient automatic circulation can be maintained by the difference of temperature between the arc and the condenser, if the former is so arranged so as to allow the warmed gas mixture to rise up, and if the cooling part of the circuit be directed downward. For this purpose the arc is placed at a low position in the rising branch of the circuit. Continuous working is possible, which requires only very little supervision. If several arcs are employed they are arranged side by side, in order to prevent the gaseous mixture from passing through several arcs in series, whereby a partial reduction of the formed oxides would be caused. For producing as cheaply as possible nitrates or compounds containing oxides of nitrogen, they use 30 to 100 volts and cheap mixtures of nitrogen and oxygen, such as atmospheric air, under a pressure of 1 to 2 atmospheres. For producing pure nitrogen tetroxide the best results are obtained by using a constant mixture of 50 per cent oxygen and 50 per cent nitrogen, and replacing the oxygen and nitrogen which have combined by adding continuously a mixture containing two volumes of oxygen to one of nitrogen; pressure 5 atmospheres, voltage 5,000 to 10,000. For the arc discharges the inventors use electrodes which contain compounds of metals, especially fluorides, chlorides, borates, silicates, or oxides of the alkalies or earthy alkalies or of magnesium or mixtures of these. A specially effective electrode consists mainly of carbon, and contains from 10 to 30 per cent of fluorspar. One effect of these additions is a reduction of the resistance of the arc and the production of a very large and voluminous arc.

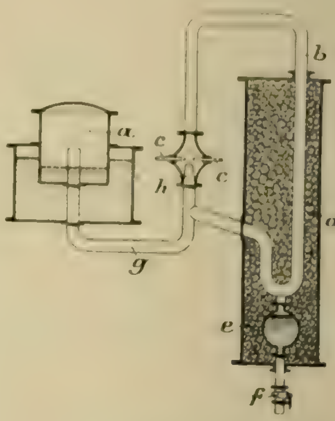

FIG. 2.—PRODUCTION OF NITROGEN OXIDES.

Production of Cyanide of Ammonia, Hydrocyanic Acid, etc.—A. M. Gow (assigned to George Westinghouse), 801,728, Oct. 10. Application filed Dec. 9, 1902.

Natural gas, the composition of which is about 95 per cent methane, CH_4, is mixed with nitrogen or air and the mixture passed through an electric arc discharge to produce hydrocyanic acid. The temperature must be high enough to result in the fixation of the nitrogen, but not high enough to decompose again a great part of the compounds formed. The arc should be "sputtering violently when the reaction goes on. When the arc remained quiet and steady between the electrodes, the desired production of nitrogen compound was not obtained, but carbonic acid was formed." The inventor has found "that a mixture of half natural gas and half air, upon being passed once through the heating chamber, would give a recovery, based on the carbon contents of the cyanides formed, equivalent to 5 per cent of the carbon originally contained in the mixture. Again, from 1 liter of natural gas and 1 liter of air I have recovered 0.065 grams C_2N_2, which would be equivalent to 0.164 grams KCN. In another experiment from the same mixture I have recovered 0.0578 grams C_2N_2, equal to 0.145 grams KCN." The following reactions are stated to occur simultaneously:

$$2CH_4 + \text{heat} = 2C + 4H_2$$
$$2C + 4H_2 + O_2 = 2C + 2H_4 + 2H_2O$$
$$2C + 2H_2 + \text{electric action} = C_2H_2 + H_2$$
$$C_2H_2 + 2N + \text{electric action} = 2HCN$$

Furthermore, the nitrogen enters into combination with hydrogen in other forms than as hydrocyanic acid, HCN; the inventor has obtained ammonia, "which in all probability, in the presence of cyanogen, would exist as ammonium cyanide."

SYNOPSIS OF PERIODICAL LITERATURE.

A Summary of Articles Appearing in American and Foreign Periodicals.

Industrial Electrochemistry.

Fixation of Atmospheric Nitrogen by Electric Discharges.—A long series of experiments on the production of oxides of nitrogen by means of electric discharges through air has been made by Otto Scheuer. The results are printed in form of a thesis in French language (for the degree of Doctor of Physical Science, University of Geneva, 1905), and a German translation in abstract may be found in the "Zeit. f. Elektrochemie," Sept. 1. The experiments were made partly with a Ruhmkorff induction coil and partly with a high-tension transformer. The author determined the output as a function of the velocity of the stream of air, the distance between the electrodes, the watt consumption, the material of the electrodes and the form of the apparatus, using both dry and moist air. The results are given in tables and diagrams. Some of his devices are quite interesting. With the induction coil good results were obtained when the arc was formed between the two horns of a horn lightning arrester, as shown in Fig. 1. He thereby obtained a series of arcs traveling upwards between the two horns. Another form is shown in Fig. 2. The air is introduced through the hollow pipe on the left hand, which serves as one electrode. The other electrode (right hand) is provided with a cone through the central opening of which the nitrous vapors are sucked off. The two electrodes were stationary, although it would have been better to revolve either the funnel electrode centrically or the pipe electrode excentrically, since in this case a true cone of flames would have been obtained, while with the stationary electrodes only several flame bands from the end of the pipe to points of the periphery of the cone were obtained. The flame bands choose the smallest roughness on the periphery, and, by previous filing, the flame bands may be directed to special points. The output with this form of apparatus was relatively high. The results obtained in three experiments of this kind are as follows:

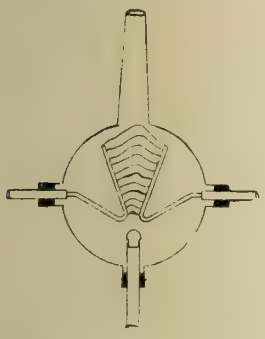
FIG. 1.—ARC DISCHARGE.

Test No.	1.	2.	3.
Primary watts	162.5	162.5	162.5
Duration of experiment in minutes	110.	110.	110.
Quantity of air in cc	10,800.	10,800.	28,800.
cc of air per minute	98.18	163.64	261.81
Watthours consumed	297.9	297.9	297.9
Total NO obtained in cc	1.8753	2.2461	2.8305
Grams of NO producing HNO_3	0.9048	1.1013	1.4003
Grams of NO producing HNO_2	0.9705	1.1448	1.4302
Percent of NO producing HNO_3	48.25	49.03	49.47
Percent of NO producing HNO_2	51.75	50.97	50.53
Percent of volume of air producing NO_2	19.42	13.95	10.97
Percent of volume of oxygen producing NO_2	61.48	44.23	34.85
Grams NO_2 produced by 1 kw hour	9.6480	11.5560	14.5630
Grams HNO_3 produced by 1 kw hour	6.3752	7.7590	9.8658
Grams HNO_2 produced by 1 kw hour	5.1026	6.0190	7.5195

The general results of the author are as follows: In spite of the great number of experiments made with the most varied apparatus the efficiency was poor compared with the theoretical one, on account of the losses in the induction coil and in the transformer employed. The frequency used in the transformer experiments was too low. The experiments show, however, very decidedly how much the efficiency depends upon proper design of the whole apparatus and of the electrodes and upon a proper method of the air supply and removal. Spark discharges are not suitable for the formation of nitrogen oxides; it is necessary to use the flame of the voltaic arc. This flame should be extended over a large active region. In order to increase the effect several arcs should be employed. The duration of the action of an arc upon a quantity of air should be reduced to a minimum, and the nitrogen oxides formed should be removed immediately from the active region of the arc. The form and dimensions of the apparatus in which the discharges take place as well as the construction of the pipes through which the air is supplied and removed are of very great importance. Moist air gives better results than dried air. For certain values of the e. m. f. current and frequency the combination of nitrogen with oxygen becomes a maximum.

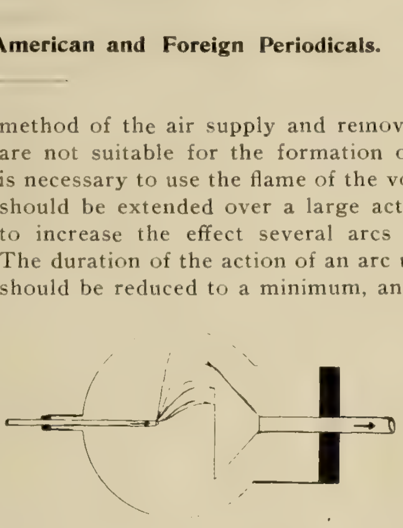
FIG. 2.—ARC DISCHARGE.

Electrolysis of Alkali Chlorides.—A somewhat peculiar method of winning caustic alkalies and chlorine by electrolysis of aqueous solutions of alkali chlorides is described in the "Zeit. f. Electrochemie" of Sept. 15, from German patent 160,967, Kl 12 l; granted June 13, 1905. Carbon plates are placed opposite to plates of solid silver-plated lead, which have numerous perforations filled with porous silver. There is no diaphragm. Alkali chloride solutions of 20° to 22° Be are electrolyzed at about 2.5 volts. The silver anode becomes covered with a layer of silver chloride, while at the carbon cathode NaOH is formed with evolution of hydrogen; 95 per cent of the NaCl are stated to be transformed into NaOH, while the Cl is bound in the silver plates. The solution is then removed, the cell is refilled and the electric current is passed through it in the opposite direction, so that AgCl is reduced and chlorine is set free at the carbon anode. Chlorine and hydrogen are, therefore, set free in different steps of the process, and chlorine does not come into contact with NaOH. "The AgCl cannot pass from the anode into the solution (although it is soluble in NaCl solution), because the Ag would at once be redeposited again by the current." The inventor is J. Heibling.

Caustification of Potassium Sulphate.—In the "Zeit. f. Elektrochemie" of July 7, Ignaz Herold investigates the equilibrium between potassium sulphate and calcium hydroxide, and discusses at the same time the commercial possibilities of producing caustic potash or soda by directly treating a sulphate solution with caustic lime, as indicated by the reaction

$$Ca\ (OH)_2 + K_2SO_4 = CaSO_4 + 2KOH$$

He thinks a commercial application of this reaction is not impossible. Former investigators who worked in this line made the mistake to use a high temperature, while the author finds that at low temperatures (O° C.) the output becomes comparatively high.

Pure Magnesia.—Extended experiments made in the laboratory of the Royal Porcelain Factories, in Berlin, have resulted in the successful manufacture of various commercial articles from pure magnesia. According to a note in the "Zeit. f. Elektrochemie" of Sept. 1, tubes up to 80 cms. length and 7 cms. diameter, with a wall thickness of 7.5 cms., can be made, also crucibles up to 50 cms. height of any diameter and any thickness of walls. When suddenly heated in the air-blast flame the articles do not crack nor do they change their form. Tubes made from this pure magnesia and heated in the electric furnace to 1,750° C., did not show any shrinkage nor—"and this is the most essential point—any trace of electrolysis."

ELECTROPLATING

"Kupron System."—At the recent electrical exposition at Olympia, London, a large assortment of various articles was shown, mostly used in the electrical industries, and ranging from small switch fronts and light fittings to an entire shop front. The idea is to make the articles of clay and then electroplate them. There is nothing essentially new in the "Kupron" system, as it is called. First, the light fitting—or whatever the article may be—is modelled in clay, and the clay is then covered with a thin film of black lead. By immersing the latter as cathode in a copper bath, a layer of copper of any desired thickness is electrolytically deposited on the mould. The finish is bronze, silver, gold, etc. According to the London "Electrician" of Oct. 27, the chief advantage of the process is the low cost of the finished article, "electric light fittings and other things of a similar kind costing not more half what they would have cost if produced in the ordinary way."

THEORETICAL AND EXPERIMENTAL.

Magnesium Anodes.—In "Zeit. f. Elektrochemie," July 28, G. Baborovsky describes some peculiar properties of magnesium anodes. In neutral solutions of chlorides, sulphates, etc., hydrogen is developed at the magnesium anode, and a black powder goes into solution, the so-called "Beetz suboxide," which, however, is probably only powdered metallic magnesium. The potential at which this takes place is approximately the same as shown by metallic magnesium when no current flows. In alkaline solutions at least three conditions of the surface of a magnesium anode must be distinguished, namely, the original condition, the passive state and the pseudo-active state. As a result of the peculiar behavior of magnesium anodes the following interesting oxidations and reductions may be produced at the same: In neutral solutions, $KMnO_4$, K_2CrO_4 are reduced, I-ions are oxidized to I^2. In alkaline solutions Br-ions and I-ions are oxidized to BrO_3 and IO_3 ions respectively, $KMnO_4$ is reduced to K_2MnO_4, Cl-ions are not changed. The author considers the passive state in alkaline solutions as due to a non-conducting film which contains scratches through which conduction takes place.

Copper and Zinc in Potassium Cyanide Solutions.—A long paper on this subejct is published by Fritz Spitzer in the "Zeit. f. Elektrochemie," June 9 and 23. The first part deals with potential measurements, the second with electroanalysis. The potentials of copper and zinc towards their potassium cyanide solutions depend upon the concentration of the salt solutions and their content of KCy. They are higher for higher concentration of the salt solution, and are increased by increasing the KCy content. These influences manifest themselves more strongly with copper than with zinc. When mixed copper- and zinc-potassium-cyanide solutions are electrolyzed, zinc is deposited together with copper long before the proper potential of pure zinc is reached. This proves electrolytic brass to be a true alloy and not a mixture. In his electroanalytic experiments the following results were obtained: When zinc or copper are deposited from potassium cyanide solutions, the platinum anodes are attacked and the platinum which dissolves at the anode is deposited on the cathode. Hence the determination of copper or zinc cannot be exact. The complete deposition of copper and zinc takes a very long time, since the last traces are deposited only from solutions free from cyanide, and the removal of the cyanide from the solution by anodic oxidation requires considerable time. The anodic oxidation is quickest with a low alkalinity of the solution (about 0.2 normal). In general the addition of potassium cyanide is a superfluous complication of the quantitative electroanalytic deposition of copper or zinc.

Alternating-Current Electrolysis.—Some further experiments are described by A. Brochet and J. Petit in "Zeit. f. Elektrochemie" of July 14. The general result confirms the view that the total result is the sum of the anodic and cathodic actions which follow each other continually at each electrode. The ions discharged during one-half wave must be removed in some way from the sphere of action in order to prevent their passing again into the ionic state. Reaction velocities have a considerable influence, so has the frequency and the current density. An increase of current density facilitates the discharge of free ions and increases their number per unit of surface. During the reactions the electrodes sometimes change their nature and become spongy. The most interesting results are the solution of Pt and Pb in H_2SO_4 and of Pt and Fe in cyanide, and the change of electrode surfaces into a spongy condition.

Iron Titration Coulometer.—In "Zeit. f. Elektrochemie, Aug. 4, Z. Karaoglanoff describes an iron titration coulometer (voltameter) which is suitable for exact measurements, especially of small currents. The principle is the cathodic reduction of a ferric salt solution to ferrous salt, together with titration with $KMnO_4$. The construction of a coulometer is described by which a quantity of electricity passing through it may be determined with a mean error of 0.23 coulomb.

Colloidal Solution of Tellurium.—In the "Zeit. f. Elektrochemie," Aug. 18, E. Mueller and R. Lucas describe the formation of a colloidal solution of tellurium by cathodic disintegration. The loss of weight of the cathode is proportional to the coulombs passed, the tellurium behaving as a monovalent element.

Temperature and Speed of Development of Animal Life.—In "Zeit. f. Elektrochemie," Aug. 18, R. Abegg points out some interesting analogies between the influence of temperature on the speed of development of the animal life from eggs and the influence of temperature on the speed of chemical reactions. For the temperature coefficient of the latter Van't Hoff has given a general rule and Abegg shows now that this rule also applies to those cases of animal life investigated, namely, the eggs of echinus microtuberculatus, and of sphaerechinus granulatis, as well as Hertwig's researches on rana. In "Zeit. f. Elektrochemie,' Oct. 20, A. Kanitz gives analogous considerations with respect to the influence of temperature on carbon dioxide assimilation by plants.

Electrolytic Production of Nitrites from Nitrates.—In "Zeit. f. Elektrochemie," Aug. 11, E. Mueller and F. Spitzer give results on the electrolytic reduction of nitrates to nitrites on cathodes of different materials. Spongy silver was found to be the best cathode material, spongy copper the next best material; gold is unsuitable.

Stearic Acid.—Some experiments on the electrolytic reduction of oleic acid to stearic acid are described by J. Petersen in "Zeit. f. Elektrochemie," Aug. 25.

Rotating Cathode.—C. P. Flora has formerly studied the use of the rotating cathode for the estimation of cadium taken as a sulphate. In the November issue of the "American Journal of Science" he describes a series of experiments on the estimation of cadmium taken as the chloride with the aid of the rotating cathode.

Tantalum and Hydrogen.—Tantalum metal, incandescing in the atmosphere of a gas, is considerably modified in its properties. M. von Pirani gives in "Zeit. f. Elektrochemie," Aug. 25, an account of the changes of tantalum when incandescing in a hydrogen atmosphere. There is both a chemical combination of tantalum and hydrogen and an occlusion of hydrogen. The properties of tantalum are modified by the chemical combination only.

Solution Tension.—In "Zeit. f. Elektrochemie," Aug. 4, C. Fredenhagen endeavors to give the foundations of a general theory of the electrolytic solution tension of different materials in different solvents.

Semipermeable Membranes of Clay.—P. Rohland remarks in "Zeit. f. Elektrochemie," July 14, that not all clays can be used for semipermeable membranes. Highly plastic clays in air-dried condition are most suitable.

Iron and Steel.

Bessemer Open-Hearth Practice.—C. Canaris describes in "Stahl und Eisen" for Oct. 1, the combined Bessemer and open-hearth steel practice now being followed at Witkowitz, in Austria. The larger part of the silicon, manganese and carbon is removed by blowing in an acid-lined converter, giving hot metal for further treatment in the basic-lined open-hearth furnace. Three converter charges, of 7 to 10 tons each, are run off in succession in 6 to 12 minutes each, and poured successively into the Siemens furnace. In the latter there is placed about 1 ton of burnt lime, ½ ton of iron ore, ½ ton of pig iron, a ton of steel turnings, and afterwards added 1-5 ton of 77 per cent ferromanganese. The output of steel is about 90 per cent of the weight of pig iron and scrap used. The open-hearth treatment lasts 3 hours. With two Bessemer converters of 10 tons capacity, and three 25-ton Siemens furnaces running, the plant turns out 450 to 500 tons of steel ingots daily. Test analyses showed:

	Pig Iron.	Blown Metal.	Finished Metal.
Si	1.40	0.044	traces
Mn	3.30	0.310	0.300
C	3.08	0.143	0.100
P	0.303	0.318	0.013
S	0.025	0.029	0.021

The advantages of this method of working are undoubtedly a large output, because of the quick heating up of the steel, but at the sacrifice of smaller percentage output than regular open-hearth work. Under the conditions prevailing at Witkowitz it has paid, for many years, to run in this manner, and thus it forms a very interesting example of successful practice, under unusual circumstances, of a combination usually regarded as impracticable.

Equilibrium in the Blast Furnace.—R. Schenck and F. Zimmermann have carried out at the University of Marburg some extensive tests on the chemical equilibrium between iron, ferrous oxide, carbon, carbon monoxide and carbon dioxide. Their results, as given in "Stahl und Eisen" for Oct. 1, are that carbon monoxide deposits carbon by the reaction $2CO = C + CO^2$ only in presence of free iron (or free metals of the iron group), and is not caused by the oxides of these metals. Carbon monoxide does not oxidize metallic iron directly, but indirectly, in that the iron causes deposition of carbon and formation of CO^2, which latter then oxidizes iron to FeO, regenerating CO. This reaction proceeds until complete equilibrium between all five substances is attained, which can only occur, according to the phase rule, for a single partial pressure of each of the two gases and at a single total pressure on both gases. If the partial pressures of CO and CO^2 in the furnace, taken together, are greater than this single critical pressure, then oxidation of the iron can take place, with deposition of carbon; if the pressure is less, carbon can be deposited but no iron oxidized.

The value of this critical equilibrium pressure varies only with the temperature, and was determined as follows:

At 500° C. = 15 mm.
550° C. = 35 "
600° C. = 70 "
650° C. = 145 "
700° C. = 305 "
750° C. = 535 "
800° C. = 800 "

Since the partial pressures of CO and CO^2 are seldom less than 250 mm., reduction can take place down to 690° C., and below that it stops. A curious conclusion from this would be that the smaller the percentage of CO and CO^2 in the furnace gases the lower the temperature at which reduction would continue to take place; and that the more nearly the furnace was run by pure oxygen, the higher the temperature would be at which reduction would stop. This is a new point of view, which may have a practical bearing on blast furnace running under some circumstances.

Blast Furnace Design.—J. L. Stevenson, in "The Engineer" for Aug. 25, discusses the manner in which a blast furnace of given capacity can be designed. His methods may be practical and all right, but they are curious. First of all, if it is intended to make, say, 260 tons of iron in 24 hours, the designer commences by the statement that a furnace to make this amount should be 80 feet high and 18 feet in maximum diameter at the bosh. The diameter of crucible is taken as 0.63 of the bosh, and that of the throat 0.67 of the bosh. The make of iron being about 11 tons an hour, the slag notch is made 4 feet 3 inches above the floor of the crucible, so as to leave ample space for the iron, and the tuyeres are placed 2 feet above the slag notch. The blast required is assumed as 140 cubic feet per minute per ton of iron made in 24 hours; and the tuyere area required as 1¼ square inches for each 140 cubic feet of blast per minute. Ten 6½-inch tuyeres will suffice for the furnace, furnishing the 36,400 cubic feet required per minute (the pressure required is not stated or calculated). The hot-blast stoves should have a heating surface altogether of 444 square feet for each 140 cubic feet of blast heated per minute, or altogether 115,440 square feet.

Some approximate formulæ (of only very limited application, however), are:

(1) The output, in tons per day, equals the cube of 0.35 times the diameter at the boshes, expressed in feet.

(2) The output, in tons per day, equals the cube of 0.08 times the height, expressed in feet.

(3) The output, in tons per day, equals the square of the diameter at the boshes, times the height (all in feet), divided by 100.

These formulæ can be used only in comparing furnaces of nearly similar dimensions, and even then they will be often far from the truth.

Avoiding Shrinkage Cavities.—F. O. Beikirch describes in "Stahl und Eisen," Aug. 1, a method of avoiding shrinkage cavities in heavy steel ingots which has been put in successful use during this year at the Gutehoffnungshütte at Oberhausen. The idea is to keep the steel in the upper part of the ingot fluid by the use of a producer gas blowpipe, so that all shrinkage cavities can be filled. The apparatus consists of a small gas producer on a track, supplied by air blast from a knee-jointed air pipe. Part of the air passes through the coke fire, making producer gas, to which the rest of the air is supplied in the pipe leading into the top of the ingot mould. The warm air burning the hot gas completely produces a very hot flame. The gas is thus burned in the empty mould for some minutes previous to casting, then the pipe removed while the ingot is cast, and then the cap replaced and the heating continued. The cost of the treatment is given as 15 to 25 cents per ton of steel; the cost of the apparatus $1,250 to $1,500. The amount of top which must be cut off the ingot is thus reduced from 10 to 15 per cent of the weight of ingot to an average of 5 per cent; and 5 per cent good metal thus obtained instead of 5 per cent of scrap is well worth the steel-master's attention.

Grondal Process.—The International Exposition at Liége is reviewed in a series of articles in "Stahl und Eisen," commencing June 1. The number of Aug. 1 contains an extensive table, taken from the exhibit of the Stockholm firm, "Metallurgiska Patent Aktiebolaget," of the results of magnetic concentration of iron ores by the Gröndal process. The fine ore is concentrated, briquetted and roasted. American ore, containing 50.65 iron, 1.603 sulphur, and 0.012 phosphorus, was concentrated to 69.95 iron, 0.036 sulphur and 0.003 phosphorus, the residues carrying only 19.20 of iron. Swedish ore carrying 1.29 per cent phosphorus was concentrated to 0.011 phosphorus. Altogether some sixty-two specimens were shown, with analyses, and formed a very striking and instructive exhibit of the capacities of this system.

Ferrotitanium.—The September issue of "Cassier's Magazine" contains a useful and interesting article by A. J. Rossi on ferrotitanium. The author first describes the manufacture of ferrotitanium, either by the Goldschmidt process or by his own electric furnace process (our Vol. I., p. 523), and then gives a summary of the various applications of ferrotitanium in the steel industry.

LEAD.

Huntington-Heberlein Process.—In our October issue (p. 363) we discussed the principal features of the Huntington-Heberlein process of roasting galena, which is now in the foreground of interest owing to its adoption at several smelters of the American Smelting & Refining Co. In this connection several articles recently published are very interesting. In the "Zeit. f. d. Berg-Huetten u. Salinen-Wesen," No. 2, A. Biernbaum gives comparative results obtained in an Upper-Silesian smelter up to 1900 with hand-roasting furnaces, and since that time with the Huntington-Heberlein process. The advantages of the latter process are indicated in figures showing the reduction of workingmen required, a very essential reduction in the cost of maintenance of the furnaces, a large reduction in the coal consumption and improved sanitary conditions. Since the roasting temperature is smaller, there is less volatilization of lead and less flue dust. Whatever the percentage of zinc in it the ore is always roasted in the same furnaces (whereas formerly different ores were roasted in different furnaces). The total charge of the blast furnace is now richer in lead than formerly, and a larger output can now be obtained. The reduction in the number of workingmen is largely due to the introduction of mechanical machinery, but the cost of power for this purpose is stated to be small. As disadvantages of the process are mentioned the difficulty of breaking up the roasted cake from the converter and the higher contact of sulphurous acid in the waste gases; it is hoped, however, that it may become possible to utilize the latter for the manufacture of sulphuric acid.

The theory of the Huntington-Heberlein process is the subject of various articles. In "Metallurgie" of Sept. 22, C. Guillemain gives a very full discussion of the probable reactions. He does not agree at all with the hypothetical reactions of the inventors, who assumed the formation of calcium peroxide as the principal reagent. In his opinion the addition of lime has no essential chemical effect, but influences the physical condition of the material in such a way that it may serve as a heat reservoir to prevent the localization of heat with a resulting melting of galena, instead of roasting.

In connection with this subject, a paper by F. O. Doeltz, in "Metallurgie" of Oct. 8, is also interesting. It refers to a claim of A. D. Carmichael to the effect that a mixture of lead sulphide and calcium sulphate, when heated to 400° C., reacts according to the equation:

$$PbS + CaSO_4 = PbSO_4 + CaT$$

Doeltz describes a series of experiments which show that this reaction does not take place. On the contrary, when lead sulphate and calcium sulphide come together there is a tendency to form lead sulphide and calcium sulphate. Nevertheless, when a mixture of lead sulphide and gypsum is heated in air, lead sulphate will be formed besides lead oxide, but not as the result of a reaction between lead sulphide and calcium sulphate, but as a result of a reaction of lead oxide and contact sulphuric acid:

$$PbO + SO_2 + O = PbSO_4$$

This is the reaction which regularly takes place when lead sulphide is roasted, and the explanation of which was already understood by Plattner.

Direct Desulphurization of Galena by the Air Blast.—It is well-known that the Huntington-Heberlein process has already found competitors. One of them, A. Savelsberg, describes his process as operated at Ramsbeck, in Westphalia, in the November issue of the "Mining Magazine." The principal distinguishing feature is the desulphurization of raw galena directly in the converter without any preliminary roasting. He introduces into the converter a mixture of raw lead ore and limestone, containing a sufficient amount of silica for the formation of silicates. A current of air is then blown through it, while at the same time that portion of the mixture which is adjacent to the opening where the air enters is ignited. Very active reactions are produced by the combustion of the sulphur of the ore. There are produced lead oxide, sulphurous acid, carbonic acid, sulphuric acid, sulphates and silicates. Sulphurous acid and carbonic acid escape, and a part of the sulphates acts directly, oxidizing on undecomposed galena. Another part of them is decomposed by the silica. During this process sulphuric acid is set free, which acts as a powerful oxidizing agent. The lead oxide thus formed combines finally with the gangue of the ore and the nonvolatile components of the additions to the desired roasted product. Several precautions must be taken during operation. A good mixture of the charge must be carefully looked after. The converter is not filled completely at once, but the charge is added gradually in thin layers. In order to prevent an unmixing of the mixture of ore and limestone, by reason of the blowing away of the light particles of limestones, it is requisite to add a comparatively large amount of water to the charge before it is introduced into the converter.

RECENT METALLURGICAL PATENTS.

Refining Copper-Nickel Matte.—In separating the sulphides of nickel and copper by the Orford process, as operated at the Bayonne works of the Orford Copper Company, the practice is to subject the molten matte to contact with sodium sulphide (produced generally by mixing sodium sulphate and coke with the matte), smelting in a cupola furnace and causing the copper and iron sulphides, which are dissolved in the resulting sodium sulphide, to separate under the influence of gravity from the heavier undissolved nickel sulphide. A recent patent of R. R. Maffett (802,148, Oct. 17) refers to a new simplified treatment of the resulting copper-iron tops. These tops are piled in the open, where the action of the weather oxidizes the sodium sulphide to sodium sulphate. They are allowed to weather about eight weeks, whereby the greater part of the sodium sulphide is oxidized to sulphate. As soon as the tops, thus treated, are smelted, the sodium sulphate rises to the top of the charge and is drawn off into pots, when it is allowed to cool and is returned to the nickel cupolas, where it is used in place of refined salt cake. The conversion of the sodium sulphide to sodium sulphate is completed by oxidation preferably by an air-blast, and the oxidation is continued until all or nearly all of the sodium sulphide is converted into sulphate. The molten contents stratify in the furnace, and the different layers are tapped separately into different molds. By skimming or tapping off the floating layer of sodium sulphate or the layers of sodium sulphate and iron silicate and leaving the copper sulphide remaining in the furnace, the inventor is enabled to proceed in the same furnace with further refining of the copper.

Detinning.—The industry of detinning tinned sheet iron for the separation of tin and iron has so far only utilized tin scrap, bought of the manufacturers of tinned-iron cans, etc., as commonly used for preserves and canned goods. The cans and boxes themselves have scarcely been utilized up to the present, for several obvious reasons. One is that in their original form they cannot be used for the electrolytic process. To make them available for this purpose, they must be compressed so as to occupy a small room only, and must also be dressed so that the dissolving agent can circulate at both sides of the plates from which the tin is to be taken off. To accomplish this, a recent patent of Hans Goldschmidt (804,530, Nov. 14) describes means for compressing the cans and boxes between mechanically-moved surfaces and for perforating the boxes to allow the electrolyte easy access to all metallic parts.

Roasting Furnaces.—In the McDougall type of roasting furnace the dropping of the charge from one hearth to the next one and the raking off of the ore after discharge produces dust, which agglomerates as "secretions," or "bearings" on the roof of the upper hearth, and must be removed from time to time. This is very inconvenient, since it necessitates shutting down the furnace periodically. To overcome this disadvantage, F. E. Marcy (802,207, Oct. 17) provides the rabble arms with an extended plate or surface adapted to come directly over the discharge openings, so that the dust is deflected and intercepted before it reaches the upper hearth. Whatever accretions form on the extended surface mentioned before, can be removed or borne off from time to time through the furnace doors without closing down the furnace.

A. R. Meyer (804,751 and 804,752, Nov. 14) patents mechanical details of construction of the hollow central shaft and the hollow stirring arms so as to properly pass a cooling fluid through them. As cooling fluid he uses air saturated with water vapor.

The well-known Herreshoff roasting furnace has proven very successful in treating dry ore or other material, such as American pyrites, which can be stirred up during the roasting process without sticking or conglomerating in large masses. In the treatment of material containing considerable moisture, such as washed Spanish fines, there is danger that, if not previously dried, this material blocks the feed and cakes and balls up in such large masses that the stirrers cannot break it up, and the stirrers themselves are often broken. For this reason H. Howard (804,227, Nov. 14) provides a drying chamber on top of the furnace where the material is stirred and dried by means of the waste heat of the furnace, before it is fed into the roasting furnace proper.

A. W. Chase (804,379, Nov. 14) patents a roasting furnace of the following construction: It consists of a series of troughs located one over the other and being U-shaped in cross-section and composed of fire-clay sections. The charge feeds from one trough to the next one. In the troughs conveyors are provided with hollow shafts. They are driven at increasing rates of speed from the top downward, while a cooling fluid is passed through the hollow shafts.

Gold and Silver.—T. B. Joseph (805,017, Nov. 21) patents the following modification of the cyanide process. The crushed ore is subjected to the action of a leaching solution containing water, sodium cyanide, calcium hydrate, ammonium carbonate, ammonium nitrate and sodium bicarbonate, while simultaneously compressed air is forced upward from the bottom of the tank from an air-compressor. If there was no carbon dioxide with the ammonia in the leaching solution, then the ammonia therein might decompose some of the cyanogen into its component parts and ruin its power to dissolve gold; but the carbon dioxide from the ammonium carbonate protects the cyanogen from such destruction. The cyanogen in the solution dissolves the gold, while the ammonium causes a better extraction of the silver, as well as of the copper or gold, with this solution than the straight cyanide process would yield. The inventor gives the following cecipe: 1 ton of water, ½ to 5 pounds sodium cyanide, 1 to 50 pounds ammonium carbonate, ½ pound ammonium nitrate, and about enough of calcium hydrate to contain 1 pound of oxide of lime, and 2 pounds sodium bicarbonate.

E. Parrish (805,229, Nov. 21) patents a slimes filter of the following construction: The lower part of the horizontal filter tank is divided into several compartments. A filter body is placed over the same, and above it is the slimes chamber, which is common to all the compartments. The tank is rotated and a vacuum is produced in these compartments. It is thereby possible to separate a large percentage of the solution from the slimes at a single operation.

BOOK REVIEWS.

Experimental Electrochemistry. By N. Monroe Hopkins. 284 pages and 130 illustrations. New York City: D. Van Nostrand Co. Price $3.00.

This work of Dr. Hopkins, the well-known professor of George Washington University, is an elementary textbook for students in electrochemistry. In the words of the author it has been his aim "to produce a book that will prove useful in both the lecture room and laboratory." It is written in an easily understood and interesting style, and the directions for performing the numerous experiments, which cover a large range, are practically full. The book is well printed and the numerous illustrations will add greatly to its usefulness.

The first six chapters, occupying 116 pages, are devoted to fundamental principles in which the theory of electrolytic dissociation, electrolytic induction and conduction, Faraday's law and energy requirements are discussed and illustrated by experiments.

In the following chapters directions are given for making many compounds, both organic and inorganic, electrolytically as well as for the isolation of the metals sodium, potassium, aluminium and calcium, and a valuable feature of the book is the short historical sketch of the work that has been done in connection with their preparation.

Eléments de Chimie Inorganique. Par Prof. W. Ostwald. Traduit de l'Allemand par L. Lazard. Two volumes. Paris: Gauthier-Villars. Price 28 francs.

Prof. Ostwald's writings are ever marked by clearness of conception and facility of diction in the original German. His "Elements of Inorganic Chemistry" is an epoch-making treatise. This has been rendered into French by Prof. L. Lazard, and the lucidness of style and thought is, in our opinion, almost enhanced in its gallicized form. For English-speaking chemists who read French more easily than German, these two volumes are to be recommended, for they are the product of a brain that looks on chemistry in a broad, philosophical light.

The Copper Handbook. A manual of the copper industry of the world. Vol. V. for the year 1904. By Horace J. Stevens. Houghton, Mich.: Horace J. Stevens. 882 pages. Price $5.00.

Mr. Stevens' annual Copper Handbook is now in its fifth year, and a comparison with former volumes shows that the author has taken great care and much pains in making the book thoroughly up to date.

The Handbook comprises sixteen chapters. The first six are of an introductory general character, and deal with the history of copper, geology of copper, chemistry and mineralogy of copper, the uses of copper and a glossary of mining terms. Then follow eight chapters on copper deposits in different countries.

The main value of the book is to be found in the last two chapters, on copper mines of the world and copper statistics. These two chapters have been entirely rewritten for the new edition, and contain an enormous amount of useful information. While the author expressly disclaims all pretensions to infallibility, he says that every honest effort has been made to verify the assertions given in the book, and that for every statement of fact therein, verification or corroborative evidence of some sort can be furnished. "Very naturally, the men and corporations denounced in former editions of the Copper Handbook are not pleased at the unenviable notoriety given them in a publication of good standing, circulating throughout the world, but it may be said, as an evidence of the thoroughness with which this work has been done in past editions, that, notwithstanding the many threats uttered against the author of this book, ranging from simple threats of corporal chastisement to promises of suits for criminal libel, the author has yet to be thrashed and yet to be sued for any statement contained in the past editions of the work."

The chapter on copper mines of the world covers 684 pages, *i. e.*, over three-fourths of the book. It contains 3,849 titles, with from two lines to fourteen pages devoted to each. The information given covers the names of the principal officers, capitalization, development and present status of the mine property and the processes used. From the statistics given in the last chapter it appears that the average decennial percentage of increase for the last century was 53.91 per cent; for the last half of the century 67.90 per cent; and for the last twenty years, during which the electrical industry became a great consumer of copper, the average decennial increase in copper output was 77.83 per cent.

The Féry Radiation Pyrometer.

In all pyrometers other than radiation pyrometers there is some part—the sensitive or receptive part—which is made to acquire a temperature identical with the temperature to be measured. With radiation pyrometers this is not the case. For measuring furnace temperatures, for example, with a radiation pyrometer, the pyrometer is entirely outside the furnace.

The advantage of this fact is self-evident, since it is difficult to construct anything of solid material which can be maintained for prolonged periods at a high temperature without suffering some permanent or sub-permanent change in its physical properties, and as we ascend higher in the temperature scale the difficulties increase in a quite disproportionate degree. A further aggravation of the trouble is to be found in the chemical activities of furnace products and furnace gases, which in some cases render difficult the adequate protection of a resistance wire or thermo-couple. All these troubles are manifestly absent in radiation pyrometers, since the instruments are, of course, placed at some distance from the furnace, while no part of them is raised above the air temperature by more than 80 C. degrees. In the following we describe the type of radiation pyrometer, invented by M. Féry, professor of Physics at the Ecole de Physique et de Chimie in Paris.

In the Féry pyrometer the radiation which emanates from a hot body, or which passes out through an observation hole in the wall of a furnace, falls upon a concave mirror or upon a system of lenses, as the case may be, and is thus brought to a focus. In this focus is a thermoelectric couple, whose temperature is raised by the radiation falling upon it; the hotter the furnace the greater the rise of temperature of the couple.

The arrangement of the instruments is such that they are uninfluenced within wide limits by the size of the hot body or observation hole on the one hand, or on the other hand by the distance which separates them from the hot body or furnace.

The absorption of some small amount of radiant heat in passing through the atmosphere cannot, of course, be strictly without effect; but in practice the error thus arising is not appreciable; it has been found, for example, that the reading obtained for the temperature of a stream of molten steel was precisely the same—1,200° C.—whether the instrument was set up 3 feet or 60 feet away.

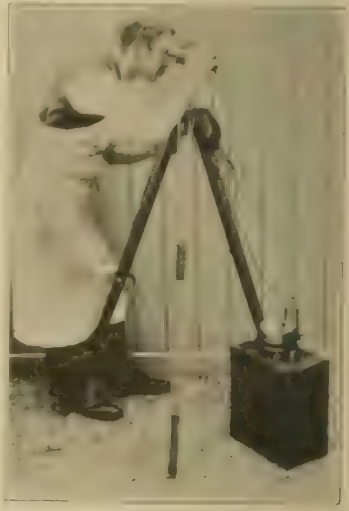

FIG. 1.—FÉRY OPTICAL PYROMETER.

A complete instrument consists of telescope and galvanometer; fixed within the telescope, at a point upon its optic axis, is the junction of a copper-constant thermo-couple arranged in the form of a cross. The two wires are attached to two brass strips D and R, which are attached to the terminals bb′, Fig. 2. The terminals are connected by leads to the galvanometer as shown in Fig. 1.

To use this apparatus for measuring the temperature of a furnace an observation hole in the wall of the furnace is sighted through the eyepiece O, the image of this hole being brought into coincidence with the thermo-junction. It is essential that the image of the observation hole should slightly overlap the junction, which appears to the eye as a black disc in the center of the field of view. The readings of the instrument are then independent of the size of the observation hole.

In the mirror instrument the image of the hole is reflected to the eyepiece O by two small plane mirrors placed close to the couple. These mirrors serve for the adjustment of focus: they are so arranged that the image of the hole appears to be split into two parts, which only blend together to form an unbroken image when the focussing is correct. Fig. 3 illustrates the changes in the appearance of the field as the focus is altered. The middle figure shows exact focus, while the left and the right-hand figures refer to too long and too short a focus respectively.

The image thus formed upon the junction produces a rise of temperature which is shown experimentally to be proportional to the amount of radiant energy which enters the telescope. The junction acquires exactly, and with great rapidity, the temperature of the image, but in no case does its temperature rise by more than 80 C. degrees above the atmospheric temperature.

The e. m. f. which is thus generated is measured by a highly sensitive galvanometer whose scale is divided and figured so as to read temperatures directly. This galvanometer, notwithstanding its sensitiveness, is of substantial construction; it comprises a finely pivoted moving coil, provided with a long pointer. The movement of the coil takes place within a single air gap between the poles of a strong permanent magnet.

These galvanometers are invariable in their indications, and unaffected by all external influences. They are self-levelling, so that no adjustment of level is required before a reading is taken. By turning a milled head the moving coil can be freed when a reading is to be taken, or clamped to facilitate the removal of the instrument. Another milled head enables the divided scale to be unclamped, adjusted in position to a small extent and reclamped, so that the reading is zero when no current is flowing. The galvanometer has a resistance of about 10 ohms, and is connected by leads about 10 meters (33 feet) long to the terminals of the pyrometric telescope.

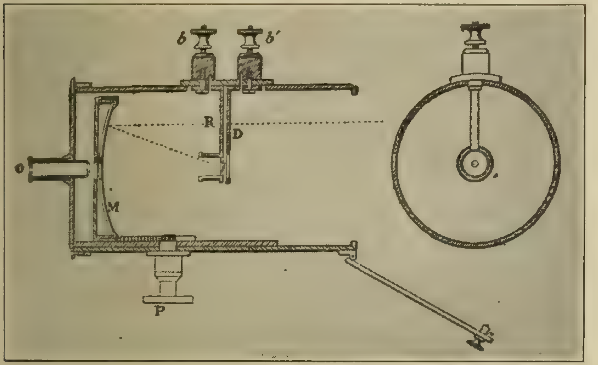

FIG. 2.—CONSTRUCTION OF FERY PYROMETER.

The graduation of the galvanometer scale is based upon the Stefan-Boltzmann law, which expresses the relation between the temperature of a body and the amount of total radiant energy which it emits. The law is as follows: "The radiant energy emitted by a perfectly black body is proportional to the fourth power of the absolute temperature of the body." This law was found by Stefan as a result of experiments, and was later deduced by Boltzmann from thermodynamical reasoning.

In the law, as above pronounced, absolute temperature means, of course, temperature in Centigrade + 273; the statement of the law should be more exact that the total radiated energy is proportional to the difference of the fourth power of the absolute temperature of the radiating body and the fourth power of the absolute temperature of the body on which the radiation falls.

The term "perfectly black," used in the above statement, is, of course, employed in its thermodynamical meaning, namely, that it emits and absorbs all waves of all different wave lengths. This does not mean at all that a body which is called "perfectly black" in thermodynamical sense, has what we call in ordinary life the black color. For instance, as was already proven by Kirchhoff, a furnace, the interior walls of which are on a high uniform temperature and which has only a small hole in the walls, through which it emits radiation (which is observed and measured by a radiation pyrometer), behaves for all practical purposes like an ideal "perfectly black body," although in ordinary life we would call the interior of the furnace red-hot or incandescent white.

If the inner wall of an enclosure E, Fig. 4, is at one temperature throughout, and if an aperture A through which radiant heat emerges is of small dimensions, compared with the depth of the enclosure behind the aperture, then the radiation proceeding from the surface B of the enclosure will be independent of the character of the surface, and the same as if the surface B were perfectly black. The arrangement sketched in Fig. 4 is one which is actually used in taking furnace temperatures by means of the Féry pyrometer.

Moreover, in many industrial processes, a knowledge of actual temperatures—in Centigrade or Fahrenheit—is not essential, provided some standard is available which serves to distinguish between higher and lower temperatures, and enables correct heat conditions, once obtained, to be reproduced with certainty. Suppose, for example, that we are sighting the pyrometer on the metal in a crucible, and that the reading obtained on the temperature scale of the galvanometer is 1,100° C. Owing to the surface of the metal not being absolutely black this reading may be lower than the true temperature, which we will suppose is 1,180° C.

If we find that the castings obtained by pouring the metal under these heat conditions are satisfactory and better than those resulting from pouring either hotter or colder metal, we shall have reached the important practical conclusion that the correct pouring temperature for the metal in question produces a reading of 1,100° C. on the temperature scale of the galvanometer—even though we do not know the true temperature of the metal on the centigrade scale.

In other words, the indications of the pyrometer are always consistent in themselves, so that it is possible to obtain a permanent record of the temperatures of a metallurgical process. When the proper temperatures have once been determined, they can always be duplicated with the aid of the radiation pyrometer—and this is all that is really needed in metallurgical practice. Even in such cases where the radiation is very different from that of a "black body," as with the white-hot gases from a Bessemer converter (where the optical pyrometer would give temperatures several hundred degrees below the actual temperature), the radiation pyrometer will give the practical man everything he needs, because the indications of the temperatures given by the radiation pyrometer (though not the actual temperatures on the gas-thermometer scale) are consistent in themselves. Therefore, when once the proper series of temperatures in a metallurgical process has been determined and reduced to a permanent record by means of the radiation pyrometer, it can be infinitely repeated. Whether the temperature scale of an optical pyrometer agrees with the temperature scale of the gas thermometer, is a question of no practical interest for the metallurgical or chemical engineer.

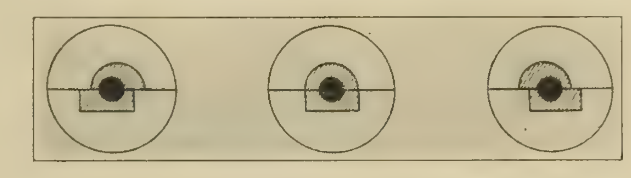

FIG. 3.—EFFECT OF CHANGE OF FOCUS.

Since the energy radiated from a hot body increases very rapidly as the temperature is raised, it follows that the Féry pyrometer is far more sensitive at high than at low temperatures. Temperatures as low as 600° C. can be read, but the instrument is most useful for high-temperature work. As examples of high-temperature measurements made with the Féry pyrometer, we may mention the temperature of the sun (7,800° C.), determined by Prof. Féry, as well as the temperature of the iron in a thermit mould, which was found to be 2,500° C. (over 4,500° F.).

The instrument is thus of especial value for taking such high temperatures as those of molten steel, of gas settings, of glass furnaces, brick kilns and electric furnaces. Its great flexibility and the readiness with which it is sighted enable it to be used for taking the temperature of metal in a crucible just before pouring, thus ensuring correct casting temperatures, a point which is now known to be of especial importance in the case of steel castings. Forgings, billets, blooms and finished rolled sections are also readily followed through the various operations which have to be performed on them, great progress having already been made in this direction.

A point which may need further explanation is the arrangement that ensures that, within limits, the readings of temperature shall be independent of distance.

Suppose that we are sighting the telescope upon a hot body of limited dimensions. The total amount of radiation reaching the aperture of the lens or mirror will vary with the distance from the hot body, and will be inversely proportional to the square of the distance. If, then, the receptive surface of the thermo-junction were sufficiently extended to receive the whole of the radiation which is converged to a focus by the lens or mirror, we might expect the indications of the galvanometer to fall off as the distance was increased. The thermo-junction, however, is not large enough to receive the whole of the radiation which converges towards it. The real image of the source, formed by the lens or mirror, overlaps the thermo-junction on all sides (Fig. 5a), so that when we approach the source more nearly, thus increasing the size of the image produced, the only effect is to increase the amount of overlapping, while the thermo-junction receives no more radiation than before (Fig. 5b). On the other hand, if we withdraw to so great a distance that the image formed is too small to cover the thermo-junction completely (Fig. 5c) the readings obtained will be too low, and will become lower and lower as we recede further and further from the source of heat.

FIG. 4.—APERTURE FOR MEASURING FURNACE TEMPERATUES.

From the above it will also be readily understood that when we are working at any given distance from the hot body, it is necessary for the body in question to be of sufficient size, otherwise the image of the body would be too small to overlap the thermo-junction on all sides. But provided the hot body is large enough to secure the necessary overlapping, no further increase in its dimensions will add to the amount of radiation actually received by the thermo-junction. The diameter of the hot body (or furnace aperture) should measure as many inches as the distance from hot body to pyrometer measures yards.

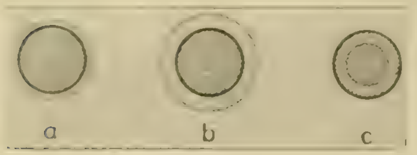

FIG. 5.—EFFECT OF CHANGE OF DISTANCE.

A similar galvanometer is also made and calibrated so as to measure directly low temperatures, the temperature of liquefaction of gases, liquid air and liquid carbonic acid, or for the measurement of moderate temperatures between 0° and 600° C. In such cases the galvanometer is connected by leads to a copper-constantan thermo-couple immersed in the medium whose temperature is required. The couple is enclosed in a tube which leaves the junction exposed. These couples should never be heated above 700° C.

Thus optical pyrometry has now conquered the whole range of temperatures, down to the lowest and up to the highest degrees, and has achieved this result by simple and practical means.

The sole agents for the sale of this instrument in America, Great Britain and her colonies, Norway and Sweden are the Cambridge Scientific Instrument Co., Ltd., in Cambridge, England.

Hot-Blast Smelting.

In an article in our October issue (page 400) we discussed at length the use of hot blast in its metallurgical consequences and the savings resulting therefrom.

It is, of course, always possible to install a special U-pipe stove for heating the blast, but this may be avoided by utilizing the heat of the furnace itself for this purpose.

Fig. 1 represents the end elevation and section of a modern copper matting hot-blast furnace, built by the Traylor Engineering Co., 118 Liberty Street, New York City, while Fig. 2 represents side elevation and section showing the arrangement of the hot blast attachment of the same furnace. This furnace represents particularly the adaptability of Mr. Joseph L. Giroux's hot-blast equipment.

The illustrations are almost self-explanatory, and it will be seen that the cold blast from the blower enters a horizontal pipe near the top of the furnace and extending transversely across from side to side. The air is then delivered to a series of vertical pipes, through which it passes alternately through the first from top to bottom and then through the next pipe from the bottom to the top, and so on, as indicated by the arrows. The air is thereby heated and is finally delivered into the bustle-pipe, which is connected through pipes with the tuyeres opening into the lower smelting portion of the furnace.

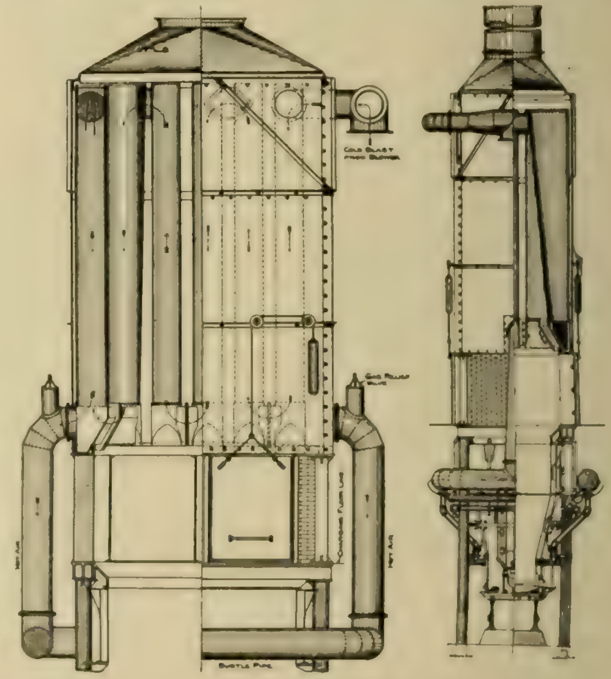

FIG. 1.—COPPER MATING HOT-BLAST FURNACE.

The use of this attachment in connection with all copper-matting furnaces means the saving of from 3 to 6 per cent of coke, this depending largely on the analysis of the ore and the percentage of sulphur available. It is stated that on ores averaging 15 per cent of sulphur and up to 30, the use of this hot blast attachment will more than pay for the cost of installing in from three to six months operation in the saving of fuel alone, as well as materially increasing the smelting capacity over any given size furnace running with cold blast.

Canadian Commission Report.—The excellent report of the commission appointed by the Canadian Government to study electric iron and steel processes in Europe (see our Vol. II., p. 479), has now been translated into French. Otherwise no changes have been made. A new important report is expected from Dr. E. Haanel after the completion of Dr. Héroult's experiments at the Soo.

News and Notes.

American Electrochemical Society.—At the meeting of the Board of Directors, to be held on Dec. 2, the names of the following gentlemen will come up for election to membership: Louis Ruhl, New York City; Arthur Haug, Peekskill, N. Y.; H. A. Prosser, Elizabeth, N. J.; A. W. Lewin, New Orleans, La.; Francis R. Pyne, Bethlehem, Pa.; Harry R. Lee, South Bethlehem, Pa.; J. S. Bridges, Jr., Boston, Mass.

Electric Steel.—Messrs. Naylor & Co., 45 Wall Street, New York City, are the American representatives of the Gröndal-Kjellin Co., Ltd., which owns the patents of Gröndal for iron briquetting and of Kjellin for the electric induction furnace for steel manufacture.

Ruthenburg Steel Process.—According to the Niagara Falls *Gazette* of Nov. 4, a petition of involuntary bankruptcy was filed on Nov. 3 with the clerk of the District Court in Buffalo against the Ruthenburg Reduction Co., on behalf of several local creditors. "The company was forced into bankruptcy owing to difficulties attending the demonstration of the Ruthenburg process for making steel at Niagara Falls. The plant occupied the 'old stone barn,' and an effort was made to adapt it to the work, but it was unsuccessful. The bankruptcy proceedings do not mean, by any means, that the Ruthenburg process is a failure. In fact, it is said that it has practically been demonstrated. It is expected that work will be resumed probably at Niagara Falls on Mr. Ruthenburg's patents by men who are confident of the success of the process. The failure of the Ruthenburg Reduction Co. was caused by the difficulty in obtaining the right equipment for the process."

Oil Concentration.—It is stated that the treatment of low-grade copper ores of the Graham mine at Massey, Ont., by the Elmore oil process of concentrating has proven very successful. During the experimental stage of operating the mine, the Massey Station Copper Co. has expended over $300,000, and the undertaking is now said to be on a sound commercial basis. From a 3 per cent ore a concentrate of 20 per cent standard is turned out. The Elmore oil concentration plant was installed at Massey to treat the tailings from the usual water concentration plant, and has given so good results that the water concentration plant has been abandoned altogether and the Elmore oil plant is now being doubled in capacity.

The Philadelphia Testing Laboratory has been established at 808 Chestnut Street, Philadelphia, for carrying on all lines of inorganic analyses and assays. Mr. J. W. Harris is the director of the laboratory.

German Chemical Industry.—At the annual meeting of the German Association for the Advancement of the Chemical Industry, held in Heidelberg on Sept. 23 and 24, special emphasis was placed on the prosperous conditions existing in this industry, which is justly regarded as the pet industry of the German empire. No other industry in Germany has been so little affected by the business depression of the past four years as the chemical industry. The dividends of 143 stock companies, representing a capital of $157,913,000, average 9.37 per cent in 1904, as against 8.56 per cent in 1903.

Messrs. Queen & Co., of Philadelphia, have taken the entire second floor of the building at Eighth and Arch Streets, and will dispense with their retail store at 1010 Chestnut Street after Jan. 1, 1906. They will pay in future less attention to photographic apparatus and supplies and will concentrate their energies toward the manufacture of instruments of precision, chemical and physical laboratory apparatus and supplies, optical instruments, etc.

Portland Exhibition Awards.—The Superior Jury at the Lewis and Clark Exposition has approved the following awards in the electrical department, relating to the exhibits of the General Electric Co., which is the largest manufacturer exhibiting in that department. The highest award granted by the jury is a gold medal. The General Electric Co. received a gold medal for the best exhibit in the electrical department and also gold medals on each of the following features of this exhibit: Curtis steam turbine, meters and instruments, time limit relays and oil switches, switch boards, meter controlling panels, circuit breakers and lightning arresters, direct and alternating-current motors, direct and alternating-current generators, static transformers, automatic voltage regulators, magnetite arc lamp, alternating and direct-current enclosed arc lamps, mercury arc lamps, magnetic starting device for mercury arc lamps, mercury arc rectifier, railway rotors and controllers, mining locomotives, searchlight and method of control, process and development in the art. For its new metalized carbon filament incandescent lamps the company also received a gold medal.

Platinum Exhibit.—Messrs. Baker & Co., Inc., refiners and manufacturers of platinum, gold and silver, Newark, N. J., and New York City, have been awarded a gold medal for the excellency and variety of their exhibit at the Lewis and Clark Centennial Exposition, just closed at Portland, Ore. Being the first exhibit of its character on the Pacific Coast, and located in the Mines and Metallurgy Building, under Government supervision, it attracted the special attention of a large number of visiting miners and metallurgists, who, in recent years, have been devoting greater care to the recovery of the platinum known to exist in the alluvial deposits along the Pacific Coast. The company's exhibit was comprehensive in its character, including numerous samples of crude platinum ore, the salts and solutions of the metal, and various forms of platinum ware, such as are used daily in the arts and industries. A more extensive exhibit, which they displayed at the Louisiana Purchase Exhibition in 1904, was awarded the grand prize.

Bacteriological Apparatus.—We have received from Messrs. Eimer & Amend, of New York City, their revised and enlarged catalogue (1905) of bacteriological apparatus. The catalogue gives descriptions of the latest types of microscopes and acessories, microtomes, incubators, etc., as well as of general laboratory appliances and apparatus, and is well illustrated.

Sampling Plants.—Catalogue 127 of the Allis-Chalmers Co. contains a great amount of useful information on the problem—which is of fundamental importance in every metallurgical plant—how to prepare a proper ore sample. The pamphlet contains well-illustrated and clearly written notes on sampling mills, ore breakers, crushing rolls, sample grinders, automatic samplers, elevators, conveyors, bin gates, grizzlies, ore barrows, scales, Corliss engines, electric generators, motors and accessories.

Acetylene Laboratory Appliances.—We have received from the Harris Calorific Co., of 2 Clinton Street, Cleveland, Ohio, their illustrated catalogue of acetylene laboratory appliances for colleges, schools, chemists, assayers and for manufacturing purposes. The flame attainable from acetylene burners has a temperature far beyond that which can be obtained with ordinary gas burners, and is suitable for a great many purposes, like welding and brazing. Temperatures approaching the oxy-hydrogen blow-pipe may be obtained at a very low cost.

Ferro-Silicon Manufacture.—The various inherent advantages of the use if high grades of ferro-silicon, made in the electric furnace and containing from 25 to 75 per cent Si over the lower blast-furnace grades of some 10 per cent, are well known to our readers (see, for instance, our Vol. II., p. 122). The fact that the higher the grade the less will be the impurities which are introduced into the steel, was soon recognized by steel men. With the increasing demand for high-grade ferro-silicon many electric furnace plants, which were originally founded to make calcium carbide, turned to the manufacture of ferro-silicon. Competition then become so strong that a syndicate was formed. But the independents are also strongly active. Messrs. George G. Blackwell, Sons &

Co., Ltd., of Liverpool, the well-known British pioneer firm in ferro-alloys, are interested in and represent the largest, newest and most up-to-date ferro-silicon works outside of the syndicate. Mr. George G. Blackwell, chairman of this firm, was recently on an extended trip through the country, visiting all the steel centers. After a visit to Pittsburg and other points he sailed for Europe on December 2.

Personal.

The wedding is announced of Mr. **Paul Oscar Abbe**, of the Abbé Engineering Co., of New York City, to Miss Henrietta Elvira Eck, daughter of Mr. and Mrs. Gustaf F. Eck, of Brooklyn, N. Y.

Mr. **Woolsey McA. Johnson**, of New York City, will deliver, next February, a series of lectures at Harvard University on the subject of "Applications of Physical Chemistry to Metallurgy."

The partnership between Messrs. A. D. Little and W. H. Walker, under the name of Little & Walker, as consulting chemists, has been dissolved by mutual consent. Mr. **Arthur D. Little** continues the business at the present office of the firm, 93 Broad Street, Boston, Mass., while Mr. **William H. Walker** carries on his consulting practice at 24 Trinity Place, Boston.

Dr. **Karl Goldschmidt**, of Essen-Ruhr, Germany, sailed on November 7 for Europe on the "Kaiser Wilhelm II." On the eve of his departure he gave a delightful dinner at the Waldorf-Astoria to his business associates of the Goldschmidt Thermit Company and to a number of gentlemen whom he had met during his short stay in this country.

Prof. **Wilhelm Ostwald**, of Leipzig, Germany, is now in this country, lecturing at Harvard University on physical chemistry. On his arrival in the United States and before he proceeded to Boston, numerous receptions were tendered to him in New York City.

Mr. **M. Fukuda**, chief engineer of the Osaka Electric Copper Refining & Rolling Co., of Japan, is visiting this country, to study the development of copper refining in the United States.

Massachusetts Institute of Technology.—Several changes have taken place in the staff of the Research Laboratory of Physical Chemistry of the Massachusetts Institute of Technology. Prof. W. D. Coolidge has accepted a position in the technical research laboratory of the General Electric Co., at Schenectady, where he will be closely associated with Dr. W. R. Whitney. Mr. Yogoro Kato has accepted a position in the Technical High School of Tokio, where he will have charge of the work in electrochemistry. Mr. Wilhelm Böttger returns as privatdocent to the University of Leipzig, at which he will conduct one of the laboratory courses in analytical chemistry. In place of these retiring members the following gentlemen have received appointments to the research staff: Dr. William C. Bray, Mr. Guy W. Eastman, Dr. Gilbert M. Lewis and Mr. Edward W. Washburn. Mr. Roy D. Mailey has been promoted to the position of research associate. Seven candidates for the degree of doctor of philosophy are now pursuing work in the laboratory.

Digest of U. S. Patents.

Compiled by Byrnes & Townsend,
Patent Lawyers,
National Union Building, Washington, D. C.

Treatment of Carbon and Production of Graphite.

No. 354,310, Dec. 14, 1886, Thomas A. Edison, Menlo Park, N. J.

Produces rods of pure carbon by electrically decomposing a hydrocarbon liquid or gas. An arc is sprung between the ends of horizontal electrodes submerged in a hydrocarbon oil, and the electrodes are gradually separated. One electrode consists of a piece of carbon; the other of a platinum or carbon rod or wire of relatively small area. The oil is decomposed by the heat of the arc, and carbon deposits on the end of the smaller electrode and gradually builds out into a rod as it is withdrawn.

No. 542,982, July 23, 1895, E. G. Acheson, Monongahela City, Pa.

Purifies carbon by electrically volatilizing the contained impurities, as silica, lime and sulphur. For example, granulates coke and feeds it downward between the opposed ends of horizontal electrodes passing through the side walls of an electric furnace. The time of heating is controlled by the rate of withdrawal of the product. The purified carbon is used for arc-light rods, electric contacts, etc.

No. 568,323, Sept. 29, 1896, E. G. Acheson, Monongahela City, Pa.

Converts carbonaceous materials, such as mineral coal, coke, charcoal, gas-carbon and carbides into practically pure graphite, by employing a material containing a considerable proportion of mineral matter, or mixing with it an oxide, or oxides, such as silica, clay, alumina, magnesia, lime or iron oxide, and heating the mixture in an electric furnace. The mixture may consist of powdered coke, 50 per cent; sand, salt and sawdust, and the heating may be effected by placing it in a furnace around a granular carbon core, extending between terminal rods of carbon in the end walls. Upon passing an electric current through the resistance core, the carbon in the charge, or portions thereof, react on the sand, and there is thus initially formed around the core a layer of amorphous and crystalline silicon carbide, which gradually increases. The current is then continued until the temperature rises to a point sufficient to decompose the silicon carbide and volatilize the silicon, the carbon remaining in graphitic form. The graphite thus formed around the core then carries the electric current and furnishes heat to decompose more of the surrounding carbide, and graphite is thus produced and deposited on the core until it nearly reaches the walls of the furnace. For a furnace having a core of granular coke 7 feet long and 4 inches wide, the initial current may be 50 amps. at 650 volts, increasing, as the graphite is formed and enlarges the core to 1,000 amps. at 100 volts. When the carbonaceous material to be treated contains an unusual amount of ash, the oxide added should be correspondingly diminished, and if it contains more than enough impurities the charge may consist of a mixture of the impure carbon with hard coal or coke. The carbonaceous material should contain little volatile matter, to prevent loss of electrical energy by distillation. The process may be carried out in two separate furnaces or stages, silicon carbide, in its crystalline or amorphous form, being made in one operation, and subsequently heated to the temperature necessary to effect dissociation.

No. 598,549, Feb. 8, 1898, Herbert H. Wing, Buffalo, N. Y.

States that amorphous carbon may be converted into graphite by prolonged heating at a high temperature, without the intermediate production and decomposition of carbides. For example, heats coarsely powdered coke in an electric furnace by employing it as a resistor, the temperature being just below the volatilization point of carbon. Illustrates a furnace having a vertical air-tight chamber with carbon lining, a depending carbon electrode and horizontal screw conveyors, with double-valved hoppers leading through opposite sides. The chamber opens below into a water-jacketed metal receiver, in which the graphite and carbon are cooled without exposure to air. To best utilize the electrical energy, only a portion of the carbon is graphitized, the remainder being dissolved out by an active oxidizing agent; *e. g.*, a mixture of a chlorate and nitric or sulphuric acid, or both.

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